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JANUARY, 1949

BULLETIN OF THE ISTITUTION OF MINING AND METALLURGY

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No. 506 **JANUARY, 1949**

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NOTICE OF GENERAL MEETING

The Fourth Ordinary General Meeting of the Fifty-Eighth Session of the **Institution of Mining and Metallurgy** will be held, by kind permission, in the Apartments of the Geological Society, Burlington House, Piccadilly, London, W. 1, on **Thursday, 20th January, 1949, at 5 p.m.**

The paper entitled **Mining and milling antimony ore at Consolidated Murchison Goldfields, Transvaal,** by Mr. Ralph Symons, *Member,* which is published in this issue of the *Bulletin,* will be submitted for discussion, and will be introduced by Mr. J. B. Dennison, *Member.*

Light refreshments will be provided at 4.30 p.m. for members and visitors attending the Meeting.

The Council invite written contributions to the discussion of papers from members who may be unable to be present at the Meetings of the Institution. The Council reserve the right to edit and condense such contributions.

INSTITUTION NOTES

Adoption of the new By-Laws

A Special General Meeting of Members and Associates of the Institution, held at Burlington House on 16th December, 1948, unanimously resolved that the By-Laws of the Institution as revised and submitted at the Meeting be approved and presented to His Majesty's Privy Council for confirmation and in substitution of the existing By-Laws of the Institution. The new By-Laws have since been submitted to the Privy Council.

Capper Pass Awards

In 1947, the Directors of Messrs. Capper Pass and Son, Ltd., Bristol, sharing the regret which had been expressed in many quarters at the dearth of papers on processes and plant used in extraction metallurgy in the *Transactions of the Institution of Mining and Metallurgy*, and of papers on processes and plant used in the fabrication of non-ferrous metals in the *Journal of the Institute of Metals*, offered to these Institutions the sum of £200 per annum for a period of seven years to be applied as follows:

(*a*) £100 per annum to be available for one or more Awards to the authors of papers on some aspect of non-ferrous extraction metallurgy;

(*b*) £100 per annum to be available for one or more Awards to the authors of papers relating to some process or plant used in the extraction or fabrication of non-ferrous metals,

contributed by persons engaged full time in industry or practice.

The Councils of the Institution of Mining and Metallurgy and of the Institute of Metals gratefully accepted this offer, and appointed a joint Adjudicating Committee. This Committee has power to make the awards on behalf of the two societies and may, at its discretion, make no award or awards of less than the money available if, in its opinion, the quality of papers submitted in any year fails to reach a suitable standard. Any sums not awarded will be carried forward to future years.

The two Councils hope that the generous offer made by Messrs. Capper Pass and Son, Ltd., will stimulate the writing of many papers of the types for which the Awards are to be made. Papers on extraction metallurgy should preferably be submitted to the Institution of Mining and Metallurgy, while those on processes and plant used in the fabrication of non-ferrous metals should preferably be offered to the Institute of Metals. Both societies are prepared to accept papers of suitable quality from non-members.

Authors should note that applications should not be addressed to the Adjudicating Committee requesting that their papers should be considered for an Award. All papers published by both societies will be examined by the Committee annually, and notices of the Awards will be published in the journals of the two societies and in the Press. The Committee will shortly consider all papers published by the two societies during 1948.

Joint Committee on Metallurgical Education: Summary of Progress Report for 1946

The following summary of the Report on Progress on the work of the Joint Committee for the period 1st January, 1947, to 30th June, 1948, is published for the information of members.

At 30th June, 1948, there were 21 members of the Committee, representing the Industry, The Iron and Steel Institute, The Institution of Mining and Metallurgy, The Institute of British Foundrymen, The Institute of Metals, The Institution of Metallurgists, the Universities, The City and Guilds of London Institute, The Association of Technical Institutes, and The Association of Principals of Technical Institutes. Professor Leslie Aitchison was appointed Chairman of the Committee in June, 1947, on the resignation from the Chair of Dr. C. H. Desch.

Nine meetings of the Committee were held during the period, including a special meeting at which members met and exchanged views with Dr. Robert F. Mehl of the Carnegie Institute of Technology, Pittsburgh, Chairman of the Advisory Committee on Metallurgical Education of the American Society for Metals.

The Committee has given information to parents and students and has made arrangements for the loan of metallurgical films. A second revised edition of 'Metallurgy—A Scientific Career in Industry' was published in May, 1948, and over 6,000 copies have been circulated.

The work of the Committee includes :

1. Consideration of the general requirements for providing education for metallurgists of all grades.

2. Assisting in every way possible in the provision of the means by which these requirements can be secured at the different levels.

3. Making known throughout the Secondary and Public Schools the opportunity offered by metallurgy as a career.

4. Providing parents, guardians, careers masters and others with full details of the educational facilities available to fit boys for various careers in metallurgy.

5. Keeping continuous touch with industry in order to ascertain in what respects the young metallurgist is considered to fall short of the appropriate standard when he enters the metallurgical industry—

(*a*) with a University Degree,

(*b*) with a National Certificate in Metallurgy or qualification of a similar standard,

(*c*) with other qualifications from a Technical School or College.

Ascertaining how far his falling short of the highest standard can be laid at the door of metallurgical education and taking such remedial steps as may be possible.

6. Taking appropriate steps for carrying its policy into effect, including :

(*a*) Discussions and meetings with the interested people in industry, Schools, Technical Colleges, Universities and Government Departments, in order that views may be exchanged.

(*b*) The preparation of memoranda addressed to the various bodies which may be concerned and the publication of the Committee's views and suggestions.

7. The consideration of special matters of an important nature, such as entrance qualifications to University Schools of Metallurgy. A pamphlet giving the Committee's 'Recommendations on Qualifications for Entrance to University Schools of Metallurgy' was published in April, 1948, and widely circulated to the various interested bodies and the recommendations have been published in the Press and in the journals of the participating Institutes [see *Bull.* 500, July, 1948].

National Certificates in Metallurgy

The following is reproduced for the information of members :—

Report by the Joint Committee on the Progress of the Scheme for the Award of National Certificates in Metallurgy for the year 1947–48

1. During the year under review The Institution of Metallurgists has joined The Iron and Steel Institute, The Institution of Mining and Metallurgy and The Institute of Metals in co-operating with the Ministry of Education in operating the scheme for National Certificates in Metallurgy in England and Wales.

2. Schemes have been approved by the Joint Committee and have been in operation during 1947–48 at the following Technical Colleges :

(*a*) Senior Courses leading to an Ordinary National Certificate in Metallurgy—

Battersea Polytechnic, London ; Birmingham Central Technical College ; Chesterfield Technical College ; Coventry Technical College ; Cumberland Technical College, Workington ; Derby Technical College ; Dudley & Staffordshire Technical College (Scheme is supplementary to that of County Technical College, Wednesbury) ; Enfield Technical College ; Ilkeston Technical Evening Institute ; Merchant Venturers' Technical College, Bristol ; Newport Technical College ; Rotherham

College of Technology; Scunthorpe Technical College; Smethwick, The Chance Technical College; Swansea Technical College; Wednesbury, County Technical College; Wolverhampton and Staffordshire Technical College.

(*b*) Advanced Courses leading to a Higher National Certificate in Metallurgy:

Battersea Polytechnic, London; Birmingham Central Technical College; Chesterfield Technical College; Middlesbrough, Constantine Technical College; Rugby College of Technology; Rutherford College of Technology; Wednesbury, County Technical College; Wolverhampton and Staffordshire Technical College.

3. Final examinations have been held in 1948 at the following technical colleges for students who satisfied the conditions laid down in Ministry of Education Rules 111 under which the Scheme is operated:

Ordinary Certificate: Battersea Polytechnic, London; Birmingham Central Technical College; Chesterfield Technical College; Coventry Technical College; Cumberland Technical College, Workington; Derby Technical College; Enfield Technical College; Ilkeston Technical Evening Institute; Newport Technical College; Rotherham College of Technology; Scunthorpe Technical College; Smethwick, The Chance Technical College; Swansea Technical College; Wednesbury, County Technical College; Wolverhampton and Staffordshire Technical College.

Higher Certificate: Battersea Polytechnic, London; Birmingham Central Technical College; Chesterfield Technical College; Middlesbrough, Constantine Technical College; Rugby College of Technology; Rutherford College of Technology; Wednesbury, County Technical College; Wolverhampton and Staffordshire Technical College.

4. (*a*) 123 candidates entered and 74 have qualified for the award of an Ordinary National Certificate in Metallurgy; 38 candidates entered and 22 have qualified for the award of a Higher National Certificate in Metallurgy.

(*b*) The Committee is pleased to report that Distinctions have been awarded to 13 candidates who have shown an exceptional grasp of their subjects, indicating a high degree of training and knowledge in the particular subject in which the Distinction has been gained.

5. The Joint Committee takes this opportunity of thanking the Assessors for their valuable assistance in operating the Scheme.

6. (*a*) Prizes, taken in books and distributed by the Technical Colleges concerned, were awarded to 19 successful candidates in the final examinations in 1947, from the Prize Fund established by The Iron and Steel Institute, The Institution of Mining and Metallurgy and The Institute of Metals for this purpose.

(*b*) Prizes will similarly be awarded to 17 successful candidates who have shown particular merit in the final examinations held in 1948.

7. The Joint Committee is pleased to note the increase in the number of candidates who entered for the final examinations for the Ordinary and Higher National Certificate in Metallurgy this year, and that there is a still greater increase in the number of students entering the first and second years of the courses.

Signed on behalf of the Joint Committee for National Certificates in Metallurgy.

H. S. TASKER,
Chairman.

December General Meeting

The Third Ordinary General Meeting of the Session was held on Thursday, 16th December, 1948, at the Geological Society of London, and was attended by about 85 members and visitors. Mr. E. G. Lawford introduced the paper entitled 'Notes on the treatment of pyrites cinders at the plant of the Pyrites Co., Inc., Wilmington, Delaware' on behalf of his co-authors, Messrs. R. C. Trumbull and W. Hardiek. A full discussion of the paper followed, and a report of the Meeting will be published in the February issue of the *Bulletin*.

The paper by Dr. G. A. Schnellmann entitled 'A note on "steel" galena' was not discussed at the Meeting owing to lack of time, and written comments are invited for publication.

Fifty-Eighth Session, 1948-49: Dates of Subsequent Meetings

The following are the dates fixed for General Meetings of the Institution during the remainder of the Session 1948–49:

17th February, 1949.
17th March, 1949.
21st April, 1949.
19th May, 1949.

(These dates are the third Thursday of the month.)

Members from Abroad

The Council are always anxious to meet members who come to England after a long absence abroad, and ask such members to make themselves known to the Secretary when attending General Meetings of the Institution at Burlington House.

Institution Awards

'The Consolidated Gold Fields of South Africa, Limited', Gold Medal and Premium of Forty Guineas are awarded jointly or separately by the Council of the Institution for the paper or papers of highest merit contributed to the *Transactions* during each Session, or for researches on the occurrence, mining, or treatment of minerals. The Council shall be satisfied that the papers or researches are of sufficient merit to justify the award.

Two prizes of Ten Guineas each are offered annually for papers contributed to the *Transactions* by Students of the Institution, provided that the papers are, in the opinion of the Council, of sufficient merit to justify an award.

Papers for the consideration of the Publications Committee should be sent to the Secretary, if possible in duplicate, and should be prefaced by a summary of contents. It is understood that all papers submitted are original communications unless distinctly stated to be otherwise, in which event exact reference should be made to any previous publication. Figures illustrating papers should be drawn in ink, suitable for direct reproduction in a reduced size, and lettering on drawings should be in ordinary pencil. If there are photographic illustrations, prints on glossy paper should be sent; it is not necessary to send negatives.

Transfers and Elections

The following were transferred (subject to confirmation in accordance with the conditions of the By-Laws) on 9th December, 1948:

To Membership—

Trevor Mort (*Roodepoort, Transvaal*).
Cyril Brock Pengilly (*Penhalonga, Southern Rhodesia*).
Gordon Murray Stockley (*Dodoma, Tanganyika*).
Henry James Reginald Way (*Mbabane, Swaziland*).

To Associateship—

Thomas Francis Victor Cooper (*Bukuru, Northern Nigeria*).
Valentine William Alan Duke (*Springs, Transvaal*).
Gordon Leslie Hatherly (*Venterspost, Transvaal*).
Hugh Hilton McGregor (*Rustenburg, Transvaal*).
Gordon Finimore Sherman (*Kakamega, Kenya*).
Christopher Percy Tremlett (*Newquay, Cornwall*).

The following were elected (subject to confirmation in accordance with the conditions of the By-Laws) on 9th December, 1948:

To Membership—

Spencer Richard Fleischer (*Johannesburg, Transvaal*).

To Associateship—

Pauli Kristen Aamo (*Vest Agder, Norway*).
Robert Bowie (*Marikuppam, India*).
Francis Eugene Fitzgibbon (*Kakamega, Kenya*).
Harvey Thomas Ford (*Bindura, Southern Rhodesia*).
William Thomas Hocking (*Oorgaum, Mysore State, India*).
Bernard Samuels (*Gatooma, Southern Rhodesia*).
Alan Guy Valentine (*Kakamega, Kenya*).
Arnold Percival Warwick (*Hexham, Northumberland*).

To STUDENTSHIP—

Douglas Fairburn Ainge (*Dunedin, New Zealand*).

Allan Francis Charles Barnes (*Sutton, Surrey*).

Roger Ernest Barnes (*London*).

Alan Sydney Bragg (*London*).

Guy Bridgstock (*London*).

Vivian Trevor Brokenshire (*Camborne, Cornwall*).

John Ashley Catterall (*Romford, Essex*).

Albert John Michael Clesham (*Camborne, Cornwall*).

Alexander Barrett Cowie (*Dunedin, New Zealand*).

George Martin Du Boulay (*London*).

Alan Dunning (*London*).

John Vivian Eplett (*Camborne, Cornwall*).

Lewis Trevor Evans (*Camborne, Cornwall*).

Ralph Patrick Grosscurth (*Camborne, Cornwall*).

Vernon Walter Hall (*Redruth, Cornwall*).

Reginald John Harbord (*Kenley, Surrey*).

Roger Geoffrey Rainbird Harrisson (*Colchester, Essex*).

Laurence Alexander Hill (*London*).

John Kelson Holgate (*London*).

George Scott Inns (*Ruislip, Middlesex*).

Sydney William Jarvis (*Dunedin, New Zealand*).

Richard George Lane (*London*).

John Louis Leroy (*London*).

Paul Bryan Locke (*Harrow, Middlesex*).

James Neil Luscombe-Monro (*Illogan, Cornwall*).

Derek Maishman (*Chesterfield, Derbyshire*).

Arthur Yelland Moon (*Par, Cornwall*).

Howard Alan Collins Moon (*Barakin Ladi, Northern Nigeria*).

Derek George Warner Norris (*London*).

Anthony Giles Gale Oliver (*London*).

Henry Desmond Osborne (*Redruth, Cornwall*).

Peter Gerard Parker (*London*).

Robin Charles Penfold (*London*).

Nicholas Robb (*Axminster, Devon*).

Alan Jeffery Robinson (*Bembridge, Isle of Wight*).

James Fordham Sadler (*Woldingham, Surrey*).

John Henry Saunders (*Wallingford, Berks.*).

Terence John Skelton (*St. Austell, Cornwall*).

Roy Slater (*London*).

David Valentine Storrs (*Rickmansworth, Hertfordshire*).

Norman Hubert Townend (*London*).

Leonard Eric Webb (*London*).

Richard Alan Christopher Williams (*Redruth, Cornwall*).

Brian Richard Woolfe (*Camborne, Cornwall*).

Candidates for Admission

The Council welcome communications to assist them in deciding whether the qualifications of candidates for admission into the Institution fulfil the requirements of the By-Laws. The application forms of candidates for Membership or Associateship will be open for inspection at the office of the Institution for a period of at least two months from the date of the **Bulletin** ***in which their applications are announced.***

The following have applied for transfer since 9th December, 1948 :

To MEMBERSHIP—

Herbert Cecil Herbert (*Mosaboni, India*).

Robert Pitman Hooper (*Broken Hill, N.S.W., Australia*).

Gerald Augustine Patrick Moorhead (*Georgetown, British Guiana*).

Frederick Charles Willoughby (*Filabusi, Southern Rhodesia*).

To ASSOCIATESHIP—

Wilfrid Henry John Luck (*Que Que, Southern Rhodesia*).

Arthur Theodore Max Mehliss (*Bulawayo, Southern Rhodesia*).

William John Palk (*Hangha, Sierra Leone*).

Dennis Frederick Reeves (*Bukuru, Northern Nigeria*).

The following have applied for election since 9th December, 1948 :

To ASSOCIATESHIP—

Walter Charles Hellyer (*Harrow, Middlesex*).

Joseph Pinder (*Geita, Tanganyika*).

Francis Stark (*Oorgaum, S. India*).

To STUDENTSHIP—

Jeffrey Kenyon (*London*).

James Harold Newman (*Birmingham, Warwickshire*).

Vincent Hugh Robert Oliver (*Mufulira, Northern Rhodesia*).

Stanley James Ramage (*Uxbridge, Middlesex*).

Everard James Ross (*Birmingham, Warwickshire*).

Dimbeswar Sarma (*Leeds, Yorkshire*).

News of Members

Members, Associates and Students are invited to supply the Secretary with personal news for publication under this heading.

Mr. A. T. Ahlston, *Member*, has returned to Sierra Leone.

Mr. W. R. Barnes, *Associate*, has joined the mill research staff of the Zinc Corporation, Ltd., and is now at Broken Hill, N.S.W.

Mr. J. G. Berry, *Associate*, is returning to England on leave from India.

Mr. W. T. M. Browne, *Associate*, has returned to England from Tanganyika.

Mr. A. Carstairs, *Associate*, has returned to England from Brazil.

Mr. K. E. Daniel, *Student*, expects to leave Burma for England in February.

Mr. E. F. Elkan, *Member*, has left Malaya for furlough in France and England.

Mr. H. G. End, *Associate*, has returned to Switzerland on leave from Malaya.

Mr. D. F. Foster, *Associate*, now holds a position with Messrs. Macdonald Adams & Co., Johannesburg.

Mr. S. F. Gandar, *Student*, is returning to England from Burma.

Mr. F. R. H. Green, *Associate*, has returned to Sierra Leone.

Mr. G. Harvey, *Associate*, is returning to England on leave from Saudi Arabia.

Mr. S. R. R. Hood, *Student*, has joined the staff of the Cerro de Pasco Copper Corporation, Peru.

Mr. R. C. A. Hooper, *Student*, has arrived in England from the Gold Coast.

Mr. H. D. M. Jager, *Associate*, is returning to England from India.

Mr. A. Leaver, *Associate*, has left England on his return to Curaçao, Netherlands West Indies.

Mr. E. Lee, *Associate*, has taken up an appointment in Egypt.

Mr. J. K. Macdonald, *Associate*, has joined the staff of the Cam and Motor Mine, Eiffel Flats, instead of the Shabanie mine, Southern Rhodesia.

Mr. C. B. Pengilly, *Associate*, has been transferred from the Rezende mine to the Tebekwe mine, Southern Rhodesia.

Mr. J. H. Polglase, *Associate*, was on 1st April, 1948, appointed assistant general manager of the Sungei Besi Mines, Ltd., Selangor, Malaya, and has since held the position of acting general manager.

Mr. R. P. Sheppard, *Student*, has returned to England on leave from India.

Mr. Diggory Stanton, *Member*, has returned to England from Sierra Leone.

Mr. W. L. Stewart, *Member*, is now at Seron, Almeria, Spain, having been appointed General Manager of Sociedad Minera Cabarga San Miguel and the Bacares Iron Ore Mines, Ltd.

Mr. K. J. St. George, *Student*, has returned to England from British Guiana, on leave.

Mr. Arnold Taylor, *Student*, is now at Brakpan in the employment of South Africa Land & Exploration, Ltd.

Mr. A. H. E. Taylor, *Associate*, has left England on a six months' visit to Hyderabad and the Kolar Gold Field.

Mr. K. L. G. Terrell, *Associate*, has returned to England from Colombia.

Mr. Ralph M. Thomas, *Associate*, has joined the staff of the Rio Tinto Co., Ltd., Huelva, Spain.

Mr. L. O. Tonkin, *Associate*, has left Tanganyika and has joined the staff of Falcon mines, Southern Rhodesia.

Mr. R. G. Wooding, *Associate*, is now chief of the efficiency study department, Crown Mines, Ltd.

Mr. F. O. Wright, *Associate*, has returned from Turkey and has joined the staff of the Directorate of Opencast Coal Production in Yorkshire.

Mr. L. Zutshi, *Associate*, has arrived in England from India.

Addresses Wanted

G. P. Anderson.	E. Dickson.
A. Armstrong.	L. E. Djingheuzian.
D. S. Broadhurst.	A. I. Scott.
J. B. Cocking.	A. Sloss.
L. Davies.	W. E. Storey.

BOOK REVIEWS

Tungsten: its history, geology, ore-dressing, metallurgy, chemistry, analysis, applications, and economics. 2nd ed. By K. C. LI and CHUNG YU WANG. New York: Reinhold Publishing Corporation, 1947. 430 p., illus., diagrs. $8.50.

This volume represents a vast amount of bibliographical research and should provide a useful compendium of original reference data on tungsten. It is therefore regrettable that certain printer's errors have escaped detection; bare numerals, for instance, frequently occur in the text and one is left to assume that these refer to temperatures in °C. These errata will be easily recognized by most. Some sections, however, contain unsound technical matter which casts a very definite shadow on the book's complete reliability. These unfortunate circumstances would have been avoided had the proofs been competently read.

It is clear that the authors' knowledge of the manufacture and fabrication of tungsten is shaky and that it has, in the main, been derived secondhand. This applies more particularly to the chapter on the metallurgy of tungsten which contains much of historical interest, details of patents (many of doubtful value) and illustrations of rather obsolescent reduction apparatus. The slavish collecting of theories on kindred subjects, derived from every possible source, frequently results in bewildering ambiguities such as is found on page 210. Here it is stated that 'offsetting' is caused by the growth of large grains in the glowing filament during use, resulting in the development of 'grain boundaries extending across the full diameter, in a plane at right angles to the long axis of the wire'. This perfect 20-word example of tautology could have been equally well expressed by three words—development of 'transverse grain boundaries.' This, it is emphasized, is a bad condition, and measures for inhibiting large grain growth are described. In the following paragraph, however, the authors point out the desirability of preventing the sagging of coiled filaments; the remedy in this case is to provide wire which possesses a structure built up of 'extremely large and long grains.' Clearly, these two statements are in conflict and the student would quite justifiably assume that if sagging were to be avoided then offsetting would be the inevitable consequence — an assumption which is happily quite untrue.

On page 207 it is stated that swaging operations are begun at 1500–1600° and that, as tungsten oxidizes readily at these temperatures, it is generally protected by a coating of graphite which is applied in the early stages, presumably at 1500–1600°C. This is frankly absurd since the danger of carbide formation at these temperatures would be very real. The presence of traces of carbide would render the metal unworkable.

The formation of oxide on the other hand would be quite harmless and, in point of fact, some manufacturers of the highest repute actually take elaborate measures to promote oxidation during swaging—this is accomplished by passing the incandescent rod through tubes filled with oxygen—as a means of removing or minimizing the overstressed outer layers of the rod.

It is customary to coat swaged rod with graphite during its passage through the last two or three swaging hammers, immediately prior to the first drawing operation, the latter being thus greatly facilitated. This, however, takes place at relatively low temperatures and results in the formation of a graphoid layer, unattended by any danger of chemical combination between the tungsten and the carbon.

On the same page (207) silica is classed with alumina and thoria as a non-volatile additive substance. It should be realized that, much below the sintering temperature of tungsten, silica is almost completely eliminated by volatilization, whilst alumina and thoria will remain behind as a non-metallic matrix existing in the intergranular spaces.

The first two sentences of the second paragraph on page 203 are mutually contradictory. In the first, it is stated that tungsten rod has never been satisfactorily made from carbon-reduced metal; in the second, we learn that the consumption of this grade of metal for hard-faced metals, welding rod, etc., amounts to several hundred tons annually.

The authors, having stated that additions are frequently made to tungsten, say on the same page (203) that 'any impurities which once get into the oxide will eventually contaminate the resulting powder. Hence, only the purest kind of tungsten ore is used for the manufacture of tungsten rods and wire.' Neither of these statements is wholly true. Regarding the first, it may be said that the normal additions to the silicated type of metal usually amount to approximately 1·25 per cent; these consist mainly of silica and alkali halides. The resulting metal in the form of rod or wire will, if properly prepared, analyse 99·97 per cent pure tungsten. The remaining ·03 per cent will consist of traces of alumina and silica. Clearly, since the sintering temperature is considerably over 3000°C, there are few impurities, intentional or otherwise, which can remain behind.

Regarding the exclusive use of the purest ores for making tungsten wires, this, as is well known by all who were engaged in the large-scale manufacture of tungsten during the last war, was a completely unattainable ideal. Whilst this was the state of affairs in the allied countries, conditions for the Germans were very much worse; nevertheless, all managed to make tungsten of the highest quality, and in all forms. The main reasons for preferring a high-grade ore are economic, not technical.

On page 348 we read that, during welding, tungsten contacts are coated with oil to prevent oxidation. It would be interesting to know what oil remains unvolatilized at the welding temperature (approx. 1100°C). In the writer's experience, oxidation is obviated by performing the welding operation in an atmosphere of hydrogen. Incidentally, on this same page we encounter one of the more colourful printer's errors: in referring to the use of tungsten contacts they are said to be employed in 'violet-rat machines.' In this section, which deals with tungsten contacts, no mention is made of the forms of ultra-fine grain tungsten (30,000–50,000 gr./sq. mm.) which give such remarkable performances when used as ignition contacts in aero engines operating in the presence of hydrocarbon vapours and at very great altitudes.

The chapter on analysis consists of a comprehensive write-up of some of the better-known schemes of analysis. The Cinchonine method is introduced as follows: '. . . requires 3 8-hour days. It is shorter and less accurate than most standard procedures. . . .' Why, then, give it priority of place?

In the case of the ordinary Brief method, it is stated that low results are to be expected from a solution containing alkaline salts. The writer suggests that the fusion be eliminated; the ore being decomposed with hydrochloric and nitric acids, followed by evaporation; the impure tungstic acid filtered off and dissolved in ammonia, filtered, the clear solution evaporated to dryness, the ammonium tungstate ignited to oxide. This is quicker and just as accurate as the so-called Brief method.

The volumetric method, using Jones's reductor, is of no use if, as is often the case, molybdenum is present.

On pages 291 and 292 the old fallacy about thorium dioxide remaining behind after volatilization in hydrochloric acid, is further perpetuated.

When the chloroform method is used the volatilization of the thorium is complete. This method is also quite useless for the estimation of alkali chlorides which begin to distil off at temperatures as low as 550°C. In most cases the amount of arsenic present is too small for the method described to be of any use. The familiar colorimetric method of Gutzeit is not even mentioned.

In the case of tin, very inaccurate results would be obtained if the method given were adhered to rigidly. For accurate results a current of carbon dioxide should be passed through the solution, not only during the reduction, but also during cooling.

The writer does not consider the method given for copper determinations to be a good one, whether the copper is present in quantity or as a mere trace, and prefer the technique which employs sodium diethyldithio carbamate.

On page 347 a list of trade names of some of the tungston carbide manufacturers in the U.S.A. is given. These include some which the reviewer thinks should not appear. One, most definitely, should not, since it is the trade name of a product being manufactured not many yards from where the writer sits. On this same page appears one of those stupid errors which seem to have escaped the authors, the proof readers, etc.: mention is made of 'the 76 min. high velocity rifle'. If this is intended to read 76 mm., then the question arises, at what calibre does a rifle become a gun ?

In the second paragraph of page 402 there is a colourful but historically incomplete description of the use of the tungsten carbide core high-velocity armour-piercing projectile, in which it is said that the famous British tanks virtually 'melted' when hit by these missiles. The text goes on to say that 'Two years later the U.S. Army perfected similar anti-tank projectiles which were used to stop Von Rundstedt's Belgian Bulge'. This is probably true, but it may be of more than passing interest that the British had accumulated vast numbers of these projectiles for use on D-Day and subsequently.

This book contains much that is good, but too much that is poor. It is hoped that when it reaches its next edition it will be all good.

T. F. SMEATON.

Prevention of iron and steel corrosion : processes and published specifications. By C. DINSDALE. London : The Louis Cassier Co., Ltd., 1948. 67 p. 5s.

There are many processes and published specifications relating to the prevention of the corrosion of iron and steel. This book is an attempt to compile a complete index of such methods and the standard specifications connected with them.

The matter is divided into three parts dealing respectively with methods of preventing corrosion (broken down into eleven groups), cleaning metal parts, and codes of practice. The two appendixes deal with authorities issuing specifications and with paint and paint component specifications.

Gold mining in South Africa. By C. W. BICCARD JEPPE. London : Todd Publishing Group Ltd., 1948. 160 p. 7s. 6d.

This book can be recommended as an introduction to the study of the Witwatersrand gold deposits, dealing as it does with the historical, statistical, technical, economic and future aspects of the industry. It is a marvel of compression and in the minimum number of words forms a very complete factual record of South African gold mining, all controversial matters being avoided or both sides simply stated in the first three-quarters of the book. Professor Jeppe makes the statement on page 122 that 'the mining operations on the Witwatersrand are fairly simple.' This is true on the basis that it is simple to do anything when one has a thorough knowledge of how to do it but, in the opinion of the reviewer, is not so on any other basis.

The last chapter of the book, dealing with the future of mining in South Africa, is naturally more an expression of opinion, but the author's beliefs regarding the position of gold in world economy in the near future and his remarks in connection with base metals in the Union are sound. His estimate that a mine laid out to mill over 2,000,000 tons per year and mining at vertical depths of from 8,500 to 13,000 ft. would necessitate a capital cost of the order of £6,000,000 seems, however, to be optimistic. It may be found possible to mine to only 11,000 ft. depth and a capital expenditure more nearly twice that estimated would appear to be indicated.

F. E. KEEP.

Electric winders. 2nd ed. A manual on the design, construction, application and operation of winding engines and mine hoists. By H. H. BROUGHTON. London: E. & F. N. Spon, Ltd., 1948. xx + 451 p., illus. 63s.

The first edition of this book was published in 1927 (not in 1928 as stated on the flyleaf) by Ernest Benn, Ltd., and it is significant that during the intervening twenty years no other comprehensive work on this subject has appeared in the English language. Indeed, another treatise on electric winders would be superfluous since Mr. Broughton has covered the design, construction, application and operation of electric winding machinery in a manner which can only be described as a notable achievement in the field of technological writing.

Nearly the whole of the text has been revised and seven new chapters have been added to cover developments which have taken place since the appearance of the first edition. The most important additions deal with recent advances in Koepe-pulley winders, skip winding of coal, intensive hoisting from shallow depths, winding from great depths, emergency braking and protective gear. A new chapter is also devoted to a discussion of the trend and practice of winding in Great Britain, South Africa, the U.S.A., and Germany, with special reference to the Reid and other reports which have been issued since the war.

The author has adhered closely to fundamental principles and there is little in the book that is empirical or hypothetical. The examples chosen to illustrate these principles are taken from actual winding installations and no fewer than 2,000 winders have been analysed for this purpose. A judicious balance is maintained between the electrical and mechanical sections and, although the book contains a very large number of equations, the text and the diagrams can be studied with advantage by the non-mathematical reader. A vast amount of information is provided for the specialist in the form of tables and graphs, and senior mining students at universities will find the book a useful guide in the drawing office as well as in the classroom.

The illustrations are in the form of diagrams specially prepared for the book and not reproductions of drawings reduced in scale to the point of illegibility, which mar so many treatises of this kind. A bibliography of international scope is appended to all but five of the 30 chapters and the index occupies 12 pages. Only a few pages are devoted to automatic winding in the present volume, and it is to be hoped that in future editions a whole chapter will be given to this subject, which is already claiming much attention.

W. DAVIES.

OBITUARY

Guy Carleton Jones died in South Africa on 3rd December, 1948, at the age of 60. He was born in Halifax, Nova Scotia, and was educated at King's Collegiate School and, from 1907, at the Department of Mining Engineering, McGill University, Montreal, where he graduated in 1912 with honours in science. In November, 1912, he was appointed draughtsman and surveyor to Messrs. J. S. Metcalf & Co., Construction Engineers, Montreal, and in March, 1914, took up an appointment as assistant assayer and general learner at the laboratories of the Consolidated Gold Fields of South Africa, Ltd., at Germiston, Transvaal, transferring after a month to Knights Deep mine, Witwatersrand, where he became assistant surveyor a year later. He was on active military service in German East Africa from 1916 to 1917, and then returned to Knights Deep mine as shift boss, joining the staff of Sub Nigel, Ltd., in April, 1918, and becoming manager in 1922.

In 1925 Mr. Carleton Jones joined the engineering staff of New Consolidated Goldfields, Ltd., in Johannesburg, and after acting successively as assistant consulting engineer and consulting engineer was, in 1934, appointed a joint manager of the company in South Africa. Nine years later he joined the board of the company, and served as resident director in South Africa until his resignation in 1947 owing to ill health. He retained his seat on the board, however, and was also a director of many other mining companies.

Mr. Carleton Jones was elected to Membership of the Institution in 1936 and was awarded the Gold Medal of the Institution for 1947 in recognition of his distinguished services to the gold mining industry of South Africa and in particular to the development of the West Rand.

Reginald Conduit Riley died on 16th August, 1948, at Lowestoft, Suffolk, at the age of 62. He was employed for six months at the Bishop Bridge Iron Works, Norwich, before entering the Camborne School of Mines in 1905, where he obtained a first class Diploma in 1908. He worked for a few months at the Dolcoath mine and in 1909 was appointed assistant mining engineer to the Central Provinces Prospecting Syndicate, Ltd. (now Central Provinces Manganese Ore Co., Ltd.), and was later in charge of some of the Syndicate's manganese mines in the Nagpur District of India. He was a trooper in the Nagpur Mounted Infantry Volunteers before the 1914–1918 war, and early in 1915 received his commission in the Indian Army Reserve Forces. He was first attached to the South Lancashire Regiment and then to the 81st Pioneers on the North West Frontier, and in August, 1916, served in Mesopotamia with the Royal Engineers, and was mentioned in despatches. He was released from the Army in 1919 with the rank of major, and rejoined the Central Provinces Manganese Ore Co., Ltd., in the capacity of mines manager of the Ramtek group of mines. From 1923 until his retirement in 1945 he was agent and general manager to the company in India.

Mr. Riley was elected a Student of the Institution in 1908 and was transferred to Associateship in 1913 and to Membership in 1929.

Arthur William Ross died at Handborough, Oxfordshire, on 14th November, 1948, at the age of 69. He was born in Australia and received his professional training at Sydney University from 1899 to 1902 and then worked for a year as acting manager of Rosedale Mining Co., New South Wales. In 1906 he was employed as metallurgist to Crœsus South Gold Mines, Ltd., Kalgoorlie, and a year later became manager. He joined Great Fingall Consolidated, Ltd., Day Dawn, Western Australia, in November, 1909, as plant foreman, but after seven months joined Vivien Gold Mining Co., Ltd., as metallurgist, shortly afterwards leaving to manage Mitchell's Creek Gold Mines, Ltd., Wellington, N.S.W.

In 1911 Mr. Ross took up the position of metallurgist to Bibiani, Ltd., Gold Coast, and from 1913 to 1914 was assistant manager of Naraguta (Nigeria) Tin Mines. He went to Burma in 1915 on his appointment as assistant manager of Hermyingyi Mining Co., Ltd., of which he was made superintendent in 1919. He left in 1921 to become manager of Rangoon Mining Co., a position which he held for six years before being appointed superintendent of Consolidated Tin Mines of Burma, Ltd., in 1928. From 1933 to 1939 Mr. Ross held the post of assistant general manager to Maroc, Ltd., Sierra Leone, and after a short period at the Dokuripe mine in Northern Territory, Gold Coast, he returned to England. Owing to ill-health he was unable to resume his profession.

Mr. Ross was elected to Associateship of the Institution in 1912 and to Membership in 1921.

The Council regret to announce the death on 16th December, 1948, of **Adam Alexander Boyd**, *Member*. An obituary notice will appear later.

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Subject to revision.] [*A Paper issued on 14th January,* 1949, *to be submitted for discussion at a Meeting of the Institution of Mining and Metallurgy, to be held in the Apartments of the Geological Society of London, Burlington House, Piccadilly, London, W.*1, *on Thursday,* **20th January, 1949,** *at* 5 *o'clock p.m.*

Mining and Milling Antimony Ore at Consolidated Murchison Goldfields, Transvaal*

By Ralph Symons†, A.R.S.M., B.SC. (Eng.) (*London*), *Member*

INTRODUCTION

The property of the Consolidated Murchison (Transvaal) Goldfields and Development Co., Ltd., comprising 2,745 gold and base metal claims, is situated in the Letaba district of the north-eastern Transvaal. The Monarch Cinnabar mine, which was described in a recent paper,‡ divides the company's area, but only the ground to the west of that mine is at present being actively exploited.

The claims are located along the 'Antimony Line' of the Murchison Range. This line, which follows a general direction of N 65°E., is so called because it marks a line of mineralization in the rocks of the Swaziland System which outcrop in this area: the length of the company's property along the Antimony Line is just under 13 miles.

The approximate latitude and longitude of the Weigel shaft, round which are grouped the mill, offices, and workshops, is 23°54'S. and 30°41'E. There is a railway siding at Gravelotte, on the line from Pietersburg to Lourenco Marques, where all supplies are offloaded and concentrate despatched. Gravelotte is 6½ miles distant from the mine on a gravel road. The head office of the company is in Johannesburg, which is 320 miles from the mine by the shortest route (Fig. 1).

The altitude of the ground on the company's claims ranges from about 1,500 to 2,400 ft. above sea level and the climate is sub-tropical, shade temperatures of over 100°F. being not uncommon in summer and light ground frosts being sometimes experienced during the winter months, which are generally dry. Rainfall is

*Paper received on 14th October, 1948.

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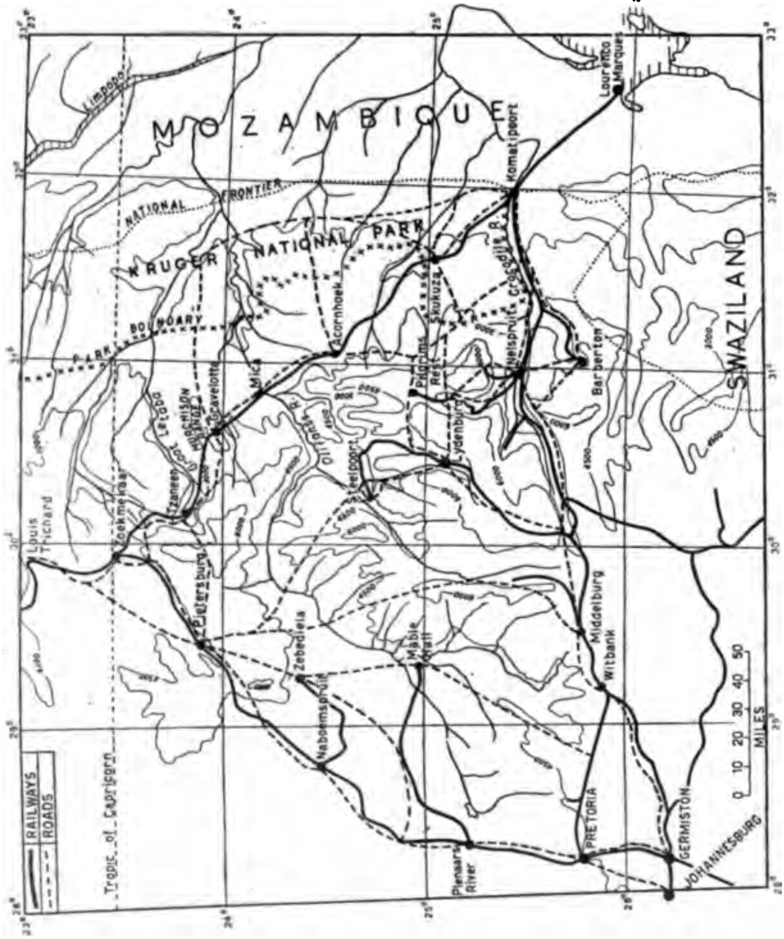
‡Wilson, N. W. Geology of the Monarch Cinnabar mine, Transvaal, South Africa. *Trans. Instn. Min. Metall.*, Vol. 54, 1944–45, pp. 169–200.


MOZAMBIQUE
NATIONAL FRONTIER
KRUGER NATIONAL PARK
PARK BOUNDARY
SWAZILAND
Limpopo
Lourenco Marques
Komatipoort
Skukuza
Crocodile R.
Nelspruit
Barberton
Pilgrims Rest
Lydenburg
Acornhoek
Mica
Gravelotte
Hutchinson
Letaba
Tzaneen
Soekmekaar
Louis Trichard
Pietersburg
Olifants R.
Steelpoort
Zebediela
Naboomspruit
Mable Hall
Middelburg
Witbank
Pretoria
Pienaars River
Germiston
Johannesburg
Tropic of Capricorn
RAILWAYS
ROADS
0 10 20 30 40 50
MILES

spasmodic and uncertain, a large portion of the average annual fall of 17 in. often occurring within a few hours. Thin bush vegetation covers the area, with sufficient grass to support the grazing of cattle on the numerous ranches which surround the mine. Local timber is useless for anything but fuel and the roughest forms of underground support. All timber used on the mine for mining purposes is purchased from a large timber company operating in the forests of the escarpment rainbelt at Duivelskloof, 60 miles west of the mine. Squared timber for constructional purposes is bought in Johannesburg.

TOPOGRAPHY

Because of their greater hardness, the quartzite beds of the Murchison schists form the major topographical feature of the district, which is otherwise flat or gently undulating. From end to end of the property stretches a remarkably straight ridge (the Antimony Range) which has been worn by erosion to serrated hills called, from west to east, Gravelotte hill, United Jack kop, Maid of Athens kop, and the Free State koppies (Fig. 2).

There are actually two parallel horizons of quartzitic rocks, the more northerly being termed the Chloritoid Bar and the southerly the Antimony Bar. The major line of antimony and gold mineralization lies parallel to and south of the Antimony Bar and all the mine workings outcrop on the southern slopes of the hills formed by it.

Water is a major problem in this field. Only one stream, the Malati, crosses the company's ground and it is dry except after rain. Shallow boreholes in selected spots sometimes yield small flows and at the deeper shafts towards the eastern end of the mine enough water is pumped to provide feed for rockdrills. About 70,000 gal./day is pumped from the Weigel shaft and is used for milling. Underground water is too hard for domestic purposes or for boilers, and the company obtains its main supply from its own pumping station on the Letaba River, 14·3 miles distant.

HISTORY

The Murchison Range has had a long, if chequered, career as a mining field. Since gold was discovered in 1870 an enormous amount of prospecting work has been done. All the mines opened up have been small and the yearly output of gold has never been large. The increase in the price of gold in the 'thirties greatly stimulated activity and in 1937 the total output was nearly 8,000 oz., a figure which has been greatly exceeded since.

Probably owing to uncertain markets the production of antimony has been erratic, although it rose sharply during the 1914–1918 war and the records show sales of 722 tons in 1916. Only since the cessation of hostilities after the recent war has it been possible to obtain reasonably secure terms for the sale of antimony ore and concentrates.

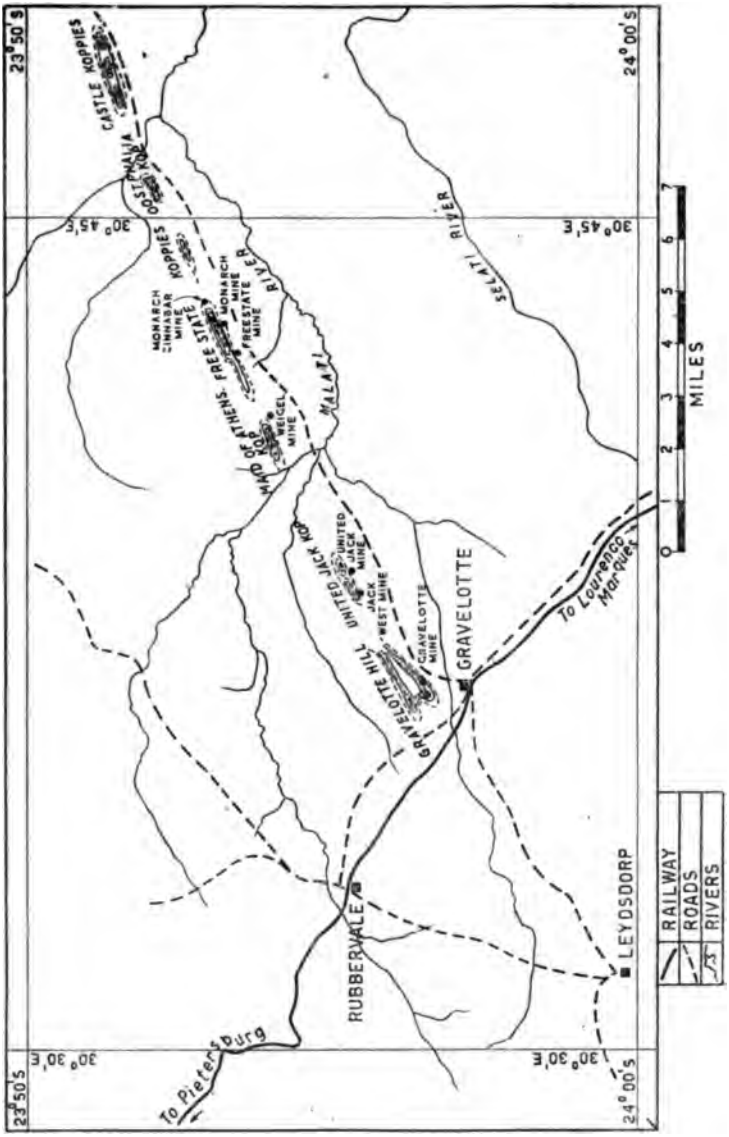


FIG. 2.—Plan showing antimony line.

Other minerals which have been mined in the district include ores of mercury and copper, emerald, beryl, corundum, and vermiculite.

The Consolidated Murchison Company was formed in 1934 and acquired the properties of several small, but locally celebrated, producers. These included the Weigel, Ruby, Monarch, United Jack, and Free State mines. Work was at first mostly confined to the Weigel section and it was here that the present mill and offices were erected. As time went on, however, the scope of operations enlarged and the tonnage mined increased until all the mines already mentioned, except the Ruby, were contributing to the production.

At first the company was interested in the production of gold only, and antimony was allowed to go to the waste dump. The firmer market and higher price for antimony gradually changed the policy of the company, until, at present, it is true to say that the aim in prospecting and development work is to bring to production as much antimonial ore as possible. Gold is still produced, but the value of the output is only a fraction of the value of the antimony output. Thus to-day the mine may be described as a base-metal producer, with gold as a by-product. It is the only mine producing antimony in the Union of South Africa.

GEOLOGY

The most recent standard work on the district is Memoir No. 36 of the Geological Survey of the Union of South Africa,* and the writer acknowledges this source as the basis for much of the information given under this heading.

The rocks underlying the company's claims all belong to the Swaziland System. A high degree of dynamic and contact metamorphism has given these rocks a schistose character and caused them to dip steeply towards the north at an angle usually between 80 and 90°, the strike being N 65°E. Most geological workers agree that compression has caused the rocks to assume a synclinal structure and that within this syncline siliceous beds—such as, quartzites and quartz schists—have, by their greater resistance to weathering, given rise to the hills of the Antimony Range. If this structure be accepted, it is probable that the Antimony Bar and the Chloritoid Bar, mentioned before, represent the same horizon. Between the two bars, and therefore stratigraphically younger, are found ancient lavas, which have been altered to schistose hornblende-quartz-clinozoisite rocks.

South of the Antimony Bar the rocks within the company's area are highly-altered sedimentary beds consisting of chlorite schists, carbonate schists, quartzite, quartz schists, and phyllites. The usual succession south from the Antimony Bar in most of the company's mines is a belt of chlorite schist averaging about 300 ft.

*van Eeden, O. R., Partridge, F. C., Kent, L. E., and Brandt, J. W. The mineral deposits of the Murchison Range east of Leydsdorp. 1939.

thick, followed by carbonate schists, which may be anything up to 600 ft. in thickness, and by 'talc' schist. The rocks change more or less gradually from one type to the next and there are no sharp contacts. The chlorite schist (called 'north schist' on the mine for simplicity and because of its position in relation to the lode) is dark-green, weathering to brown on the surface.

Practically all the Murchison rocks contain carbonate, so that the carbonate schist zone merely denotes rocks which are exceptionally rich in carbonates. N. W. Wilson* gives his opinion that these beds were originally impure magnesian limestones with arenaceous and argillaceous intercalations; this view is shared by the company's present geologists. The mine name for this zone, which contains the antimony-gold orebodies, is the 'dolomitic' or 'dolomitized schist' zone. Schistosity is usually weak in this zone and is sometimes entirely absent. The carbonates are ankeritic, with the result that a fresh exposure, greyish-white in colour, oxidizes to a light brown.

On the south side of the carbonated zone the rock grades, sometimes comparatively sharply, to a 'talc-schist'—a name given to denote its soft and soapy nature. It is, however, probably a chlorite schist, which has been subjected to more than usually intense dynamic metamorphism. This 'south schist' is paler in colour than the north schist and does not darken appreciably in weathering. The weakness of these rocks gives rise to difficulties in mining.

Banded ironstones are also found on the company's property and have been worked in two mines for their gold content—viz., at Gravelotte and at the Banded Ironstone shaft, where they lie north of the antimony lode. They are banded rocks consisting of alternate layers of grey chert and magnetic quartzite. They are considered by geologists to be of sedimentary origin.

The Murchison schist belt is surrounded by Old Granite, which is commonly a medium-grained grey biotite-granite. This granite has so far been found only in two places on the mine—viz., at the Weigel shaft, where a small aplite body occurs, and at Gravelotte mine, where there is a vein of sheared granite.

Dykes are fairly common on the property. Two types are distinguishable—viz., a pre-Karroo diabase, found at Gravelotte mine, and a dolerite of Karroo age, which is found at the Weigel and Jack shafts and in other places on the surface. These intrusions have not affected the mineralization, but they are a nuisance in mining, since they cut off and displace the reefs.

N. W. Wilson† has given an admirable account of the drag-folding and faulting at Monarch Cinnabar; similar conditions exist at Consolidated Murchison. Subsequent to the formation of the syncline, compressive forces acting upon the weaker rocks

*loc. cit.

†loc. cit.

against the resistance of the stronger quartzite bars caused pronounced folding, shearing, and strike-slip faulting. The fractures set up by these movements provided means of access for the mineralizing solutions which deposited the orebodies in the shear zones.

In mining the practical effect of this structure is that when a drag-fold is being approached in horizontal development, as is evidenced by an apparent change in the strike of the schistosity and the presence of numerous shear planes and joints, an improvement in the mineralization can be expected. Similarly, a flattening of dip usually coincides with an unusually rich patch of ore. The presence of strike faults, some of which occur in close proximity to the reef, usually on the south side, give rise to problems of support.

The lines of mineralization are parallel to the schistosity of the rocks. The reefs being worked at Consolidated Murchison are all in lines of mineralization which are in rocks south of, and therefore stratigraphically older than, the Antimony Bar on the southern limb of the syncline. There are three types of mineralization :

(1) Consisting of a zone of rocks fractured and mineralized with quartz and sulphides. Reefs of this type were worked for gold for many years at the Banded Ironstone mine and the Gravelotte mine. No antimony is present in these ores, the principal sulphides being pyrrhotite, pyrite, and arsenopyrite.

(2) The Antimony type, in which the principal sulphide is stibnite. Although the line of mineralization is continuous the payable reefs are lenticular in character, with poorly-defined walls. They consist of coarsely-crystalline ankeritic carbonate in a highly silicified zone. The stibnite may be finely disseminated, and this is usually the case where the rock has retained its schistosity—e.g., at the Jack West shaft—or it may occur massive when the rock has the appearance of being composed of solid sulphide—e.g., at the Monarch—or it may occur as sporadic blobs in the dolomitized schist—e.g., in the bottom levels of the Weigel shaft—or, finally, as coarsely-crystalline masses on the sides of cross fractures, as at the Free State mine. Occasionally, stibnite occurs in acicular form on fault planes. As a general rule the higher the grade of antimony in the reef the more gold it contains, but this is not universally true, the Gravelotte mine, for example, running low in gold values, even in places where antimony assays show over 40 per cent Sb. The antimony mineralization is not confined to a single line. At the Gravelotte, Jack West and United Jack mines there is a subsidiary line about 90 ft. north of the main strike, while at the Monarch there are two lines about 30 ft. apart, linked in one place by an oblique fault, and a third, pipe-like, body about 200 ft. to the south. All these occurrences are being actively exploited.

(3) Quartz reefs containing sulphides and gold. These reefs are in close proximity to or actually adjoining the antimony reef. At the Weigel shaft, the quartz reef is just north of the antimony reef.

but in several stopes has been mined together with it. At the United Jack and Free State it is also on the north side of the main antimony line, but at the Monarch it is about 40 ft. south of the antimony reef. The quartz reef is very patchy in size and value; it frequently contains visible coarse gold. The usual sulphides are pyrite and arsenopyrite with a little chalcopyrite and pyrrhotite.

A point mentioned by van Eeden* is that all the payable antimony orebodies opened up so far are in the vicinity of prominent hills. The explanation given by him that the mineralization was accompanied by a more intense silicification of the rocks seems reasonable.

Stibnite is found in surface exposures, but down to a depth of about 100 ft. is always accompanied by oxidized minerals, such as stibiconite. Since these minerals cause difficulties in the extraction process, assays for total antimony and antimony sulphide are necessary when determining the value of an orebody near the surface.

Although the orebodies are lenticular in character, they are remarkably persistent in depth. Those towards the eastern end of the mine generally pitch east and those at the western end pitch west, although it appears there may be one or two exceptions to this rule.

The following metallic minerals have been identified at Consolidated Murchison: gold, silver, stibnite, chalcostibite, berthierite, tetrahedrite, corynite, stibiconite, kermesite, valentinite, arsenopyrite, gersdorffite, löllingite, chalcopyrite, covellite, chalcocite, pyrite, pyrrhotite, rutile, and ilmenite. Pieces of native antimony have been recovered from the jigs when milling ore from the vicinity of dykes; no authentic instance is recorded of native antimony having been seen *in situ*. Cinnabar has been found on the north side of Monarch kop, in line with the outcrop on the claims of Monarch Cinnabar mine.

PROSPECTING

Prospecting work is in the charge of a geologist, who has the assistance of a team of diamond-drillers.

In earlier days all prospecting was devoted to the search for payable auriferous deposits, but since the most persistent values were often found on the Antimony Line many of the antimonial orebodies were opened up and mined for gold. It was generally assumed that orebodies would outcrop and prospecting work was confined to cross-strike trenches, followed by winzing at favourable spots. More recent experience has tended to show that surface indications are not an infallible guide to the presence of ore at depth. Several promising outcrops have failed to persist even to shallow depths, while two large orebodies discovered by exploration at depth pinch to very meagre outcrops at surface. The failure to recognize this tendency, together, probably, with the lack of

*loc. cit.

facilities for other methods of prospecting, was responsible for the great waste of money in unfruitful work in the early days.

At present prospecting is done mostly by diamond drilling. There are two surface drills, both Consolidated Pneumatic type CP15 with mechanical feed and driven by petrol engines. The two underground drills are a Sullivan 6 and a Sullivan 12 driven by compressed air. Surface and underground holes are drilled with B crowns $1\frac{21}{32}$ in. in diameter. Crowns are either handset or castset, the setting being done in Johannesburg. The castset bit gives cheaper drilling in this class of ground, the average cost over a number of months being 8s. 8d. per ft.

The surface drills are engaged in a systematic programme of drilling of the entire Antimony Line lying within the company's boundaries. This will take many years to complete, since none of the lenses so far exposed has a strike greater than 600 ft. Consequently, holes are placed about 200 ft. apart in the first instance and are inclined so as to intersect the reef at about 200 ft. vertical depth. The most likely areas—i.e., those in the vicinity of strong orebodies which have already been proved and those placed to explore at depth old gold workings now inaccessible—are being examined first.

In the first hole drilled in a new locality cores are taken for the complete hole, but to promote speed cores are taken in subsequent holes only at likely horizons, as shown by an examination of the core of the first hole. Drilling advance is fairly rapid, averaging 14 ft. 3 in. per shift. Care is necessary for satisfactory core recovery when the drill is approaching the orebody, owing to the sheared nature of the rocks, but in the orebody itself the high degree of silicification renders a good core recovery usually possible. A recovery of 91 per cent core has been maintained since drilling began. Recovery of water is usually very poor, and, in spite of precautions to minimize the loss, the supply of sufficient water to keep the drills going at full capacity is often difficult.

Owing to the sporadic distribution of mineralization no great reliance is placed upon assays from individual holes. If signs of antimony or gold, or both, are showing in a core, a second hole is drilled to intersect at 400 ft. depth and additional holes are directed to intersect the orebody at shorter intervals on strike.

Underground holes are drilled to test for parallel orebodies, for extensions both laterally and in depth of the antimony orebody from old gold workings, and for portions of the reef cut off by faults or dykes.

MINING

The mining department is under the control of a mine captain, assisted by five shift bosses. Between 11,000 and 12,000 tons of ore are broken in the mine monthly and delivered to the mill.

There are at present six separate mines being worked. They are, from west to east, Gravelotte, Jack West, United Jack, Weigel, Free State, and Monarch. Gravelotte is 6 miles from the central mill and Monarch 2½. All the mines are equipped with vertical shafts except Gravelotte, where only prospecting work from adits is in progress.

(1) *Hoisting*

Owing to the comparatively small tonnage developed on each level no underground ore bins are cut and ore is brought to surface in one-ton solid cars by means of cages.

The oldest and deepest mine is the Weigel, where the bottom level is 1,380 ft. below the collar. The double-compartment circular shaft, which is 13 ft. in diameter, goes as far as the 9th level (946 ft. below collar). The shaft is untimbered, the cages each having four rope guides, and is served by a twin-engine geared double-drum Robey steam hoist. Access to the mine below the 9th level is via a sub-incline shaft.

At the Monarch mine the shaft is rectangular (11 ft. by 7 ft.) and is at present being sunk from the 7th to the 8th level, which will be 825 ft. below the collar. The shaft is timbered and divided into two compartments, one of which is used as a ladderway and the other, which has timber guides, for hoisting. The hoist is a 130-h.p. single-drum Holman E.U. type, electric drive. At the Monarch, as at all the outside shafts, the cars are handtrammed to tipplers over timber ore bins, which have radial doors for discharging into tipper lorries to convey the ore to the central bin, near the Weigel shaft.

At the United Jack mine the shaft is down to No. 9 level, 922 ft. below collar. It is circular, 8 ft. in diameter, the cage running in rope guides against a balance weight of 4,300 lb. running in rail guides. The hoist is a 76-h.p. double-drum Holman with electric drive.

The shaft at the Jack West is 7 ft. by 7 ft., and is down to the 2nd level, 190 ft. below collar. Here the hoist is a 40-h.p. single-drum, electrically driven.

The Free State shaft is 7 ft. by 7 ft. and is also down to the 2nd level, 190 ft. below collar. A small Funkey hoist, coupled through a gear-box to a Mercury petrol engine, serves this shaft.

Neither of the last two shafts mentioned is equipped for hoisting men. Side-tipping bodies are lifted direct off their chassis by bridle chains and hooks and hoisted to surface, where they are lowered on to similar chassis run over the shaft on trap doors. Trap doors are also used on intermediate levels.

(2) *Method of Mining*

The method of mining is full shrinkage, flat back. This method has the great advantage of cheapness, with good control of the working face. No support is required in the stopes, even when the width is 45 ft., as the orebody itself is well silicified and normally strong. Light sprags only are required for the support of doubtful blocks in fractured zones. However, the walls are weak and occasionally give trouble when drawing a stope. One major subsidence due to this cause occurred at the Monarch, and it may be that in some sections of the mine a modification of the existing method will have to be made at deeper levels to overcome this danger.

The method has the disadvantage common to all shrinkage systems, that large tonnages of broken ore are locked up in the mine for considerable periods, but in a mine with no underground ore-bins to give capacity this is not altogether a drawback.

Most of the shafts are sunk in the foot-wall of the lode—i.e., on the south side—and the normal interval between levels is 100 ft. On the upper levels the method of development is to cross-cut to the reef horizon and then to run drives east and west on the lode for the full length of the orebody. Cross-cuts, or, latterly, diamond-drill holes, are put out from the drives to examine for values in the walls. By this means the boundaries of payability are defined. All development ends are 7 ft. high by 6 ft. wide.

In preparing a level for stoping the orebody is first sliped out to its full width and the broken ore removed. Then the roof is blasted down to give a height of 13 ft. above track level all over, the broken ore being left for the drillers to stand on. A timber gang then follows on behind the rockbreakers and, after the removal of the broken ore, sets are erected in the drive. The sets are made of 10-in. round timber and consist of a cap and two legs, the caps being blocked against the side walls : the spacing between sets varies according to the nature of the walls, but is never more than 5 ft. centres. Then 4-in. lagging is nailed round the outside of the sets and the space between the sets and the side walls filled in with broken ore to give stability to the timbers. Stope boxes spaced 15 ft. apart and facing along the drive are constructed of 9-in. by 1½-in. planks, with U irons for control boards. When the width of the orebody exceeds 20 ft. the track is split and a row of 4-ft. by 4-ft. flat lagging pigstyes is aligned down the centre of the drive to form the central support for the half sets erected on each side. Boxes are constructed along each track (Fig. 3).

When timbering is complete, the back is drilled and blasted lightly on to the timbers to form a cushion, after which the place is ready for normal stoping. Travelling ways are provided at each end of the stope and if the orebody is a long one additional cribbed travelling ways are taken up at intervals in the stope ; withdrawal of shrinkage does not disturb these travelling ways (Fig. 4, Plate I). A pillar 15 ft. thick is left beneath each level. These pillars are partly extracted later by reclamation work.

The weakness of the side walls in some sections of the mine causes excessive crushing of the drive timbers, necessitating heavy maintenance work. At deeper levels a modified method of development has been introduced to overcome this difficulty.

Foot-wall haulages are turned off from the main shaft cross-cut and run parallel with the orebody and 25 ft. horizontally away from it. Short cross-cuts are then run into the orebody at 50-ft.

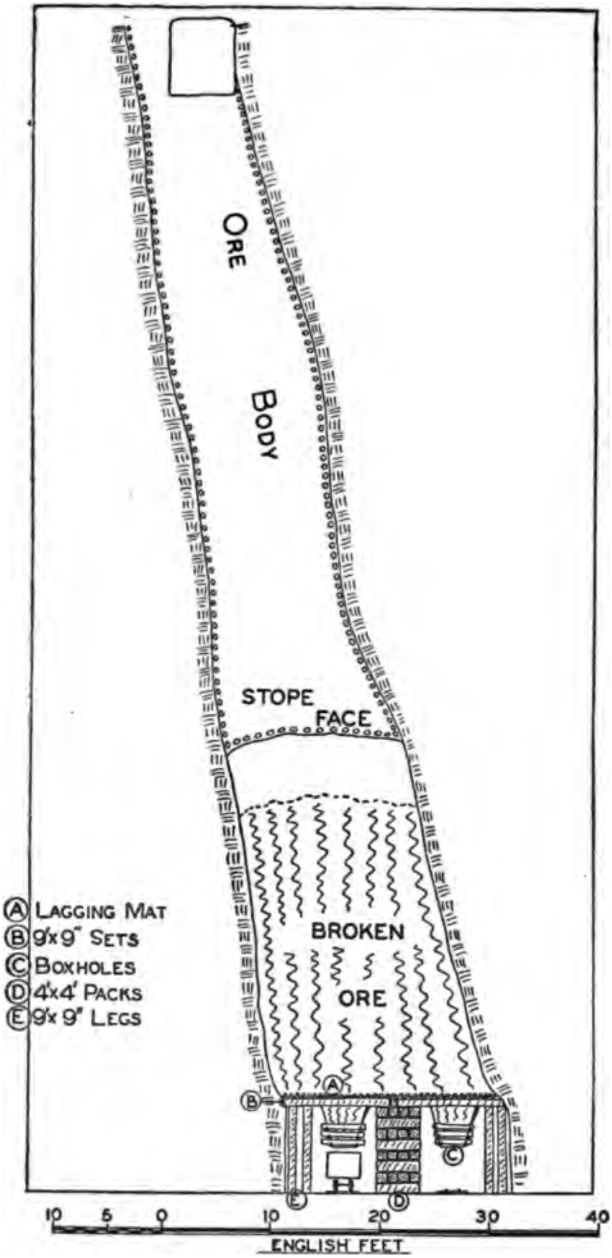


Fig. 3.—Timbered drives, double track stoping, transverse section.

intervals, from each of which two 55° boxholes, east and west, are raised up to the orebody to give a vertical height of 25 ft. The tops of the boxholes are then connected to form the stope drive, after which stoping proceeds as before.

The advantages of this method are that no timbering, apart from the boxholes, is required and that the haulage is secure and gives no trouble. Against this must be set the double amount of development required. In point of time, there is not much difference between the two methods, because the timbering of drives and the hand-lashing of broken ore into cars in the older method was a slow process. To reduce costs of development per ton mined a wider spacing between levels is being tried. It is also hoped that a much smaller tonnage will be left behind in pillars.

In a mine of this nature speed in vertical development is essential. Winzing has been found to be more rapid than raising; the mine is dry and the rockdrill water usually seeps away into the surrounding country or is absorbed by the broken rock, so that no pumping is required. In raising, the constant replacement of blasted sprags wastes much time. Consequently, in a raise-winze connection not more than 25 ft. is done by raising. Winzes are pushed down to new levels as soon as possible, so that very often they are ahead of the foot-wall haulages, and by the time the haulage reaches the connection, a considerable amount of stope driving has already been completed. Winzes are always kept vertical, even if they go outside the limits of the orebody, which is rarely the case. They are eventually used as travelling ways into the stopes, for transport of timber and drill steel, and as upcast airways.

In order to comply with the Mining Regulations, which require a second outlet for every mine, one winze is sunk in the foot-wall from each haulage to the one on the level beneath. These are equipped as permanent travelling ways which remain after stoping is completed.

(3) *Breaking Ground*

Wet drilling by compressed-air rockdrills is used throughout the mine, which is non-scheduled—i.e., it is not one to which the Phthisis Regulations are applicable. The type of machine in use is the Holman Silver Bullet jackhammer, although the Silver Dart, which is a lighter machine using less air, has recently been tried with success. In raises and boxholes the jackhammer is mounted on a jack-leg of the screw-feed type, otherwise it is hand-held.

Straight carbon $\frac{7}{8}$-in. hexagon hollow drill steel is used. Blunt steels are brought in every day to be sharpened at the central drill-sharpening shop. Two steels are used for drilling a stoping hole 3 ft. 6 in. deep, the starter being $1\frac{3}{8}$ in. and the second $1\frac{3}{16}$ in. in dia., both cruciform; shanks are $6\frac{1}{4}$ in. with rubber collar. Similar steel is used for development.

The explosive used is 50 per cent dynamite, although 40 per cent dynamite is used for blasting choked boxes and for popholes, with 60 per cent gelignite for primer cartridges. All cartridges are

8 in. long, and of 1 in. dia., except primers, which are 4 in. long by $\frac{7}{8}$ in. dia. Electric blasting is carried out in shaft-sinking operations with a shotfiring battery. Paper cartridges filled with sandy loam are used for stemming.

Blasting takes place once daily, at the end of the day shift at 2.45 p.m., except at the Monarch and the United Jack mines, where a double shift is run and the blast is at the end of the afternoon shift at 12 midnight.

Breaking efficiencies are as follows :

Stoping	
Footage drilled per machine shift	83
Fathoms broken per machine shift	2·16
Fathoms broken per case of explosives...............	5·22
Development	
Footage drilled per machine shift	66
Feet advanced per machine shift	3·25
Feet advanced per case of explosives	5·21

The tonnage broken per shift per underground native employed is 1·32.

(4) *Lashing and Tramming*

No lashing natives are employed in stopes. Pinch bar natives for dressing down the backs and sides and for spalling large rocks do all the handling of rock required. The stope boxes deliver direct into 1-ton cars and are hand-trammed to the shaft.

In development work no mechanical methods are used and there is a lashing gang consisting of two natives.

Tracks throughout the mine are of 18-in. gauge, constructed of 20-lb. rails dog-spiked to wooden sleepers spaced 2 ft. 6 in. apart. ‘ Extension rails ’—i.e., loose rails laid on their sides with the crown fitting into the web of the fixed rails—are used for conveying the car beyond the end of the permanent track when cleaning development ends. Steel plates, when available, are used as shovelling floors.

(5) *Ventilation*

Although Consolidated Murchison is a non-scheduled mine, the Mines Department has directed that many Regulations applicable to scheduled mines must be observed. For example, blasting may take place only once in every 24 hours and water-blasts must be fitted in development ends. Back stopes are not permitted and only approved types of rockdrills are allowed.

Except in the Monarch and United Jack mines, ventilation is natural. In spite of the depth of the Weigel shaft, the large area of old outcrop workings causes sufficient natural updraught for satisfactory ventilation. At the Monarch mine, the outcrop workings have mostly caved and a force fan is in operation on No. 7 level ; this is a Brown Sirocco-type fan electrically-driven by a 15-h.p. motor delivering 25,000 cu. ft./min. at $2\frac{1}{2}$-in. water gauge. The fan at the United Jack is similar but smaller, driven by a $7\frac{1}{2}$-h.p. motor and capable of delivering 15,000 cu. ft./min. at 2-in. water gauge.

The system of ventilation is the same at both shafts. The shafts down-cast from surface and are isolated by doors on all levels except the bottom, where the air is coursed along the foot-wall haulages and split to travel up through the travelling ways into the stopes. The permanent foot-wall travelling ways are isolated from the haulage by doors, but receive a small quantity of fresh air through a regulator on the bottom level. Thus, if a stoppage of the shaft at blasting time should occur and the travelling ways have to be used, they are free from fumes and gas.

In development ends compressed-air-driven Sirocco axial-flow fans or venturi blowers, forcing air through 14-in. galvanized iron or collapsible canvas tubes, are used. For the smaller ends, 7-in. galvanized tubing with venturi blowers are used.

(6) *Pumping*

The only shafts which have any appreciable influx of underground water are the Weigel, Banded Ironstone, and the old Free State. The last two are not at present being used, but are kept partly unwatered.

At the Weigel, there is a 10-h.p. centrifugal pump at the bottom of the sub-incline shaft, delivering to No. 11 level, whence another similar pump delivers to No. 9 level sumps at the bottom of the vertical shaft, where there is a 5-stage Sulzer centrifugal pump. From this level the water is pumped to the surface tanks in three stages, similar pumping stations existing on Nos. 6 and 3 levels. The water from this shaft is used in the reduction works. The rising main in the shaft is of 3-in. dia., supported on steel buntons. No sludge pumps are used ; most of the solids settle out very quickly in the haulage drains and the sumps are cleared of accumulated sludge at infrequent intervals. All underground water is strongly alkaline, pH 7·8.

At the Banded Ironstone mine pumping stations equipped with electrically-driven centrifugal pumps, similar to those at the Weigel are situated on Nos. 8, 6, and 3 levels. A three-throw Hunter pump delivers the water from the surface tanks to the Monarch mine through a 2-in. pipeline.

The old Free State shaft is kept unwatered down to No. 3 level, where there is a No. 3 Cameron pump delivering to surface. This water is used in the present Free State workings.

At the United Jack a ten-stage Sulzer centrifugal pump, driven by a 40-h.p. motor, pumps in one lift to surface, the water being used again for drilling purposes. Make-up water has to be supplied here by means of a 500-gal. tank transported by lorry. The Monarch is also a dry mine, the small quantity from the rock-drills being pumped to surface in two stages by Hunter three-throw pumps driven by electric motors.

(7) *Safety and Accident Prevention*

All underground employees are issued with hard hats, boots and carbide lamps. Stretchers and first-aid boxes containing bandages, splints, and dressings are kept underground on all mines at suitable places and are subject to periodical inspection by the medical officer. All boss boys are required to hold a Red Cross certificate. and instructional classes for the natives are held weekly by the medical officer. A bonus of 2d. per day is paid to boss boys who have obtained their certificates.

The percentage of shifts lost through mine accidents in 1947 was 0·48.

SURVEYING AND SAMPLING

The survey department is in the charge of a responsible surveyor, who has one surveyor and three samplers to assist him. Routine work follows usual Rand practice, except that the small size of the shafts and the fact that each mine has only one shaft calls for more than ordinary care in transferring azimuths underground. The frequency of very steep sights necessitates the use of instruments of a high order of accuracy.

Development samples are taken at 5-ft. intervals across the roofs of drives, continuously along both sides of cross-cuts, and across both sides of raises and winzes, also at 5-ft. intervals. In stoping the sampling interval is 20 ft., the sample being taken across the back from wall to wall. Samples are cut with a chisel and 4-lb. hammer; no attempt is made to cut a dimensioned groove, a ‘ chip ’ sample only being taken. The maximum length of sample cut permitted is 12 in.

Every working stope is sampled once a month, or oftener if circumstances demand. All samples are assayed for antimony and gold content. For gold determinations each sample is assayed individually but adjacent samples are often mixed for a composite determination of antimony at the discretion of the sampler.

The assay plan factor for 1947 was 82·75 per cent for antimony and 107·9 per cent for gold.

ASSAYING

The standard determinations required in connection with samples from the mine are for antimony and gold. For some mill samples arsenic is required and a complete analysis showing the content of antimony, gold, silver, arsenic, iron, nickel, copper, sulphur, and lead is periodically called for in connection with shipments of concentrates.

As some of the methods used on this mine are a departure from standard practice, the following details may be of interest:

Preparation of Samples for Assay.—Mine samples are crushed in a jaw crusher and quartered down to about 1 lb. by a Jones riffler; they are then pulverized by a disc crusher to approximately *minus* 60-mesh. Mill feed samples are crushed and quartered to 2 lb. and pulverized to pass 100-mesh; concentrates are given the same treatment.

Gold.—The stock flux used is: soda ash, 60 parts; litharge, 20 parts; borax, 7 parts; fluorspar, 7 parts; and meal, 2 parts.

Per assay, 100 g. of the stock flux is used, but where antimony is present over 3 per cent, nitre and additional litharge are added, the quantity required depending upon the amount of antimony and being regulated by experience.

Fusion is carried out in a coke-fired furnace of the Cornish type; this pattern seems to be the best suited to this type of ore, since it is often necessary to add fluxing material during fusion when occasion demands.

Antimony.—0·5 g. of sample is weighed out into a beaker and 10–15 c.c. of concentrated hydrochloric acid added. In the case of oxidized ores about 0·2 g. of potassium iodide is added to effect reduction of the higher oxides which are not easily soluble in acid. The mixture is allowed to simmer for a few minutes, 5 c.c. of 25 per cent tartaric acid solution added, the volume diluted to about 100 c.c. and, after heating to about 60°C., H_2S is passed to precipitate the antimony as sulphide. The precipitate is washed two or three times to remove iron, which interferes with the titration. There is nothing in the Murchison ore in sufficient quantity to warrant separation from the precipitate of antimony sulphide. The precipitate is then washed into a beaker and an equal volume of concentrated hydrochloric acid added. The sample is then boiled until the sulphide has gone into solution and all traces of H_2S removed. A little water is added and the solution is made alkaline to phenolphthalein by adding a 25 per cent solution of caustic soda, then re-acidified slightly with HCl and finally made alkaline with sodium bicarbonate solution. There must be present at this stage no other alkali than $NaHCO_3$, as other alkalis form hypoiodates which interfere with the result.

Oxidation of the Sb_2C_3l to Sb_2Cl_5 is now effected by standard iodine solution made up of 10·42 g. iodine to one litre distilled water. One c.c. of this solution on 0·5 g. of ore is equal to 1·0 per cent Sb. Starch is used as the indicator. The iodine solution is standardized daily.

In dealing with shipment concentrates the ore is dissolved in strong HCl, diluted, neutralized and titrated directly without previous precipitation with H_2S.

Arsenic.—0·5 g. of the sample is heated to fumes with 10 c.c. 50 per cent nitric acid and 5 c.c. 50 per cent sulphuric acid. The sample, after fuming, is cooled, 5 c.c. distilled water added, *plus* 1 g. each of hydrazine hydrochloride and potassium bromide. The addition of 10 c.c. of concentrated hydrochloric acid is made and adhering salts are washed down with 5 c.c. distilled water. A delivery tube and cork are inserted and the arsenic is distilled off as As_2Cl_3 into a solution of 12 g. sodium bicarbonate in 60 c.c. water. The distillation is taken to half volume and the distillate removed; a second distillation to half volume after adding another 5 c.c. concentrated HCl is then carried out into another solution of 8 g. sodium bicarbonate in 60 c.c. water. The two distillates are titrated separately with iodine solution, using starch as indicator, and the results added.

A blank determination is carried out at the same time using the same quantities and chemicals, and the result is deducted from the total readings.

Nickel.—10 g. of the concentrate is taken up with 50 c.c. of 50 per cent nitric acid, and the antimony oxides filtered off. To the solution a few grams of tartaric acid is added to hold iron in solution and the solution then neutralized with ammonia, any precipitate formed being filtered off. The filtrate is acidified with acetic acid, heated to 90°C., 15 c.c. of 0·1 per cent alcoholic solution of dimethylglyoxime added, and the solution just neutralized with ammonia. The red precipitate of nickel dimethylgoxime is filtered off, washed a few times with hot distilled water, ignited at red heat and weighed as NiO.

Lead.—5 g. of the concentrate is weighed out into a beaker and 50 c.c. of 50 per cent nitric acid added. The mixture is boiled to half the original volume. The oxides are allowed to settle and as much as possible of the liquid decanted off through filter paper; 50 c.c. of hot water is then added and the liquid again decanted off. To the residue is added 25 c.c. more of 50 per cent nitric acid, which is boiled and decanted off as before. The residue of oxides is now treated with 100 c.c. of hot concentrated sodium thiosulphate solution, filtered and washed; the residue is then discarded.

The first filtrate is cooled to room temperature, nearly neutralized with ammonia and H_2S passed for 10 minutes. The lead is precipitated along with any remaining antimony and arsenic. The sulphides are filtered and washed with hot water.

To the second filtrate 3 g. of sodium sulphide is added and the solution heated to boiling. The sulphides from the first filtrate are now washed into this solution and boiling continued until the liquid is clear.

The solution is now filtered and well washed and the paper and contents are transferred to a beaker containing 25 c.c. of concentrated nitric acid and 20 c.c. concentrated sulphuric acid. The liquid is heated to strong fumes and from time to time a few c.c. of a saturated solution of potassium chlorate in concentrated nitric acid is added from a dropping bottle to oxidize and burn off the paper. After cooling, the solution is made up to 50 c.c. with water and an equal volume of alcohol added. The lead sulphate is allowed to settle and the supernatant liquid decanted off through a fine filter. The lead sulphate is then carefully removed from the beaker with hot water and dried and ignited at low temperature in a porcelain crucible. After cooling 2 drops of nitric acid and 1 drop of sulphuric acid are added and the lead sulphate heated gently to dryness, then more strongly for a few minutes. It is finally weighed as $PbSO_4$.

SURFACE TRANSPORT

Incoming supplies and outgoing antimony concentrates are handled by railway road vehicles.

The company has a fleet of petrol lorries for transporting ore from the outside shafts to the Weigel bins ; timber, explosives, and stores from the central area to the outside shafts ; water to the United Jack, West Jack and Gravelotte mines, and personnel to and from work. The type of lorry standardized for transport of ore is the Ford 5-ton tipper model, although old lorries of different type are still in use. The total fleet consists of 12 vehicles. All drivers are natives who work on a task and bonus system. The cost of transporting ore (including depreciation) is 4·67d. per ton-mile.

ORE REDUCTION PROCESS

A flowsheet of the reduction plant is depicted in Fig. 5.

I.—Crushing and Sorting

Ore from the mine is collected in the main storage bin, whence it is transported by endless rope haulage to the plant, where it is screened over grizzly bars, and the oversize washed, sorted, and crushed in a primary crusher. The crushed ore joins the screened fines and is elevated to a vibrating screen in closed circuit with secondary crushers, the oversize passing through the crushers back to the screen and undersize being conveyed to the mill bins.

(1) *Transport of Ore*

Ore from the Weigel shaft arrives on the bank in 1-ton solid cars which enter a tippler delivering direct into the main storage bin. Ore from the outside shafts is transported in 5-ton Ford tipper lorries, which deliver their loads into the same bin. The bin, which is covered by a 14-in. by 14-in. grizzly, has a capacity of 400 tons, and it is constructed of reinforced concrete, about two-thirds of the total depth being in excavated ground, the upper third being enclosed in filled ground to form ramps for the approach of the lorries. There are six discharge chutes operated by radial doors underneath the bin which deliver into 1-ton solid cars. These, when filled, are attached by jockey to the rope of an endless-rope haulage, which pulls them up an incline to the level of the mill. A short tram by hand delivers them to a tippler discharging into the receiving surge bin. In the event of a shutdown of the endless-rope haulage the lorries may deliver the ore on to a floor next to the surge bin, whence it is shovelled by hand into the bin.

(2) *Primary Screening*

The crushing plant normally runs two 9-hr. shifts per day, the rate of feed being about 23 tons/hr. The ore is fed by a stationary chain feeder to an incline belt 24 in. wide, set at 17° elevation and driven at 150 f.p.m. by a $7\frac{1}{2}$-h.p. motor. This belt delivers the ore to a 3-ft. by 5-ft. grizzly screen set at 45°, with bars spaced $1\frac{1}{8}$ in. apart. Fines pass through to a 20-in. wide horizontal conveyor driven at

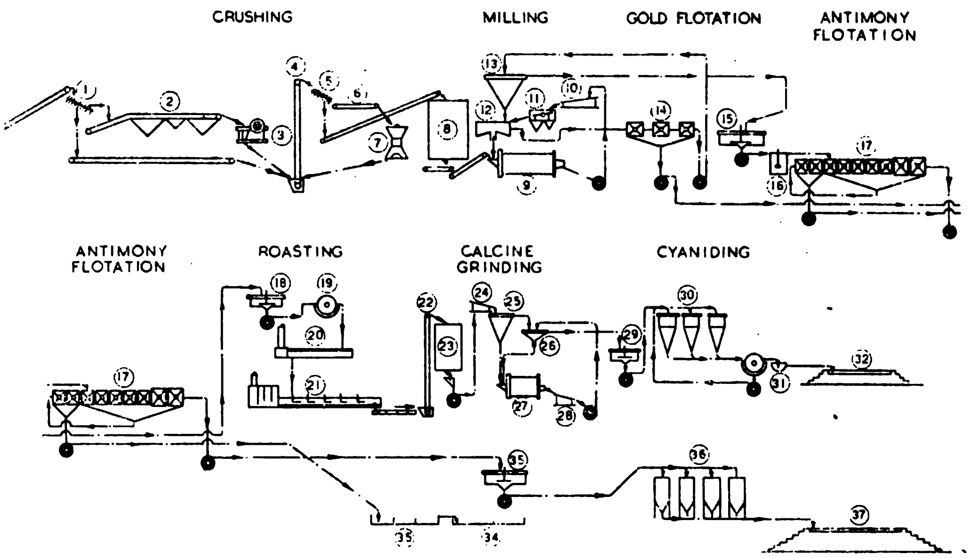


Fig. 5.—Flow-sheet of reduction plant.

1. Grizzly
2. Sorting belt for waste and cobbed ore
3. Stag jaw crusher
4. Bucket elevator
5. Vibrex screen
6. Secondary sorting belt
7. Kennedy crusher—secondary
8. Bin for crushed ore
9. Ball-mill
10. Corduroy tables
11. Denver mineral jig
12. Spiral classifier
13. Cone classifier
14. Denver flotation cells
15. Thickener
16. Conditioner
17. Denver flotation cells
18. Thickener
19. Drum filter
20. Drying furnace
21. Edwards Simplex roaster
22. Bucket elevator
23. Calcine bin
24. Corduroy table
25. Cone classifier
26. Hydrosizer
27. Ball-mill
28. Corduroy table
29. Thickener
30. Three Crosse tanks
31. Drum filter
32. Calcine residue dam
33. Antimony settling sumps
34. Antimony drying floors
35. Thickener
36. Brown tanks
37. Tailings dam

150 f.p.m. by a 5-h.p. motor. This belt discharges into the feed hopper of the bucket elevator. The grizzly oversize passes to the washing and sorting belt.

(3) *Washing*

The ore is washed by a water spray on the first portion of the sorting belt, which is here inclined at 15°. Washings pass down a launder to a box fitted with a ⅛-in. mesh screen. The fines and water flow by gravity to the main mill spillage sump, whence they are pumped by a 2-in. E.R.G. pump to join the east ball-mill discharge. The material remaining on the screen is removed at intervals through a chute and fed by hand into the east mill.

(4) *Sorting*

The washed ore is sorted on the horizontal part of the belt, which is 30 in. wide, driven by a 5-h.p. motor at 40 f.p.m. There are six natives sorting on each shift. About 30 tons per day (8 per cent) of waste is sorted and trammed to the waste dump. In addition, about 3 tons/day of high-grade antimony ore is selected at this point for subsequent cobbing.

(5) *Primary Crushing*

The sorting belt delivers the ore to a primary crusher, which is an Allen ‘ Stag ’ jaw crusher, 16 in. by 9 in., 270 r.p.m., 35-h.p. motor, set to 2–2¾-in. discharge. The capacity is approximately 16 tons/hr. Discharge from this crusher passes down a steel chute 12 in. wide at 32° to the feed hopper of the bucket elevator. A standby crusher of similar type is available.

(6) *Secondary Screening*

Primary fines, together with all crusher discharge, are elevated to the screen floor by an 11-in. wide bucket elevator, driven at 140f. p.m. bya 5-h.p. motor. Buckets are Linkbelt 10 in. by 6 in. The elevator delivers the ore to a 5-ft. 6-in. by 3-ft. 6-in. Robins Vibrex screen, set at 20°, and driven at 1,860 r.p.m. by a 5-h.p. motor. Screening is ¾-in. square aperture, oversize passing to the secondary crushers and undersize to a feed chute. This chute delivers the ore either to the West Mill ore bin (capacity 30 tons) or to a conveyor belt 20 in. wide, driven by a 5-h.p. motor at 180 f.p.m. and inclined at 17°. This conveyor delivers to the East Mill orebin of 300-tons capacity.

(7) *Secondary Crushing*

Screen oversize is carried by a 30-in. wide horizontal belt conveyor to the secondary crusher, a No. 19 Kennedy gearless springhead crusher, driven by a vertical 32-h.p. motor at 450 r.p.m. Two natives are employed on removal of tramp iron and some secondary sorting of waste on this belt. The waste thus sorted amounts only to about 3 tons/day. Another Kennedy crusher of similar type is available as a standby. The secondary crusher is set to give a discharge of about 1-in. size and the capacity is about 20 tons/hr., of which about 50 per cent is probably the normal circulating load. The crusher discharges through a chute to the bucket elevator, which returns the ore to the Vibrex screen.

II.—Milling

The milling section consists of two ball-mills, each in closed circuit with its own classifier. The circuit is slightly different in each mill.

(1) *East Mill*

Ore is drawn from the steel mill bin and fed by a Comet feeder and conveyor to a 5-ft. by 16-ft. by 6-in. mill, driven at 30 r.p.m. through a countershaft, pinion, and spur wheel at the inlet by a 150-h.p. motor. The mill is mounted on steel trunnions and lined with cast white-iron bricks. It is fitted with a ⅜-in. round-hole screen and lifting scoops. Three-in. steel balls are used and the load is maintained 8 or 9 in. below the axis.

Water is added to the mill feed with the classifier return to make the pulp discharge about 20 per cent moisture. The pulp flows out through a trommel with ½-in. holes, and pebbles are removed and returned to the mill by hand.

Discharge pulp is pumped by a 3-in. E.R.G. pump, 880 r.p.m., 15-h.p., to two blanket tables, a 16-in. by 24-in. Denver jig, and a 7-ft. dia. spiral classifier, which is driven at 7 r.p.m. by a 7½-h.p. motor. Sand is raked back to the mill, and overflow pulp at 80 per cent water flows to the flotation cells. Mill duty is 235 tons/24 hr. ground to 70 per cent *minus* 200-mesh.

(2) *West Mill*

The ore is hand-trammed from the steel bin in 1-ton cars to a 3-ton mill-feed bin. From the bin it is fed by a Comet feeder to a 5-ft. by 7-ft. by 4-in. ball-mill. The drive is similar to that of the East Mill, the speed being 25 r.p.m., with a 80-h.p. motor.

Discharge pulp flows to blanket tables and a 12-in. by 18-in. Denver jig and is pumped thence to a 7-ft. dia. spiral classifier. Sand from this classifier returns to the mill, but the overflow is passed to a cone classifier, the overflow from which flows to the gold flotation cell and the underflow returns to the jig. Mill duty is 115 tons/24 hr. ground to 70 per cent *minus* 200-mesh.

III.—Gold Concentration

All gold concentration equipment is in the mill circuit and the bulk of the gold is recovered here. At the same time the ore is prepared for the flotation of the antimony. Gold is recovered by straking, jigging, and flotation.

(1) *Blanket Strakes*

There are three sets of strakes: two tables, each 3 ft. by 7 ft. by 6 in. by 20 per cent slope in the East Mill circuit, one table 3 ft. by 4 ft. by 6 in. by 17 per cent slope in the West Mill circuit, and two tables each 3 ft. by 5 ft. by 6 in. by 7½ per cent slope treating final pulp. The tables are covered with corduroy blankets and are dressed at 2-hourly intervals. The concentrate from the final pulp tables is sent direct to the roaster, as it contains little free gold.

Concentrates from the mill tables are dressed daily on a James table. The top streak is taken and stored and the tailing collected in a sump. Once a week the top streak is re-dressed and the richest portion is removed for amalgamation, tailing running to the sump. The tailing is taken to the roaster at intervals.

Recovery by straking is about 20 per cent of mill feed, of which 17 per cent is by amalgam and 3 per cent is in the James table tailing. About 2½ tons/month of this material is collected for roasting: it assays 7–10 oz./ton.

(2) *Jigging*

There are two Denver mineral jigs in use; one 16 in. by 24 in. in the East Section and one 12 in. by 16 in. in the West Section, both double-compartment run at 300 r.p.m. by 1-h.p. motors. The bottom screens are standard wedge-bar type with ⅛-in. apertures. The bed is of iron slag broken to ¼–½ in. size and 2–4 in. deep, depending on the nature of the ore. The jigs are tapped hourly; no oversize concentrate is made and only the hutch product is drawn.

The concentrates from both jigs are dressed on the James table. The gold and pyritic minerals (about 95 per cent) are taken off the head of the table for roasting. The gangue passes into the tailing and is returned to the mill.

About 2 tons of dressed concentrate per day is produced, assaying 5–10 oz. gold per ton. This represents about 18 per cent of the gold in the mill feed.

(3) *Gold Flotation*

Flotation cells of the Denver Sub A No. 24 type are used for gold flotation, driven at about 300 r.p.m. by 7½-h.p. motors. Three are in the East Mill circuit and one in the West Mill circuit. The cells are all run as roughers; no cleaning is done. The concentrate obtained is roasted.

Normally, about 20 tons concentrate per day is produced at 7–12 dwt./ton. Talcy gangue will float in preference to antimony at this stage. This represents about 10–15 per cent of the gold in the mill feed.

The froth from both East and West Mill circuits is broken up by sprays and pumped by a 2-in. E.R.G. pump (1,260 r.p.m., 10-h.p. motor) to a Dorr thickener 18 ft. in dia. by 10 ft. (0·35 r.p.m., 1-h.p. motor). Clear water overflows back to mill water tanks and thickened concentrate is pumped by a 2-in. E.R.G. pump (930 r.p.m., 10-h.p.) to a Feinc filter, 4 ft. 6 in. in dia. by 6 ft. This machine is fitted with a plain scraper discharge and is run as necessary (about 12 hr./day). The filtered concentrate is shovelled to the drying floors.

IV.—Roasting of Gold Concentrate

All gold concentrate is dried and then roasted in an Edwards roaster. Calcine is stored for subsequent treatment.

(1) *Drying*

Concentrate from the mill and the Feinc filter is either sun-dried in paddocks or dried on a hand-operated drying plate 5 ft. by 25 ft. Coal consumption is about 1 ton/day and gravity concentrate is dried out completely, while flotation concentrate is dried to 5–10 per cent moisture.

(2) *Roasting*

The dried concentrate is weighed and sampled in barrows and hand fed to a screw conveyor, which feeds the material into the roaster. The jig and gravity concentrates are mixed with low-grade concentrate.

The roaster is an Edwards Simplex with 6-ft. 6-in. by 72-ft. 6-in. hearth (area 400 sq. ft.). There are 18 water-cooled rabbles, each fitted with 5 replaceable tines. The rabble arms are driven by a 15-h.p. motor through a reducing gear, line shaft, and bevel gears at 1 r.p.m., except the last two arms, which travel at 2 r.p.m. Opposite each arm is an inspection and draught door. There are two furnaces, one at the discharge end and one on the side about half way along the roaster. Both are hand stoked with steam coal. Temperature is determined by means of a Cambridge electrical pyrometer, with connections at Nos. 6 and 15 ports. Dust is caught in a system of flues and the gas is finally discharged through a steel smoke stack. Draught is natural and is controlled by a damper in the flue.

The roaster is stopped every morning. Rabbles are cleaned in rotation, inspection ports cleared and clinker chipped off the arch and bridges. Flue dust is removed from the flues. The roaster capacity is about 14 tons/day. It is difficult to determine the gold loss in roasting ; stack tests have indicated a loss of 3–5 per cent of feed. The object in roasting is to produce a 'dead' sulphide-free roast, with volatilization of as much arsenic and antimony as possible.

The loss of weight in the concentrate in roasting varies from 20 to 30 per cent. Iron compounds are converted to ferric oxide, Fe_2O_3 ; the bulk of the sulphur volatilizes, roasted calcine containing about 0·52 per cent sulphate and 0·18 per cent sulphide sulphur. Arsenic appears to be volatilized almost completely without much difficulty.

Antimony causes trouble, the cause appearing to be the fusion of stibnite particles, which prevents further oxidation and encases gold. Helpful factors seem to be insulation of stibnite particles by gangue as in low-grade concentrate and the existence of the antimony sulphide in coarse particles, e.g., in jig concentrates. In either of these cases antimony content can be much higher without causing trouble. Normally, about 60 per cent of the antimony present is volatilized, the bulk of the remainder being, apparently, in the form of the innocuous tetroxide Sb_2O_4.

The amount of coal used depends to some extent on the nature of the concentrate; more is required for low-grade material, owing to the absence of sulphur. Normal consumption at present is 2·4 tons/day, or 20 per cent of ore roasted. With high-grade feed only the end fire is stoked. Temperatures for high-grade feed are about 430–450°C. at No. 6 port (if higher, the stibnite fuses) rising to 600°C. at No. 15 port. The stack temperature where gas leaves the flues is about 220°C.

The flue dust removed daily from the flues is fed back to the roaster with the concentrate. Cyaniding tests on this material indicate excessive chemical consumption and only about 15 per cent recovery, but when roasted in this way it causes no trouble.

(3) *Calcine*

The calcine drops into a push conveyor, 12 in. wide by 50 ft. long, and is dragged to a bucket elevator, cooling off on the way. The elevator lifts it to a storage bin, where it cools further.

V.—Treatment of Calcine

About 10 tons of calcine is produced per day and is sluiced and pumped to a mill in closed circuit with classifiers and blanket strake tables. The final pulp is thickened and then cyanided and filtered.

(1) *Milling*

The calcine is fed from the storage bin to a chute, down which it is sluiced with water to a 2-in. E.R.G. pump (1,580 r.p.m., 7½-h.p.). The pump is run for one 8-hr. shift/day and delivers the pulp to a corduroy table 3 ft. by 5 ft. by 6 in. at 20 per cent slope.

Tailing passes to a cone 4 ft. 6 in. in dia. by 6 ft., the coarse underflow running into a mill 4 ft. in dia. by 9 ft., driven at 28 r.p.m. by a 50-h.p. motor. This mill is similar in design to the ball-mills, but is loaded with slugs. The discharge from the mill passes over a corduroy table 3 ft. by 5 ft. 6 in. by 20 per cent slope and is then pumped by a 3-in. E.R.G. pump (730 r.p.m., 7½-h.p.) back to the cone.

Cone overflow gravitates to a pan classifier, or hydrosizer, 7 ft. in dia. (3·1 r.p.m., 7½-h.p.). Coarse material is raked to the central discharge and flows into the mill. The pan classifier overflow is from 60 to 70 per cent *minus* 350-mesh and gravitates to a Dorr thickener 18 ft. in dia. by 10 ft. (0·35 r.p.m., 1-h.p.). The overflow returns to the mill circuit. Thickened underflow is pumped by a 2-in. E.R.G. pump (880 r.p.m., 7½-h.p.) to a tank for cyaniding.

(2) *Concentration*

There are two blanket tables in the circuit (already mentioned). The primary table is dressed once per shift, the other at hourly intervals. The concentrate is amalgamated without dressing. About 70 per cent of the gold in the calcine is recovered in this way.

(3) *Cyaniding*

Three Crosse tanks, 10 ft. in dia. by 4 ft. cyl. and 18 ft. 6 in. cone, are in use. Each has an inner and outer circular compartment served by separate air jets, so that pulp can be agitated in the inner compartment while settling in the outer one. Treatment is by the batch system.

Pulp is pumped in at a 1 : 4 liquid/solid ratio and is aerated with 5 lb. of lead nitrate for 2 hours to precipitate soluble sulphides, oxides and ferrous matter. Cyanide is then added in the form of a drip of 5 per cent solution until the strength is about 0·3 per cent KCN. Normally this takes about 6 hours, and 50 lb. cyanide (Cassels) is used. Agitation is continued for 12 hours in all, and the outer jet is then stopped. For settlement and precipitation 30 lb. lime is added and barren wash solution is run into the inner compartment at the rate of 1·8 tons/hr. For this purpose, precipitated solution from calcine treatment is used. The tank is full in 3–4 hours and clear solution at about 0·1 per cent KCN begins to overflow. Washing is continued for 16 hours and the feed is then stopped and clear solution decanted off.

The pulp is then gravitated to the Paxman filter and filtered and the residue is repulped with barren solution and pumped back to another Crosse tank for a second treatment. It is agitated for 20 hours with a further 10 lb. of cyanide at about 0·03 per cent KCN and is then filtered again and the residue is gravitated to a special dam.

The Paxman filter used for this work runs 3–4 hr./day and all solution from the filter and Crosse tanks runs to special clarifiers. Cyanide consumption is about 5·5 lb. NaCN/ton.

VI.—Antimony Recovery

The thickened pulp is conditioned with the necessary reagents and is then floated in a series of machines. The antimony concentrate floated is cleaned, dressed by gravity methods to recover gold and arsenic, and then collected, dried, and bagged for sale. Tailings are sent for thickening and cyanide treatment or are sent direct to the tailing dam.

(1) *Conditioning*

Pulp from the East Mill thickener is pumped at 1 : 1 to 1·25 : 1 liquid/solid ratio by a 2-in. Wilfley pump (1,330 r.p.m., 10-h.p.) to a conditioner 4 ft. 6 in. in dia. by 6 ft. deep (200 r.p.m., 7½-h.p.). The reagents are added at this point. A xanthate promoter is used. Pine oil is used as a frother.

The pulp is conditioned for about 10 min. and then passes to the flotation section.

In the West Mill circuit the reagents are added in the flotation cells.

(2) *Flotation*

The flotation plant consists of 7 No. 18 Special Denver Sub A machines (400 r.p.m., 5-h.p.) and 3 No. 24 machines (290 r.p.m., 7½-h.p.) in the East Mill, and one No. 18 and 4 No. 24 machines in the West Mill.

In the East Mill circuit the pulp from the conditioner enters No. 3 cell (the first rougher) and flows through seven rougher cells. The concentrate flows by gravity assisted by sprays to the two cleaner cells. From these cells final concentrate is drawn, and pumped by a 2-in. E.R.G. pump (700 r.p.m., 7½-h.p.) to the dressing plant.

Cleaner cell tailing passes back through the rougher cells, the tailing from which flows to a scavenger cell. The concentrate from this cell passes back to No. 5 rougher cell and the tailing is pumped by a 2-in. E.R.G. pump (890 r.p.m., 7½-h.p.) to a thickener, 35 ft. in dia. by 8 ft., similar to the mill thickener. Underflow is measured in Brown tanks and then runs to the tailing dam, while clear water overflows to a storage tank of 15,000 gal. capacity, which supplies the mill and flotation water service pumps.

In the West Mill circuit there is no conditioner. The pulp flows through four rougher machines, the concentrate from which is passed through one cleaner cell. The final concentrate from this cell is pumped by a 1½-in. E.R.G. pump (800 r.p.m., 5-h.p.) to the dressing plant and the tailing flows back to the roughers. The final tailing from the roughers joins the tailing from the East Mill circuit.

The control of the flotation plant is largely visual. For high-grade concentrate, froth should be dense and matted: for high recovery, it should be fairly loose. In practice a compromise is necessary. Pine oil must be carefully controlled; any excess leads to a drop in grade in the concentrate, while a deficiency causes high tailing losses. Xanthate is very helpful in stabilizing conditions.

The aim is to activate the stibnite sufficiently to cause it to float in preference to other minerals and schist.

(3) *Dressing of Concentrate*

The final antimony concentrate is pumped to a James table for dressing. In this operation some pyrite and arsenopyrite is separated from the main bulk.

(4) *Collection of Antimony Concentrate*

The concentrate flows to four collecting sumps, each consisting of a series of 4 compartments 2 ft. deep by 6 ft. by 12 ft. The bulk of the concentrate settles in the first compartment and the overflow passes on, clear water overflowing from the last compartment to a safety pump, whence it is pumped to the James tables and West Mill circuit water storage.

Each morning the settled solids are shovelled on to the drying floors. There are four concrete floors (total area about 8,000 sq. ft.) and additional drying space of about 10,000 sq. ft., some of which has been tarred. The maximum drying capacity is about 25 tons/day, in normal weather concentrate taking 3–4 days to dry out to 5 per cent moisture.

Jute pockets with inner paper liners (old cement bags) are filled to a weight of approximately 120 lb. net. The concentrate is sampled for moisture before bagging and the assay sample is made of portions from each bag.

The concentrate is despatched in 50 long-ton lots and transported to the station by railway lorries.

(5) *Cobbing*

The high-grade ore selected on the sorting belt is hand-cobbed, and selected ore over 55 per cent antimony is bagged and sold directly. The grade of cobbed ore has sometimes reached as high a figure as 69 per cent antimony, against a theoretical maximum of 71·4 per cent. Cobbed ore is sent away in 25-ton lots, the total monthly production being 30–50 long tons.

(6) *Distribution of Antimony*

Of the antimony in the ore, about 5 per cent is recovered as cobbed ore and 74 per cent as concentrate. The bagged concentrate contains 60–65 per cent antimony. If the grade is reduced, recovery can be improved, but, to satisfy market demands and for reasons of mining policy, the grade mentioned has been found to yield the best results.

As far as possible, nickel, copper, arsenic, and lead minerals are removed from antimony concentrate. All these impurities are heavily penalized. They are, however, largely removed with the gold from the ore.

VII.—Cyanidation of Flotation Tailings

The tailing pulp when treated is cyanided in Brown tanks and then filtered in a Paxman filter and discharged to the tailings dam. Gold-bearing solution is sent for further treatment.

(1) *Brown Tanks*

The thickened pulp from the tailing thickener is pumped by a 2-in. Wilfley pump (1,270 r.p.m., 7½-h.p.) to 4 Brown tanks, 15 ft. in dia. by 7 ft. 1 in. cyl. and 12 ft. 11 in. cone. The Browns are run on the batch system and as each tank fills the tonnage is determined by shortage and S.G. measurement.

The pulp in the tank is air-agitated for one hour and 10 lb. cyanide is then added, giving a strength of about 0·004 per cent KCN. Each tank holds about 50 tons dry slime. The charge is agitated for 8 hours, by which time all soluble gold is dissolved.

(2) *Paxman Filter*

From the Brown tanks pulp gravitates to a Paxman filter. This is a continuous rotary drum, 8 ft. 6 in. in dia. by 8 ft. long

(200 sq. ft. area). The drum is of steel with steel grating, covered with a coarse woven backing cloth and a twill filter cloth, and divided into airtight sections, each connected by a pipe to the filter end and automatic valve. The filter is driven by a worm and pinion at 0·53 r.p.m. from a 7½-h.p. motor. This motor also drives the agitator (15 r.p.m.) and puddles (54 r.p.m.) through a train of gears. The filter runs in a steel pulp hopper, with concrete spillage sump and a 3-in. E.R.G. spillage pump.

Vacuum is maintained by a Worthington horizontal pump, driven at 120 r.p.m. by a 5-h.p. motor. Filtrate is pumped by an Attack A3 water pump (1,850 r.p.m., 7½-h.p.) to sand solution clarifiers.

The pulp flows into the hopper and solution is drawn through the cloth, leaving a residual cake on the cloth face. The residual moisture is washed from this cake by barren solution wash sprays and the cake is then allowed to dry and is discharged into the puddler and repulped. From the puddler, the residue gravitates to the tailing dam at a 1 : 1 liquid/solid ratio.

The Paxman filter duty is very high, about 1·2 tons/sq. ft./day. It is used for tailing treatment for about 20 hr./day, and for Crosse tank pulp (calcine) 3–4 hr./day.

Gold recovery in this section is low and is not always profitable, and in that case the Brown tanks are used purely for measurement purposes.

(3) *Tailing Disposal*

The residue is run to a special dam. Walls are built by shovelling and water is drawn off by means of a penstock from a central pool. The water is not returned to the plant but is used for the compound garden.

VIII.—Gold Amalgamation and Precipitation

Blanket concentrates from the mill and from calcine treatment are amalgamated. The amalgam is pressed, retorted, and smelted into bullion. Gold-bearing solution from calcine and tailing treatment is clarified. The gold is precipitated in zinc boxes and these are cleaned up monthly. Slime is acid-treated, calcined, and smelted to bullion.

(1) *Amalgamation*

Calcine blanket concentrate is amalgamated daily, and dressed mill concentrate once a week, the amalgams being kept separate. The procedure in both cases is the same. The concentrate is placed in a steel amalgam barrel with about 5 lb. of chloride of lime, 400–600 oz. of mercury, and about 100 lb. of steel balls. The barrel is filled with water and rotated for 4 hours during the night. Next morning the barrel door is opened and the charge is run over a bucket, which collects nearly all the iron and amalgam. Overflow passes over an amalgamated copper plate and is pumped to the calcine thickener. The amalgam is washed, iron is removed by hand magnet, and the amalgam is squeezed in a pneumatic press.

The pressed amalgam is weighed and locked away and once or twice a month is retorted. A vertical pot retort is used in a Cornish furnace and mercury is collected in a condenser. The sponge gold is subsequently melted to a bar in a Cornish furnace (coke-fired). Amalgam contains 33–37 per cent fine gold, and mercury consumption is 0·05–0·1 oz./ton milled.

(2) *Solution Clarification*

Solution from calcine treatment and tailing treatment is clarified in four filter-bottomed clarifying tanks, each 16 ft. in dia. by 5 ft. with about 12 in. of sand on the bottom. Clean river sand is used. Suspended solids are filtered out and clear solution flows to the zinc boxes.

(3) *Precipitation*

Three six-compartment zinc extractor boxes are in use, each compartment being 2 ft. by 2 ft. 6 in. by 2 ft. 3 in., giving a total volume of 162 cu. ft. About 10 lb. of solid cyanide is added to the boxes per day. The boxes are ' dosed ' daily with lead nitrate, about ½-pint of 10 per cent solution being added to each box.

Precipitation is fairly efficient, box tails containing about 0·02 dwt. gold per ton. Copper, if present, may cause trouble owing to the zinc becoming plated : this can be overcome in part by the lead nitrate ' dosage.' In addition to gold and silver a great deal of base metal is precipitated—lead (from the lead nitrate), iron, antimony, and arsenic have all been found, and also sulphides. No ' white precipitate ' has been found as on the Rand. Filiform zinc is purchased, either cut on a lathe or ' spun.'

(4) *Clean-up*

Zinc boxes are cleaned up once a month. All zinc from the first two or three compartments is dissolved in dilute sulphuric acid in a wooden tub with stirrer. The balance of the zinc is jigged in wire screens in a bath of dilute acid in the open air to clean off impurities and is then returned to the first compartment. New zinc is then added to fill the boxes after being dipped in lead nitrate solution.

The washings and excess liquor from the boxes are pumped through a filter press and the slime is then removed and added to the acid tub. This is filled with clear water and, after settling, the clear water is decanted off and a second wash applied in the same way. The settled slime is then filter-pressed and dried by blowing air through it. The pressed slime is removed from the press and roasted in an oxidizing atmosphere in a calcining furnace (three-tray, single compartment). The filter press is a 24-in. Johnson press, served by a 2-in. Gwynne acid-resisting pump.

The calcine slime is charged into pots (No. 100) lined with clay liners. 200 oz. slime, with about 20 per cent borax, 20 per cent fluorspar, 35 per cent silica and 2½ per cent fine iron, is charged into each pot. No oxidizer is added. The pots are smelted 6 to 8 at a time in an 8-pot reverberatory furnace, poured into button moulds and allowed to set. Buttons are chipped off and remelted into bars.

The filter-pressed slime contains about 1–2 per cent fine gold, calcined slime 2–5 per cent fine gold, and bullion is about 900–970 per 1,000 fine gold.

POWER GENERATION AND DISTRIBUTION

The power plant is rather heterogeneous, having been added to from time to time during the life of the mine. There are two centres of generation. At the central power station, next to the reduction works, the prime movers are three Crossley-Premier 365-h.p. 4-cylinder horizontal gas engines, each of which drives a 250-kW 2,200-V. 3-phase 50-cycle alternator. The gas is produced from bituminous coal in three Crossley 'F' type 9 ft. 6 in. by 8 ft. dia. gas producers, the gas passing through scrubbers, tar extractors, and filters before being drawn into the engine cylinders. Also at the central power station is a 600-h.p. 4-stroke, 8-cylinder Nelseco diesel engine (from a scrapped submarine) driving a 300-kW alternator; and a 200-h.p. Ruston 4-stroke, double-cylinder, horizontal diesel engine driving a 128-kW alternator. The other power station is at the Weigel shaft, where there are four Babcock and Wilcox single-drum hand-fired boilers. A 200-h.p. Allen high-speed, compound, non-condensing steam engine is direct coupled to a 125 kVA alternator, which is used to drive the West Mill and is not synchronized with the central power station. Steam from this station is also used for the Weigel hoist and for a Belliss air-compressor.

The total monthly output of power is approximately 420,000 kWh at a cost of 1·83d./unit generated.

Power is transmitted to the Jack and Monarch sections by overhead lines supported on steel poles, using copper conductors of 0·06 sq. in. cross-sectional area. To reduce line losses, 3-phase transformers situated near the central power station step up the generated voltage to 6,600-V. Step-down transformers at line terminal points reduce voltage to 2,200. Power is supplied to the reduction works at generated voltage (2,200). For underground purposes, the voltage is stepped down to 500 and for lighting, surface and underground, to 220.

COMPRESSED AIR

Air compressors are situated at the chief shaft heads. At the Weigel there is a Belliss and Morcom vertical, steam-driven compressor, 2,000 c.f.m., and a Worthington, horizontal tandem, steam-driven compressor, 500 c.f.m.; at the central power station there is an Ingersoll-Rand horizontal compressor, 750 c.f.m., driven by a 130-h.p. electric motor. These units supply air to the Weigel and Free State mines, to the reduction works and to the workshops.

At the Monarch mine, air is supplied from a Sullivan angle compressor, 450 c.f.m., driven by a 75-h.p. motor, and a Holman vertical compressor, 180 c.f.m., driven by a General Motors diesel engine. At the United Jack mine, the compressors are an Ingersoll-Rand horizontal, 700 c.f.m., driven by a 120-h.p. electric motor and a Dingler horizontal, 400 c.f.m., driven by an 80-h.p. electric motor. Two portable compressors, a Holman 180 c.f.m., and a Le Roi 210 c.f.m., supply air for the Gravelotte mine.

The monthly total of air compressed is approximately 110,000,000 cu. ft. at a cost of 3·79d./1,000 cu. ft.

WATER SUPPLY

The main water supply comes from the Letaba River, on which the company has a pump station about 14 miles distant from the Weigel shaft. Here there are two units, each serving as a standby for the other. The first unit consists of an 8-stage Pulsometer pump driven through a countershaft by a 66-h.p. Ruston horizontal single-cylinder diesel engine; off the same countershaft is driven a 7½-kW D.C. generator, which supplies power to the small river pump. Connected to the suction side of this pump is a booster pump (Pulsometer 3-stage) driven also through a countershaft by a 28-h.p. Tangye oil engine. The second unit consists of a 10-stage Harland pump belt-driven by a 120-h.p. Ruston six-cylinder vertical diesel engine; a standby D.C. generator can also be driven by this engine.

The river pumping unit, which is mounted on a trolley that can be moved by a crab-winch up or down a track according to the height of the river, consists of a Pulsometer single-stage centrifugal pump driven by a 7½-kW D.C. motor. Suction is by a 4-in. hose equipped with foot valve and delivery by a 4-in. pipe line through a strainer to a 20,000-gal. masonry dam, which also serves as a cooling dam for the engine jacket water.

The rising main to the mine is a 6-in. dia. galvanized pipe line (buried) with screwed joints and flanges at every 200 ft. The line delivers to two 40,000-gal. galvanized-iron tanks situated on the top of Maid of Athens kop. There is sufficient static head at this height to feed to any part of the central area.

Each pumping unit is capable of delivering 10,000 gal. of water/hr. to the mine, the actual delivery being about 220,000 gal./24 hr. The cost per 1,000 gal. delivered is 1s. 0·93d., and per ton milled, 8·47d.

In addition to the above water 70,000 gal./day is pumped from the Weigel shaft, 12,000 gal./day from the Banded Ironstone shaft, and 2,000 gal./day from the old Free State shaft. The cost of underground pumping is 1·53s./ton milled.

Water from the Letaba River contains a great amount of suspended solids during the summer months and is unsuitable for domestic purposes. Recently a filter plant has been constructed to overcome this defect. Water from a reservoir passes through a meter and ball valve to a 1,000-gal. steady head tank, from which it flows through a regulating valve down a wooden launder into the first compartment of the filter. At the head of the launder, two wooden boxes, containing respectively 10 per cent solutions of alum and sodium aluminate, drip feed into the stream, the quantity of reagent added varying between $1\frac{1}{2}$ lb. and 5 lb. of each per 10,000 gal. of water, according to the dirtiness of the water. No automatic feed is used for the reagents: with practice, a native attendant can set the feed as required. A beaker of treated water is taken daily from the end of the feeding launder and tested for flocculation and *p*H value by the assayer.

After leaving the launder the water travels through three settling compartments in series, the overflow from each compartment decanting into the next. Coconut fibre matting curtains are hung in the settling pits across the flow of the water to reduce turbulence. The three settling pits are drawn off every 2 hours through 2-in. sludge pipes when the river is in flood and at 8-hr. intervals when the river is normal.

At the end of the third settling compartment, a cross-launder delivers the water to either of two filter compartments. These have 4-in. pipes with 3-in. laterals drilled with $\frac{5}{16}$-in. holes laid on the bottom and connected through inverted U-tubes to a short launder delivering to the clear water reservoir. The collector pipes are covered for a depth of 4 in. with *minus* $1\frac{1}{2}$-in crushed stone, then 6 in. of *minus* $\frac{3}{4}$-in. gravel and 4 ft. of coarse river sand. The filter beds are cleaned as required by water under pressure and air at a pressure of 5 lb./sq. in. being forced up through from the bottom and running away through a pipe placed just over the filter beds. The filter beds have a capacity of 80-gal./sq. ft./hr. and are capable of a much larger throughput than the present 1,250,000 gal./month. During its passage through the short launder into the clear water reservoir, the water is sterilized by the addition of 2 lb. of chloride of lime and 1 lb. of ammonium chloride per 100,000 gal., fed at 10 per cent concentration from glass bottles.

LABOUR

The total European labour force is 116, made up as follows:

Administrative and secretarial	8
Medical, compound and police	3
Mining	30
Engineering	50
Surveying, sampling and assaying	7
Geological	3
Reduction works	15
	116

Of these, 31 are monthly paid officials and the remainder day's pay workers.

The total native labour force is 1,030, made up as follows:

Mining	**368**
Reduction works	**228**
Engineering	**271**
Compound	**69**
Offices	**35**
Survey	**14**
Assay	**7**
Sanitation	**18**
Diamond drilling	**20**
	1,030

The tribal classification of natives employed is:

Transvaal Basutos	**85**
Transvaal Shangaans	**90**
Rhodesian Shangaans	**70**
Rhodesian Blantyres	**225**
East Coast Shangaans	**560**
	1,030

The surrounding country is sparsely populated by natives and practically all employees engaged come from further afield. All labour is voluntary and the company has no recruiting agents. Natives are engaged on a four-months' service contract, but mostly work on far beyond this period. The mine has a good name among the natives, but owing to the great demand for native labour difficulty is experienced in maintaining the required labour force.

MEDICAL AND SOCIAL SERVICES

The company employs a full-time medical officer and maintains a native hospital. European cases are sent to Leydsdorp hospital, which is 18 miles distant. There is a strong Benefit Society covering members and their dependants against ailments which are outside the company's responsibility.

About two-thirds of the European employees live in company houses or in the mess quarters which the company provides for the single employees; the remainder live in houses of their own construction. A building construction programme is in progress. Houses are built of bricks with thatch roofs, the bricks being machine-made on the mine from a mixture of 9 parts screened boiler ash to 1 part cement.

The district is malarious, but the company's houses are well screened and cases of fresh malarial infection in residents are rare. A regular programme of internal spraying with DDT is carried out during the summer months and pyagra is available free at all times to employees.

A recreation club with canteen, library, and billiard room has been built by the company, and facilities are provided for tennis, swimming, boxing, and rugby football; there is a cinema show twice a week. A Government primary school exists on the property which has at present about 120 pupils.

Native labour is controlled by a compound manager. The compound is better described as a native village, since it is unenclosed and boys are allowed to have their wives and families living with them. Huts are built of wattle and daub with thatch roofs. The company maintains a large vegetable garden and meat is purchased from local butchers. Rations consisting of mealie meal, vegetables, and meat are issued uncooked. An organized native association football club exists and regular matches are played on the compound field. There is a mission school in the compound for native children and the native teacher also conducts adult classes in the evenings. A cinema show is given in the compound once a week.

Compound feeding costs for 1947 were 6·26d.; hospital expenses, 1·25d., and other compound expenses, 2·8d., all per shift worked. The percentage of shifts lost through sickness was 0·71.

COSTS

The costs given in Table I are for the year 1947. Advantage is being taken of the present good price of antimony to carry out a heavy development programme, extensive renewals of equipment, and improvements in facilities which are reflected in working costs.

TABLE I

	Cost per ton milled Shillings	
Stoping	7·41	
Developing	4·85	
Diamond drilling	0·55	
Hoisting	5·00	
Pumping	1·53	
		19·34
Transport of ore		2·36
Sorting and crushing	1·64	
Roasting	1·35	
Tube milling	5·49	
Flotation	4·93	
Cyaniding	2·05	
		15·46
General expenses—mine		5·14
General expenses—head office		0·88
Gold realization charges		0·10
Total		43·28

The unit prices of commodities delivered on the mine given in Table II are presented for comparison purposes:

TABLE II

	s.	d.
Mining timber, per cu. ft.	1	1
50 per cent ammon. dynamite, per 50-lb. case	29	8
⅞-in. hexagon hollow drill steel, per lb.	·	7⅛
Cement, per 94-lb. pocket	3	5
Diesel oil, per gal.	1	3¾
Lubricating oil, per gal.	5	6
Petrol, per gal.	2	2
Coal, per ton	25	5
3-in. steel balls, per lb.		2⅜
Cyanide, per lb.		9¼
Sulphuric acid, per lb.		3¼
Xanthate 301, per lb.	1	6
Lead nitrate, per lb.		9
Mealie meal, per 90-lb. bag	12	11
Compound meat, per lb.		5

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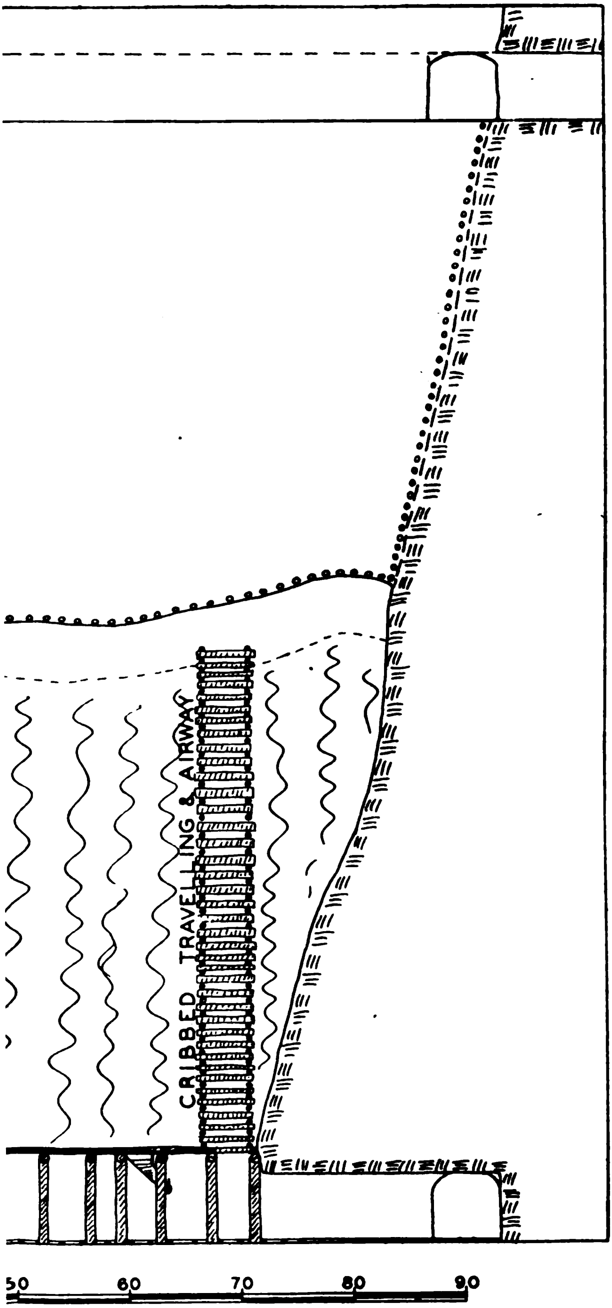
Plate I.



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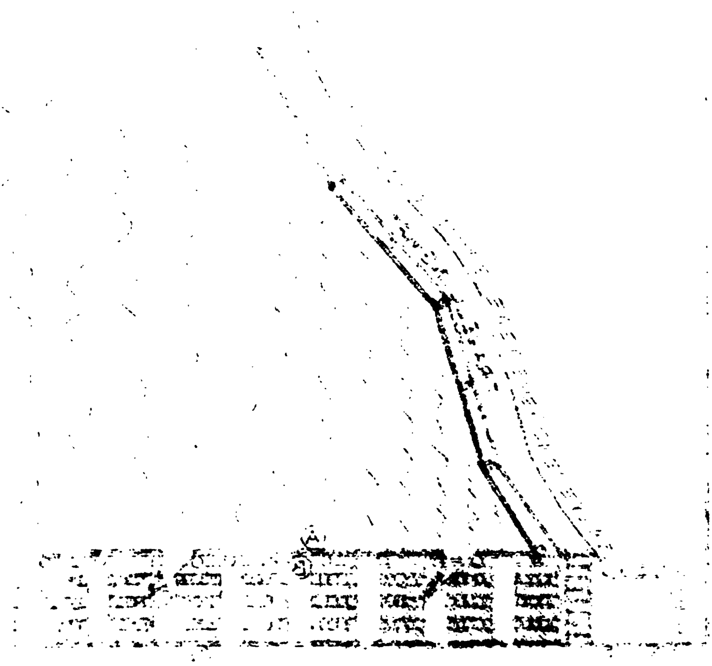

FIG. 6.—Panorama of central area, looking east (continued in Fig. 7 below).

FIG. 7.

The small-scale plan showed mines on the Far East Rand, the bend in the reef, and also the important position occupied by No. 5 Shaft in opening up a new and a key area, as the farm Withok to the south had not been touched except for bore-holes, and there was an almost unknown area to the west. From the commencement this mine had needed courage and optimism in its development, and perhaps the greatest courage was shown in the decision taken in 1945 to sink vertical and sub-vertical shafts to the Main Reef Leader, which was expected to lie at a depth of between 7,000 and 8,000 ft.

The larger-scale plan showed Van Dyk with its neighbours, E.R.P.M., New Kleinfontein, and Brakpan mines, and the pay-shoot which had kept Van Dyk going. The length on the strike, in a N-S. direction, of the Van Dyk property was about 5 miles, while its width on the dip, in an E-W. direction, was about 2½ miles.

The decision to sink No. 5 vertical and subvertical shafts to a great depth had, therefore, to be taken without any knowledge of the width or value of the reef, such as might have been gained from bore-holes or underground workings in the vicinity. The shareholders had had to forego dividends to pay for the shafts, and the producing area of the mine had to be subjected to secondary and tertiary development because, as happened so often on the Far East Rand, when primary development was poor in value, subsequent splitting up of large blocks by subsidiary development exposed payable blocks in a seemingly-unpayable area.

Mr. Reid then read Mr. Blunt's own introductory notes which he had sent to the Institution, as follows:

'Shaft sinking must always play an important rôle in mining, particularly where lodes lie at depth. Improvements in technique in this work are of vital interest, as shafts are sunk on a capital charge and expenses are heavy with no revenue with which to offset such expenses. Where depths up to and over 6,000 ft. are normal the question of ventilating the workings is of vital importance. This is a governing factor in determining the shape of shafts.

'The elliptical shaft is not new by any means; such shafts were sunk half a century ago and were lined with dressed stone. Examples of this are still existent in many parts of Britain, although it is doubtful whether the reasons for sinking such shafts were identical in the past with those of the present. The present-day elliptical shaft is essentially designed to give the hoisting capacity of the rectangular shaft with the ventilating capacity of the circular shaft. The No. 5 shaft at Van Dyk is intended to hoist a minimum of 100,000 tons per month, as well as to carry a minimum of 600,000 c.f.m. of air to the deep-level workings of the mine. As the workings in this area are estimated to be at a depth of from 6,000 ft. to possibly 9,000 ft. or even 10,000 ft. vertical depth from surface, ventilating capacity is a major problem. It is the author's considered opinion that more heat can be dissipated by large quantities of air circulating in a mine than by smaller quantities of conditioned air being employed. The cost of conditioning air is exorbitant as compared with that of using fans to circulate large quantities of air. Shafts on the Witwatersrand have to be large to handle the men, material and rock usually associated with this field, so that if one adds shape to these large shafts, air capacity may also be obtained.

'Speed of sinking has received the attention of mining engineers for a very long period in southern Africa, so that due consideration to this aspect must be given. In both rectangular and circular shafts 30 years ago 300 ft. of sinking in a month had been accomplished. From that time on world records were constantly broken. The West Rand Consolidated raised the record for one month's sinking to 420 ft. in 1929; Vlakfontein Gold Mining Company in 1936 sunk 422 ft. in one month; in 1940 West Rand Consolidated again raised the figure to 454 ft. in one month, and in 1941 Van Dyk Consolidated Mines sunk 461 ft. in a month. Lined shafts had, of course, not gone down at such speeds as these, although Crown Mines and New Modderfontein had, I think, sunk 300 or more feet in a month in lined circular shafts.

'It was our opinion that in order to place a monolithic concrete lining in a shaft and obtain some measure of speed the shuttering should be of a permanent nature, as in the use of temporary shuttering time has to be lost waiting for concrete to set and also in withdrawing the shuttering. For this reason the concrete bricks were designed.

'The use of a clamshell for cleaning out a shaft should help to speed up the work in European countries where labour for this work is not as freely obtainable as in South Africa. It is this availability of labour which makes the footages sunk here possible, as the general aim is to clean out, blow over, drill, and blast at least a 4-ft. round per shift, providing no difficulties are encountered with water, bad ground, etc. To introduce a mechanical contrivance for this work is, however, more difficult here, as the feeling is that it is easier for the ganger to supervise 70 or 80 shovellers than to learn to operate any machine. The mechanical tradition is not yet strongly enough developed to be a great deal of assistance to mining engineers in South Africa.

'The care used in employing 16 plumb bobs in setting up the curb ring at each lift of walling was definitely worth while, since when the shaft was completed and the equipping with steel buntons at 10-ft. centres was commenced, the whole shaft was equipped, including the installation of the steel guides, in a period of 7 weeks. The buntons were placed by means of a jig hung on the four stage ropes. This jig was held in position, once the buntons had been correctly positioned, by four jack bars and the buntons were then grouted in the bunton pockets with wet concrete (rapid-hardening). An average of 10 sets of buntons per day was accomplished by this means'.

Concerning the grab Mr. Blunt had written to the speaker:

'From the British and European standpoint, one of the biggest points in the paper is the use of the grab for cleaning out a shaft where labour is not as plentiful as it is here. The Priestman level-cut grab is much more suited to cleaning spoil out of a shaft than the ordinary clamshell, as it has the action of two scraper scoops being drawn towards each other, and therefore it is able to pick up even the most difficult rock. There were certain improvements which we made on a model which would be most difficult to explain in an airmail letter but, should anyone be interested, it could be covered in a drawing'.

The author, being a mechanical and mining engineer, was obviously regretful that the grab experiment was discontinued. Under 'Labour' on p. 12 he said that the underground complement was 98 natives per shift and of these, say, 70 were shovelling broken rock towards or into the bucket. Assuming that a perfected grab would save half that labour, there would be a saving of 100 natives on the three shifts; multiplying that by the 16 large shafts being sunk in the Odendaalsrust and Far West Rand areas, it would seem that the grab was as important in the Transvaal as anywhere in the world, particularly as there had not been a sufficiency of native labour in the last 20 years.

The cleaning out of a round in the confined space of a vertical shaft by a gang of Basutos working on contract was something to be seen to be believed. They established themselves in rows on both sides of the bucket, the natives on the front bent low and threw the rock upwards, while the natives in the intermediate and back rows threw the rock with their shovels perhaps 20 ft. horizontally at a high velocity over the heads of the front row to land with a thud against the inside of the skip or bucket. The unwary visitor, however important, found himself unceremoniously pushed to the sidewall if he got in the way. One felt that a lasher in the front row had a poor chance of survival should he raise his head above the top of the bucket. It would have been noticed, however, that over the period of 21 months there were only 3 fatal and 199 non-fatal accidents, of which only 75 were reportable—by which was meant that the injured person was incapacitated from doing his normal work for 14 days or more—an average of 3·6 per month. Quite a few of the 75 were eye injuries due to blowing-over before drilling commenced.

Mr. Blunt had told him that the sequence of carrying on with the sinking of the sub-vertical was delayed owing to the fact that the stage hoist arrived 16 months and the main winder 18 months late, and also because water had been encountered numerous times. Whenever water was struck work had to be stopped in order to cement it off, which was a tedious job and wasted a tremendous amount of time. The sub-vertical had reached a depth of 1,400 ft. at the present time (November, 1948).

The hydraulic cylinders for taking up differences of less than 1 ft. in the length of the stage ropes (mentioned at the bottom of the second page of the paper) were mounted directly on the stage. Each piston and crosshead was in guides and the crosshead was attached to the guide rope by means of a strap. Four were used and all that was needed for trimming was to release a little oil from the required cylinder or to pump them a little with a portable pump. This was much quicker than turnbuckle adjustment.

Concerning the birdcaging of the main winding ropes (also second page), Mr. Blunt informed him that : ' Birdcaging starts by making the rope look like a mild form of corkscrew and later develops into a complete opening-up of the strands. Its cause is obscure, some engineers being of the opinion that it is a fault in the making of the rope. Personally, I think our trouble was due to the deflections we were using. In ordinary winding it is usually due to tight grooves in the sheaves or on the drums of the hoist, or both'.

It was interesting to record that the 110-hole round consisted of 6-ft. 6-in. holes in an endeavour to obtain a 6-ft. round. The average tonnage hoisted was 40 tons per ft. advanced, so that after a good round was blasted the shovelling crew was faced with the task of cleaning out 240 tons. The average time to clear a round was 5–5½ hrs. on a trouble-free run.

In the second paragraph of the section ' Progress and Alterations ' it would be noticed that the author said that the curb ring was placed on matt packs, that is, sections of 4 or 5 short timbers, joined side by side, each timber usually being 2 ft. 6 in. long by 6 in. wide and 4 in. high. The curb ring was removed after the completion of each lift of concrete, so that the bricking and its concrete backing were self-supporting. The bricking and concrete fill were as strong as a monolithic concrete wall in that each course was a combination of four horizontal arches, which were tightened by the pressure of the concrete fill. It seemed a feat of masonry that nearly 4,000 ft. of vertical elliptical shaft with a long axis of 35 ft. and short axis of 15 ft. had been bricked with sufficient accuracy of workmanship and high quality of material to render unnecessary the periodic support of the bricking by the conventional curbs supported on the side wall of the shaft.

Including the first two months, December, 1945, and January, 1946, when only 70 ft. and 64 ft. were advanced during the learning period, and despite the delay in September, 1946, due to cementing a water fissure and the delay in three of the first four months of 1947 due to rope trouble, the average monthly footage sunk and lined for the 21 months was 181 ft., a high average which reflected the detailed planning and careful organization, as well as the co-operation received from the men doing the work both underground and on the surface.

The sinking of vertical shafts had been of peculiar interest and significance to mining engineers on the Witwatersrand since the days toward the end of the last century when plans were implemented first by the engineers of Consolidated Goldfields of South Africa to sink what were then deep-level shafts of between 3,000 and 4,000 ft. in depth. Shaft names such as Rhodes, Catlin, and

Howard, on what was now the Simmer and Jack Mines, had been followed 50 years later in the same mining group by the Annan Shaft at Doornfontein Gold Mining Co. on the Far West Rand.

Close connection between the Rand and the Institution in respect of shaft sinking had been maintained since the earliest days. In preparing his notes the speaker had read the section headed 'Shafts' in Vol. I of the Papers and Discussions for the years 1942–1945 of the Association of Mine Managers of the Transvaal. He thought that a debt of gratitude was due to the Transvaal Chamber of Mines for publishing those papers, which were classics in their respective fields and he hoped that they were available to all members of the Institution. In that volume, in his introductory notes on his paper (p. 103), 'Some notes on the re-opening of the Catlin Shaft of the Simmer and Jack Mines, Ltd.,' Mr. H. H. Taylor referred to Professor Truscott's book, *The Witwatersrand Goldfields*, as follows: 'As Truscott says, "All the rates of sinking which are now being made are very great advances on those of two or three years ago, when 50 ft. per month was a fair average over an extended period".' The date of publication of Professor Truscott's book was 1902 and the period referred to was 1898.

In the same volume there was a paper by Mr. J. Daniel, bearing particular reference to the Union Corporation Group, entitled 'Notes on the design of an elliptical shaft,' which was introduced by the author early in 1945, at a time of great shaft-sinking activity on producing mines. Many contributions of the greatest interest were made to the discussion, in particular by managers of mines on which elliptical shafts were being sunk. In Daniel's paper reference was made to the constructional details of the Rand's new vertical shafts given by Brigadier R. S. G. Stokes in his paper 'Recent developments in mining practice on the Witwatersrand' which was presented to the Institution in November, 1935, at another period of intense shaft-sinking activity both on producing and non-producing mines. In October, 1941, a paper by R. B. Smart entitled 'Some notes on the sub-vertical shafts, Vlakfontein Gold Mining Co., Ltd.' was submitted for discussion before the Institution after having been introduced by the late Colonel Edgar Pam.

Those few examples, out of many, illustrated the personal touch which had been maintained between members of the Institution and shaft sinking on the Rand, and Mr. Blunt's paper was, in effect, a thread in a pattern spread over many years—a varied pattern, because shafts, like human beings, had their own problems and idiosyncrasies, and no two were alike.

The speaker asked, 'What of the future?' Perhaps they could learn from the past. In 1930 before the Union went off the gold standard there were four non-producing but developing mines whose shaft-sinking footage was 3,546 ft. In 1935 the equivalent figures were eighteen mines and 16,710 ft. In 1940 the war brought the figures back to four and 3,708 ft., in 1945 to one and the footage nil. The climax of the present vertical shaft-sinking programme, due to past and continuing discoveries in the Orange Free State, would probably not be reached until 1950 or later, but in the meantime it was of interest to summarize the vertical shafts being sunk at the present time on non-producing mines.

New Consolidated Goldfields Group accounted for six; on Doornfontein the Annan Shaft and No. 1 Shaft were both circular, with inside diameters 22 ft. 1 in. and 24 ft. 1 in. and estimated final depths 4,500 and 5,500 ft. respectively. On West Driefontein, of Nos. 1, 2, 3, and 4 shafts, the first and the last two were circular, 22 ft. 1 in. inside diameter, with estimated final depths of 4,500 ft., while No. 2 was elliptical with long axis, inside concrete, of 33 ft. 1 in. and short axis 16 ft. 7 in., and with an estimated final depth of 5,500 ft. The vertical shafts on Libanon, Venterspost, and Vlakfontein had recently been completed, and on Vogelstruisbult the elliptical shaft (33 ft. long axis and 16 ft. 6 in. short axis, inside concrete) was more than 6,000 ft. in depth, with an estimated final depth of 6,500 ft.

On non-producing mines in the Anglo American group, the Welkom Gold Mining Co. had two vertical sinking shafts, Free State Geduld Mines one, and Western Holdings two, while on the President Steyn shaft-sinking had not yet commenced. No. 2 shaft on Welkom was a seven-compartment rectangular shaft, 46 ft. by 10 ft. with concrete bearers throughout, no timber being used in its construction. That type of shaft was, he believed, being adopted as standard by the Anglo American group in the Orange Free State.

In the Barnato group, the Free State Development and Investment Corporation (' Freddies ') had four sinking shafts in its large leased area, two in the north and two in the south. Those were rectangular shafts 47 ft. by 11 ft. 6 in. inside timbers, with estimated final depths in the north of 5,600 and 5,300 ft., and in the south, 5,600 and 5,700 ft.

Last, but by no means least, Union Corporation, which was first in the Orange Free State field at St. Helena, had completed a 30° inclined shaft and was now sinking No. 4 vertical shaft, elliptical in shape, with long axis approximately 40 ft. and short axis approximately 12 ft.

In conclusion, the speaker thanked Mr. Blunt for giving him the privilege of introducing his paper. He wished to leave with members an impression of the independence of thought and action shown by the engineers of the different mining groups in their approach to shaft-sinking problems, and to express the hope that an even closer contact would be maintained between them and the Institution in their present and future activities.

He also thanked Mr. C. T. Pott, of Union Corporation, for lending of the slides he had exhibited.

Mr. Jack Spalding said that during a recent visit to the Rand he had the pleasure of being shown by the author the work described in the paper. The author's design of concrete brick to form a mould for the monolithic lining of the shaft had much to recommend it, as the time taken to remove steel shuttering was saved. He, the speaker, was prepared to find that when the concrete was poured it would tend to bulge the bricks inwards on the sides of the shaft, but saw no evidence of that when riding down in the bucket. He understood that, during pouring, the concrete in the ends was kept a little in advance of that on the sides.

When he saw the primary shaft it was very wet, considering that it was to serve workings 8,000 ft. deep, where the provision of air with a low wet-bulb temperature was essential. It was anticipated by the staff, however, that the natural self-sealing properties of the water would close all the smaller fissures within a year. Certainly a number of points which had obviously been making water in the past were already dry; nevertheless, in sinking the secondary shaft, the policy was to cement every fissure, thus causing many delays. It would be interesting to hear in due course whether the small fissures left unsealed in the primary shaft did eventually dry up.

On p. 10 there was a small error in the text—the ' 48-ft. Brown fan ' should, of course, be ' 48-in.'.

The sliding pipe within the main fan pipe for rapid extension and withdrawal for blasting was an excellent idea. The amount of air drawn into the small space at the junction, and therefore lost to the sink, was not large, as the resistance to flow of the short length of smaller diameter pipe was trivial. That method had much to recommend it. The Rand practice of interconnecting the lights with the signal bell was also an excellent one, for it was impossible to hear any signal in a shaft sink when the machines were running, whereas the temporary flicker of the lights was immediately obvious to all without being sufficient to hinder operations.

The author mentioned the erection of a central wall to divide the shaft, after completion, into two portions for ventilation purposes. That was common Rand practice with elliptical shafts. The method used in other mines there was to cast a groove down the concrete lining of the shaft on both sides and on completion of sinking to build a wall across from groove to groove, either of monolithic concrete or of precast concrete beams lowered into place and rested one on top of the other. There seemed to be several disadvantages to this method. First, the shaft sides were weakened by the groove made in them, although in the very good standing ground on the West Rand that did not seem to matter. Secondly, several weeks' extra time was taken to build up the central wall during which no other work could be done in the shaft. Thirdly, the wall was not available to provide two-way ventilation until after sinking was completed.

There would seem therefore to be a good case for casting the central wall integrally with the lining during sinking, which would incidentally strengthen that lining considerably. That would be of particular advantage on the Far West Rand, where sinking was done through very bad ground. The stronger construction of the lining would permit the use of a very flat ellipse—the most desirable shape for fitting the equipment into the shaft. In bad ground the elliptical shape could be departed from in favour of two segments of circles backing on the central wall as a common chord. Other advantages of monolithic construction of lining and centre wall were that no appreciable extra time would be taken in sinking and as soon as the shaft were down it would be ready for equipping. Further, with a central wall, fan pipes would not be necessary and with a low water guage fan a very much larger supply of fresh air could be provided for the sinkers than when piping was used. Thus the sinkers could return down one side of the shaft while the blast smoke went up the other.

There would, of course, be a number of practical difficulties to be overcome in instituting such a method of sinking and lining. It would be necessary to divide the Galloway stage into two separate pieces, one in each part of the shaft. There was no insuperable objection to this. It would also be necessary to leave a window in the central wall for communication between the two sides, and for the supply of concrete mix from the one side to the other when the concrete being poured was being joined up to that already in place above. Those windows could be built up solid as soon as the lift of concrete was complete.

He would congratulate the author on the practical way in which he had described the operation, thus rendering the paper of maximum value to mining engineers.

Prof. J. A. S. Ritson said he assumed that the shaft was required for hoisting as well as ventilation, otherwise he saw no reason why its shape should not be circular instead of elliptical. Circular shafts up to 3,000 ft. in depth and 25 ft. finished diameter were at present being sunk. Those shafts each carried four skips or cages and left ample room for ventilation and service pipes, etc. A circular section eliminated the inherent weakness of an ellipse which was bound to be unstable unless the prolongation of the smaller radius passed through the centre of the larger circle. When bricks were laid in bond, the junction of the two arcs was bound to vary, hence the weakness at the junction.

With regard to the rope, a Lang's lay rope was notoriously liable to twist. He assumed that the outer strands were given a clock-wise twist and the inner strands an anti-clockwise twist. Solid wire cores were rarely satisfactory, because the core tended to break into short lengths. He did not think there was any justification for a solid core unless the rope was going to be wrapped in two layers on the hoisting drum. With a drum of 14 ft. diameter, there did not appear to be any necessity for double wrapping, because the fleet angle would not be excessive.

The breaking strain of the rope was high, a more usual figure being 100–110 tons for 1½-in. rope. That might mean the use of 'extra special' improved plough steel for the wires. He had found that the higher the breaking strain of an individual wire the more brittle it became.

Birdcaging or spiralling were much the same thing; birdcaging occurred in ropes with crossed lays and spiralling in ropes of unidirectional lays.

Rotation of a rope about its own axis was a common cause of trouble. He asked if the tread of each pulley was a correct fit for the rope—i.e., diameter of tread 6–10 per cent greater than diameter of rope. Was it possible that one of the pulleys (deflection or head-frame) was not properly aligned in the temporary lay-out? If so, the rope would land on the inside of one flange and have to roll down into the tread, thus upsetting the balanced lays of the rope.

The President said, apropos of Mr. Reid's remarks, that the paper under discussion was only one of a series of interesting papers on the subject he understood their friends in South Africa, the Chemical, Metallurgical and Mining Society of South Africa, were about to publish in the form of a symposium. The President of that Society, Mr. F. G. Hill, was at the Meeting and he extended a welcome to him and hoped he would be able to make some remarks.

Mr. F. G. Hill* said that an approach had been made to the six mining groups of South Africa asking whether they would be prepared to give their most up-to-date knowledge of shaft sinking for a symposium planned by the Chemical, Metallurgical and Mining Society of South Africa. They were all very helpful and for the next three months two papers would be presented at each meeting of the Society, each giving the views of the representative of a particular group. Most groups could provide from experience the most up-to-date information on shaft sinking, and the Orange Free State would receive a good deal of attention. He had been in the Orange Free State the previous month and it seemed to him that all the shafts were going to be rectangular, with the exception of St. Helena. Mr. Reid had said that the shafts were 47 ft. by 11 ft. 6 in. inside diameter, but his own recollection was that the dimensions were inside the concrete. In other words, it would be a concrete *cum* steel shaft and a significant change in shaft-sinking practice.

On the Central Rand, generally speaking, timbered shafts had been the rule for years, and on one of the mines with which he was associated they were about to sink a deep-level shaft and a special vertical shaft to go down to 8,000 ft. There was a conflict of ideas—some people favoured timbered shafts, others steel and concrete—and figures were prepared to see the difference it would make in the amount of air going down the timbered shaft and the concrete shaft. The results were startling. It was found that with the same water gauge the timber would take down 700 c.f.m., whereas with a steel and concrete shaft there would be 1,800 c.f.m. It was tremendously important to have a large volume of air in a shaft

*President of the Chemical, Metallurgical and Mining Society of South Africa.

8,000 ft. deep ; indeed, it was the most important single factor. If there was a small volume of slowly-moving air it heated up so quickly that it did not give the relief which was derived from quickly-moving volumes of air. The East Rand Proprietary mine calculated that the saving on power costs on sending down 900,000 cu. ft. was tremendous, and that it would, in fact, pay to spend up to £35 extra per ft. to put in steel and concrete—that figure being equivalent to the extra power which would be needed in a timbered shaft in 15 years.

Those were interesting facts. What was being done in the Orange Free State was significant. A great deal of thought had been given to the best type of shaft for the hot conditions which prevailed in those areas, and in the forthcoming symposium reasons would be given for the way the shafts were being designed. It was important to know not only how a shaft was constructed, but the reasons why a particular shape was adopted. He thought the discussions after the papers would be most illuminating. It was intended to have discussions not at the meetings when the papers were presented, but at further meetings, so that the views of all the people interested in the subject could be heard.

The President thanked Mr. Hill for his contribution, and said that it had been suggested that the Institution might like to have copies of that very interesting series of papers and after the meeting he proposed to discuss the proposal with Mr. Hill.

Mr. J. B. Richardson said it was a great pity that the successful attempt to clean out the broken rock from the shaft bottom mechanically had to be abandoned for the reasons given. Could not the operational training on the machine be mostly done on the surface, Europeans induced to operate it and the power transformed down to a much lower voltage ? The much-discussed serious shortage of native underground workers, coupled with the large future shaft-sinking programme outlined by a previous speaker, surely justified the cost of investigating from every angle the use of such machines. On the Rand, where groups were constantly co-operating in so many other ways, might not the expense be shared by those groups concerned in sinking deep shafts in the near future and not borne wholly by a single company that had already half-completed their shaft ?

In June, 1948, a compressed-air operated, power-shovel type of machine was described that was operating successfully after many months' work in an American mine. The timbered, rectangular, 17-ft. by 7-ft. shaft was smaller than the one described in the paper. It was wet and the rock hard to drill and blast. It was claimed that that machine could be operated by two men and could clean the shaft in a third of the time taken by hand-shovelling.

His sympathies were with the author who was forced to stop working on an ingenious and labour-saving mechanism which, perfected, might have benefited shaft-sinking operations all over the world. He hoped he would be given another chance.

CONTRIBUTED REMARKS

Mr. H. C. T. Brown : This is an interesting and practical paper, which raises one or two points regarding the use of concrete in shaft linings that call for some criticisms, as they appear to violate accepted standards of the application of concrete.

From the author's description and from the dimensions given in Fig. 2, it seems safe to assume that the principal object of the pre-cast brickwork was to form shuttering, and that the strength of the brickwork in itself was of secondary importance, the poured concrete constituting the lining proper. An adverse factor of considerable importance in the use of such permanent shuttering is that the concrete placed behind it can have no opportunity of becoming properly cured and that it will not develop its maximum strength. Proper curing of the concrete cannot take place before the shuttering has been struck and the surface of the concrete exposed to the atmosphere. If the brick were of substantial design and constituted the main strength of the lining, the nature of the back-filling would have less importance. With regard to the hollow spaces enclosed by the sides of the brick it seems doubtful whether the whole would be filled with poured concrete, particularly the undersides of the top face, and whether proper adherence would be obtained with the surfaces of the brick. The leaving of cavities in this space and of honey-combing at the surfaces of the brick seem difficult to avoid unless an excessively wet (and weak) mix were used, and unless adequate tamping were done. No mention is made of the nature of the mix, apart from a statement of aggregate proportions ; its water content, slump, etc., are not given. Proper tamping would appear to be impossible in the spaces enclosed by the bricks, particularly when at least two courses, or 40 in., are left unfilled above the top of the concrete. The use of wooden dowels for connecting the segments is a further source of non-homogeneity in this section of the lining. Under these circumstances, the bricked section of the lining contributes little to the total strength.

The use of the 'octopus' is a great time- and labour-saving device. This method of placing concrete is not recommended if a first-class, impermeable lining is required, however, because of the great height from which the concrete must fall from the receptacle. Although no figures are given in the text, the impression is that the concrete travels through a vertical distance of 25–30 ft. During each discharge the heavier constituents become segregated near the bottom, with the final result that a lining of variable consistency is obtained. The careful preparation of the aggregates, grading, washing, mixing, etc., are wasted if the concrete is to be mishandled in placing. Is the author convinced that he has obtained a lining of adequate strength, and of impermeable construction, that will withstand the stressing caused by heavy traffic running on rigid guides ? It might be argued that a lining of thinner section and of really first-class construction made with steel shuttering, with careful placing and tamping, or pneumatic vibration, etc., would have proved no more costly. It may well be that the reduced volume of ground to be excavated, raised and disposed of, reduction in the quantities of materials used, and of power consumption, and so on, in the case of the thinner lining, would have permitted as great a saving in time and in cost as that gained by the use of permanent shuttering in a shaft of this size. Would the author state what proportion of the total cost of the concrete lining, as shown in Table III, is for permanent shuttering, and would he give the lining thicknesses employed at different depths of the shaft ?

In regard to the use of common bricks as an experiment in the early stages of the sinking, and with which it was apparently the intention to line the whole shaft, apart from the time factor and the need for bricklayers, it would have been thought that the fact that ventilation difficulties were to be expected would have dismissed any idea of using this type of lining. The shaft rubbing-surface friction on the air current becomes an important factor in deep shafts and the relatively-smooth surface of poured concrete compared with ordinary brickwork, particularly after the brickwork has been in place for some time and requires pointing, offers considerable advantages in this respect.

It is a pity that the mechanical mucker was not in operation for a sufficient length of time to enable the author to give results of its performance and to allow of some comparison with other types of shaft mucker. Would it not be possible to adapt the machine to compressed air for use in wet conditions ? It is not clear how the skeleton is supported in the shaft ; the lower end appears to rest on the material being excavated and seems to be in danger of being undercut. Is the machine supported directly by its hoisting rope during operation ? An independent form of support would seem to be preferable. Has the driver some form of protection, other than his hat, from falling material ?

DISCUSSION ON

Diamond-Drill Blast-Hole Practice at the Roan Antelope Copper Mine*

By H. F.-C. Nevill, *Associate*, and W. K. Burgess, *Associate*

The President said that the second paper presented for discussion, dealt with diamond-drill blast-hole practice at the Roan Antelope copper mine. The joint authors were sectional and acting sectional engineers for Roan Antelope Copper Mines, Ltd., and had presented a most interesting paper on that comparatively new mining technique. It was an excellent example of the value of such papers in disseminating knowledge for the general advancement of mining practice.

Diamond-drill blast-hole practice already had applications probably not thought of when it was first used, and the presentation of the paper describing experience would stimulate ideas and further applications of the practice. Unfortunately both authors were in Africa but Mr. R. M. Peterson was present and had undertaken to introduce the paper.

Mr. R. M. Peterson† regretted the absence of the authors who had collected and recorded considerable data on the practice of diamond blast-hole drilling at Roan Antelope over a period of five years from its inception in 1942. The paper covered a wide field and included a reference to the geology and to the mining methods employed at Roan Antelope in which the use of diamond-drills played a part. It might be useful at that stage to make some reference to the mining conditions which existed at that time and which bore a relation to the date on which diamond-drilling became an active programme.

When the Roan Antelope mine started production in 1931 the factors which were important in deciding a choice of mining methods were: (i) the primary necessity for a low-cost method, in order to survive at the prevailing low cost of copper, and (ii) the need for effective use of native labour, which had no opportunity to become trained and which in any event did most effective work in repetitive operations. For those reasons a sub-level method was decided upon and adapted to the dips varying from 13° to 90°. Pillars of the minimum size were left. In several sections of the mine the stopes were carried without leaving any pillars until such time as the hanging caved and the maximum span which could be carried was determined by experience. From

*Bull. 504, Nov., 1948.

†Technical Director, Roan Antelope Copper Mines, Ltd., and Mufulira Copper Mines, Ltd.

a production point of view it was necessary to lay out the pillars so that the opening of the next stope could be done in a systematic manner and a regular production provided from the various sections of the mine.

That method of mining was successful and offered no difficulties until the mine approached the 820-ft. level. Previously, local caving had taken place and in some cases had broken through to surface. Complete relief was obtained in some areas, but not in all. As a result, pressures were built up which made it necessary to adopt a new method of mining which relieved those pressures.

Conditions varied between the flat dip sections at the eastern end of the mine and the steeply-dipping sections on the limbs. In the flat-dipping sections a soft schist occurred below the economic foot-wall. That distributed the pressure and also caused the pillars to punch into the foot-wall. Complete caving could not take place under those conditions and the pressure was transmitted to the working faces so that a considerable amount of timbering had to be done concurrently. In the steeply-dipping section the open sub-level stopes had been carried from the 820-ft. level to the oxide zone, which was roughly at the 100-ft. level. Those high stopes were successfully mined and then partial caving of the hanging-wall took place and in some cases the caving took place to surface and the stopes were filled with the caved material. The result was that the hanging-wall was not broken and the cantilever beam exerted pressure on the workings at the base of the fulcrum. It was necessary, therefore, to break that cantilever beam and the first application of blast-hole drilling took place in that connection. Previously that shearing had been done by means of powder tees.

The paper described the drilling of those blast-holes in Section II. The holes were drilled to create a line of weakness by shearing the hanging-wall beds. The work was very successful.

The principle used in all the new methods was to work out a small unit in a short space of time and to break and partly recover the pillar. No support was left and the country had to cave progressively once the initial break-through had been accomplished. Diamond drilling fitted into that programme, because the drilling could be done ahead of time and therefore the number of days required to work out the stope was governed largely by the rate at which the broken material could be drawn through the draw point, rather than the time consumed in drilling and blasting.

The authors had gone into great detail in describing the various applications of diamond drilling and he was certain they would be glad to reply to any questions which might be put forward. As to the mining methods, the members of the Roan Antelope organization would be very glad to enlarge on any of those aspects which might be of interest. The authors had concluded with a note on the probable trend of blast-hole drilling at the mine. The tendency was, he thought, to take advantage of diamond blast-hole drilling in the lay-out of new mining methods.

Mr. A. R. O. Williams said that he was glad to hear Mr. Peterson say that his company would supply further details of the Roan stoping practice. The description in the paper was rather too concise and, to him, difficult to follow.

On the main theme of the paper he had one or two comments to make. The authors in their discussion of the relative merits of small and large diameter holes at the end of the paper stated that the larger ones gave better fragmentation. That rather surprised him. During the recent war he had been associated with the excavation of some large chambers in a hard limestone at Gibraltar. Some of them were stoped with diamond drills and their experience was that the larger the diameter of the holes the smaller the number required, but the higher the proportion of large blocks in the spoil and, therefore, the greater the amount of secondary blasting. They had tried diameters up to 3 in., but found that approximately 1½ in. was the most suitable dimension.

Priming of the holes at the Roan was with Cordtex, which, he assumed, meant that a length of Cordtex lay along the full length of each hole. He would like the authors to give further details of their practice. He had himself used that method, but on one occasion an explosion occurred during the charging of a horizontal hole which injured, not seriously, the charger and his helper and killed a miner who was passing beneath the drilled ground. That accident was very closely investigated and, although the cause was not definitely established, it was thought to be due to the Cordtex. It seemed that the brass-jointed drain rods that were being used for charging the powder had, during their forward and backward movement over the Cordtex, kinked and then split the wall of the latter where it bridged one of the small fissures commonly found in the limestone of that locality and had spilled out some of the filling. Experiment had shown that whereas Cordtex did not detonate when struck quite heavily the filling itself was highly sensitive to even light blows.

He was particularly interested to read that European women did the diamond setting at the Roan. He had been told on more than one occasion that the acquisition of this craft required many years of apprenticeship, but he recalled that at Gibraltar two R.E. sappers, with no experience of diamond setting or, in fact, of diamond drilling, had, after a few weeks' training with the Royal Canadian Engineers, done the work with great success.

In Table IV under ' Blasting factor ' the reference to 80 per cent collars was not clear.

On p. 9 the authors stated that two bits of different diameters, an Ax and an Ex, penetrating at similar rates, had sustained similar diamond losses. He might have misunderstood their argument but it seemed to him that the larger bit had done more work and that therefore the diamond loss per foot drilled would have been greater.

Assemblies of observations covering extensive and well-planned tests such as had been carried out at the Roan and recorded in the paper were, he believed, of very great value and especially so to operators of the smaller type of mine where facilities for experimental work were often much restricted. He felt that all would be grateful to the Roan Antelope Company for permitting the excellent paper to be laid before the Institution for the benefit of the mining industry.

Mr. G. F. Laycock said that the part of a most useful and instructive paper which he found of particular interest was the comparison made throughout between the advantages and disadvantages of coring bits as against non-coring. As one who had been on the side of coring bits for a long time past, he was interested to learn that the trend at Roan Antelope was away from non-coring types, except in crushed ground. He was afraid that non-coring bit enthusiasts would not derive much satisfaction from the paper, but he realized, of course, that no two mines were exactly alike—so much depended on the nature of the ground to be drilled. One of the chief advantages claimed for non-coring bits was that, as there was no core to pull, there was less delay, since there was no plugging or emptying of core-barrels. It was, however, often overlooked that even with non-coring bits it was necessary to pull and grease the rods at frequent intervals which took up time. In crushed ground the coring bit was admittedly at a disadvantage, but he was not at all sure that the non-coring bit was the complete answer.

He appreciated that the paper was about diamond-drill blast-hole practice and consequently any reference to ordinary rock-drill holes might be out of place but he wondered whether a heavy-type rock-drill specially arranged for drilling long holes had been tried and, if so, with what results. It was also possible that that kind of drill-hole might obviate a great deal of the difficulty experienced in drilling through the manganiferous gouge containing loose rounded pebbles on the foot-wall of the ore shales, which must be a drill operator's nightmare ; he could not think of anything more terrible to have to drill through with diamonds.

Very little reference was made in the paper to the length of the holes drilled, but judging from some of the sections given it would appear that many of the holes were comparatively short. There was a limit, of course, to the depth to which rock-drill holes could be drilled, but it might be of interest to members if he mentioned that he knew of one mine where a lot of that kind of drilling had been done, some of the holes reaching 100 ft. or more in length. The trick in that achievement lay in the design of the rods and the couplings. The cost of that particular kind of drilling for short holes was only a fraction of the cost of diamond drilling.

Mr. J. A'C. Bergne said that he had read the paper with great interest as he was on the Roan Antelope about 20 years ago, when they were just starting to mine after the initial development period, and he congratulated the authors on writing an interesting account of an innovation.

The greatest item of cost besides labour was that of diamonds, and he noticed that the diamonds used were labelled West African boart, at a cost per carat of 15s. or approximately $3, which was a very cheap grade of diamond. It might be that one of the outstanding reasons why that method was adopted in preference to ordinary drilling with percussive drills was that the diamonds were exceedingly cheap—the present market price for West African selected boarts was several times that figure. It would be interesting if the authors could give any information they had as to any experiments made with different types of diamonds. He would have thought that Congo ‘ drill rounds ’ which were cheaper would have suited them better at an even lower cost.

Another point was the character of the ore at the Roan Antelope. It was a very peculiar rock in that it was soft, yet strong enough to need no support in development openings, and just hard enough to avoid the clogging of drill-steel water-holes with a muddy sludge ; thus it was the percussion drillers' ideal. They used to achieve routine rounds of 11 ft. 6 in. and better with mounted machines and even hand-held drills used regularly to obtain 7 ft.–6 in. rounds in one shift in the main haulages, so that under those conditions it was hard to understand why the rock-drill should have been discredited for the stoping as distinct from the drilling of the long holes to bring down the hanging-wall. He would be very interested if the authors would elucidate that point.

Mr. D. H. Shute wished to make a few remarks on the manufacture of diamond bits. He noticed (p. 18) that some difficulty had been experienced in peening the diamonds and their subsequent brazing in handset bits. Also there had been some criticism that mild steel was insufficiently resistant to abrasion when used for the bit body.

There was a technique which had proved very successful in making diamond dressers for grinding wheels in the engineering industry which could well be applied to handset diamond bits. The method consisted of hot-pressing iron or bronze powder round the diamond in the form of a cylindrical slug of diameter slightly greater than the diamond, which was set at one end of the slug. In making wheel dressers the slug was pushed into a hole drilled in the end of a steel shank. It was set firmly into the shank by brazing, using an acetylene torch and one of the commercial brazing alloys. That method of setting diamonds was not nearly so expensive as it sounded.

The abrasion resistance of the body might be improved by using an air hardening nickel-chrome steel instead of mild steel. It would be necessary to drill and turn the body with the steel in the annealed condition, insert the hot-pressed slugs, and then heat the body to about 900°C. for brazing. Simple air cooling from that temperature would ensure the necessary hardness in the steel.

Comment was made by the authors that, in general, the matrix of the powder-set bits was too soft. It was possible by powder-metallurgy technique to achieve almost any degree of abrasion resistance, even to the extent, if necessary, of setting the diamonds in a cemented carbide matrix. That technique had proved successful in making wheel dressers. In most rocks, however, cemented carbides would prove excessively abrasion resistant and would probably fail to wear sufficiently to enable the diamonds to be free cutting. It was also possible to provide localized areas of the bit which had a greater abrasion resistance. Greatest wear usually took place on the periphery and around the water holes. It was possible to introduce powdered cast tungsten carbide into the metal powder mix in those areas and so reduce wear.

Finally the speaker asked if the authors had had any experience of reversed water flow drilling for shot-holes.

Dr. A. W. Groves said that it would be of general interest if it could be made clear whether the average price mentioned of 15s. per carat was based on standard market prices or whether the company was in a specially-favoured position with regard to the purchase of drilling boart. He had marked recollections of the Royal Canadian Engineers giving most valuable assistance in diamond drilling in Great Britain in 1942 and some of those drillers were very fond of a mixture consisting of two-thirds West African drilling boart, costing (at that time) about 12s. 6d. a carat, and one-third fine brown stones at 30s. a carat. The fine brown stones were a little smaller than the drilling boart. Had the authors carried out any experiments with mixtures of that kind?

Professor J. A. S. Ritson referred to the expression '' average dynamic air pressure ' used by the authors at the bottom of p. 8, and asked if it should not be ' average working air pressure '.

The author discussed Ex and Ax types of bit, but it would have been much more useful to student engineers if the authors had mentioned what the various ' x 's meant.

In Table IV, one column was labelled ' loading efficiency', etc. Did that not mean ' length of hole to contain 50 lb. of explosive ' ? The question of ' efficiency ' did not appear to arise.

Part III of the paper was headed ' Theory of Drilling', but theory was not discussed and the section described the results gained from use of the drills.

Under the section on explosives and blasting practice, there was a reference to stemming, and he noted that they used a sand-clay mixture. That was introduced many years ago, but was becoming outmoded. In many British mines a coarse sand was blown into the holes by a compressed-air injector, and was working very well, giving better results than sand-clay and being infinitely better than plain clay, or no stemming at all.

Mr. J. B. Richardson said that it was stressed at the beginning of the paper that Roan Antelope were the first people to apply the diamond-drill blast-hole system described to a soft orebody with weak walls. On p. 27, large holes were preferred by the authors. The ore was soft and friable, however, and drilling was presumably done well in advance of blasting, as elsewhere. On the same page it was indicated that some of the holes were lost by the sides of them falling in. The two important facts the authors had left out were: (*a*) The percentage of holes drilled that were lost, and (*b*) the maximum time a hole could safely be left before charging.

The President said that there was no time for further discussion and asked other members who wished to contribute to do so in writing. Owing to the lateness of the hour, he proposed an omnibus vote of thanks to the authors and the introducers of the two papers and to all those who had taken part in the interesting discussion.

AUTHOR'S REPLY TO DISCUSSION* ON

The Tin Mines of the Pahang Consolidated Co., Ltd.

By F. H. Fitch, *Associate*

Mr. F. H. Fitch: I must first apologize for the delay in replying to the discussion on my paper. At first, I was digesting the very interesting comments which were passed at the meeting and waiting for publication of the remarks which I knew Messrs. Way and Abel were making; later I was fully occupied on special duties due to the present emergency in Malaya.

I am most grateful to Mr. Willbourn for his excellent introduction and summary of my paper, undertaken at short notice owing to uncertainty as to the date of publication, and to all members who took part in the discussion.

The point which raised the most discussion was the origin of the lode channels. Several speakers were critical of the theory presented in my paper, but none of the alternatives suggested appears entirely satisfactory. The main difficulty is to explain the disposition of the Bells and Willinks East series of lodes relative to Willinks fault. Bells lodes are strongly developed near Willinks fault and should be found on its east side, but lodes with the same east-west strike do not occur immediately east of the fault. Again, Willinks East lodes are strong in the upper levels near Willinks fault, but no lodes with the same 120° strike are known west of the fault. Under east-west compression movement on Willinks fault was such that if Bells lode channels were broken by it their eastward dislocated extension would have been moved south-west—that is, in the direction in which Willinks East lodes are now found. It seems to me imperative to correlate the Willinks East and Bells series of lode channels and to assign to them the same primary origin. All the alternative explanations of the fracture pattern suggested by members in the course of the discussion, except that put forward by Dr. Williams, attribute these two series of lode channels to different causes.

The idea presented in Dr. William's Fig. 12 at first appears to offer a solution which still assigns the same primary origin to lode channels east and west of Willinks fault, but does not necessitate the introduction of the idea of rotation. It does not, however, account for the formation of the lode channels of Simons and Kabang lodes, which strike east-west, although they are between Willinks and Kabang faults. In the case of Simons lode it has been proved that practically no movement took place between the walls (see p. 18): the lode cannot therefore occupy a transcurrent fault, as it would be expected to do if the strain ellipsoid shown by Dr. Williams on Kabang fault is correctly orientated for the time at which that lode channel was being

*Bulls. 495 and 498, February and May, 1948.

formed. In my proposed solution of the problem Simons and Kabang lodes were considered to occupy tension fractures developed after the main rotation of the Willinks East lode channels had been completed, although I am quite willing to admit that I have no direct evidence of that order of events. Moreover, it is possible that the orientation of strain ellipsoids in various parts of the area changed at different stages of the development of lodes and that the addition of the idea of late formation of the Simons and Kabang lode channels to Dr. William's hypothesis would provide a satisfactory solution of the whole problem.

Another point which raised some discussion was the failure of payable lodes to penetrate deeply into granite. Several members realized that my effort to explain this on temperature differences alone did not work out in detail, and Messrs. Way and Abel rightly warned against any assumption that temperatures would have been the same in widely-separated parts of the property. Dr. Schnellmann drew attention to the possible importance of the country-rock as an ore-control. As a result of the discussion and of further work in the area, I now believe that we must look to the chemistry of reaction between country and igneous emanations for the explanation. I would like to suggest as a working theory that in the Sungei Lembing area tin was carried as stannate or thio-stannate in solutions which also contained abundant magnesia and possibly iron, with little or none of such alkalis as potassium or sodium, and that volatiles such as fluorine and boron were not essential to the transport of tin.

Although tourmaline was observed at Sungei Lembing by Mr. Scrivenor, and recently confirmed by Dr. Ingham in the same specimens, it is certainly rare. I have examined mill concentrates, as suggested by Mr. Scrivenor, and can find no tourmaline. In *every* specimen of ore that I have collected, chlorite, or secondary minerals which I consider to have been derived from chlorite, are present with the cassiterite, usually in contact with it. In some specimens of slate breccia from the lodes the fragments have been partly chloritized, leaving a core of unaltered slate, and cassiterite fills the interspaces between the fragments. It is evident that the elements required to convert the clay minerals of the slate into chlorite—namely, magnesia and iron—were extracted from the ore-fluid and that this led to the deposition of tin in cassiterite. I note that Dr. Webb has also observed an increase in iron in the country associated with tin lodes: both he and Mr. Scrivenor remark upon the common association of cassiterite with chlorite in Cornwall. Although not as closely associated with cassiterite as the chlorite, sulphides and sulpharsenides are common in the lodes at Sungei Lembing, so that the ore-fluid must also have contained variable amounts of sulphur and arsenic. The presence of tin and sulphur in the ore-fluid recalls a recent paper by F. Gordon-Smith* in which he described experiments proving that

*GORDON-SMITH, F. Transport and deposition of the non-sulphide vein materials. II—Cassiterite. *Econ. Geol.*, Vol. 42, May, 1947, No. 3, p. 251.

alkali thio-stannates can exist in conditions analogous to those of nature. As there are no minerals rich in potassium or sodium in the lodes at Sungei Lembing but there have been increases in the magnesium and iron contents of the country enclosed in the lodes I suggest that the tin was carried as stannate or thio-stannate in solutions which also contained abundant magnesium and probably iron, reaction with the clay minerals of the slate removing magnesia from the ore-fluid, reducing the alkalinity, and bringing about the deposition of tin oxide.

When the type of ore-fluid suggested was passing through unaltered granite there would probably be little or no reduction in alkalinity, as magnesia would not be expected to replace the stronger alkalis of the felspars and micas. This would explain the lack of cassiterite deposition in granite at Sungei Lembing, but in Ulu Reman, 12 miles north of Sungei Lembing, cassiterite occurs in granite. The most marked difference in mineral association between the two areas is the abundance of tourmaline in Ulu Reman. The abundance of this mineral in one tin deposit in granite and its rarity in another nearby in argillaceous sediments may signify a fundamental difference in the composition of the ore-fluid and in the chemistry of deposition of cassiterite.

Although there is some evidence in the field for these theories, it is clear that they will have to be tested by analyses of unaltered country from the lodes. It is hoped that these analyses will be made before the memoir on the area is published by the Geological Survey.

It now remains to answer some of the less general questions and comments raised in the discussion. I have to admit that Mr. Burdon's criticism of the evidence given for the age of Willinks fault (p.25) is justified ; that given under (*a*) is valid, but the other points are capable of alternative explanation. Mr. Burdon was of the opinion that the fault might have acted as a dam to mineralizing solutions and caused the mineralization on the two sides to differ. Certainly it acted as a dam to meteoric waters, but even if it were also impervious to mineralizing solutions, it would not have given rise to any difference between the lodes on the two sides of the fault unless there were distinct and differing centres of tin emanation. I do not think that alternative emanative centres existed, as the lodes are too much alike in mineral content throughout the east-west length of the mines.

Dr. Williams criticized my use of a quotation from Bailey Willis (p.20) ; this did not refer to the swinging of the fractures out of alignment but to their being torn open and rendered capable of mineralization. The idea can be appreciated if Dr. Williams's Fig. 11 is treated as a plan and not a section, which is reasonable, as movement on transcurrent faults like Willinks fault was largely horizontal. If such movement can give rise to tension gashes when no planes of discontinuity exist the opening of any earlier fractures formed by another process and suitably orientated with regard to the fault is most probable.

The manner in which the granite was emplaced was another point which interested Dr. Williams. I have studied the structure around the Sungei Lembing granite and, although dips of the rocks of the sedimentary cover on the north-west are variable in direction, the granite appears to occupy the core of a dome. On the other hand, the southern granite, just seen in the south-west corner of Fig. 1*a*, is transgressive to the bedding of the sedimentary rocks from south of Sungei Lembing to the River Kuantan, south of Pasir Kemudi, a distance of more than 12 miles.

Thrust faulting was also mentioned by Dr. Williams. There is some evidence for it in the crosscut south to the ventilation winze on Bells adit level, mainly coincident with the bedding planes of the slate. I have not the data necessary to answer the other query as to the net effect of vertical movement along the minor breaks east of Willinks fault, nor do I think it can be answered unless a marker horizon is discovered.

I have re-examined Kabang lode in view of the comments made by Messrs. Way and Abel on its quartz content and now realize that our difference is one of terminology. I have used the word ' lode ' to include the full payable width—i.e., stoping width—of a deposit, but it is customary at the mines at Sungei Lembing to restrict it to the quartz veins or zones of gouge which were followed in development. Values usually penetrate into the walls of such structures and I still think it is fair to say that quartz rarely forms more than 10 per cent of the payable width of lodes, using that term in my sense.

A similar difference in terminology appears in the last paragraph of the remarks of Messrs. Way and Abel. I have been careful to make a distinction between ' lode channels ' (potential or actual sites for tin deposition) and ' lodes ' (restricted to tin-bearing structures). Making this distinction, their comment would read ' Lode *channels* certainly penetrate the granite . . . ' but lodes, in my sense, fail to penetrate deeply.

With regard to the value of Kabang lode, I stand corrected ; I had not seen the more strongly-mineralized parts of that lode before the war.

Concerning the future development of Myah mine (p. 22) I should perhaps have said that the ' zone of *potential* payable mineralization ' would be carried down below economically-accessible depths. Lodes only occur in this zone where there are suitable lode channels to be mineralized. Myah and Myah South lodes pinch out in depth, but other lode channels may come in to replace them and carry values.

In reply to Mr. Neil ; there is certainly a tin belt including Gambang, Sungei Lembing, Ulu Reman, and Bundi, described under ' The Eastern Tin-Belt ' in Mr. Scrivenor's *Geology of Malayan Ore Deposits*, p. 116.* Since the war, interest in the Trengganu part of this belt has been shown by numerous mining

*London, Macmillan, 1928.

companies and the Government has posted a geologist and an inspector of mines to Kemaman to investigate it.

I agree with Mr. Robinson that it would have been better to quote levels in all the mines relative to the datum level of Willinks adit, but I know of no existing survey which relates the workings in the outlying mines to that datum. The confusion over the naming of the ventilation shafts arises from the fact that the Company use the word 'shaft' only when they intend to install winding gear: in the text of the paper (p. 9) it seemed wrong to speak of a circular *winze* when referring to something 14 ft. in diameter, although the Company still uses that term. Simons lode is not affected by either Willinks or Kabang fault, but by minor shears complementary to them, striking NE-SW. Fig. 5*b* shows that the lodes increase in number as Willinks fault is approached from the west.

Dr. Dixey criticized my use of the term 'radial' in referring to the movement of the block of slate between the two major faults around the granite buttress. The term was used because I wished to give the impression that tangential movement of the parts of the block farthest from the granite was greater than that near it, similar to the movement of the spokes of a rotating wheel. I should, however, have used the expression 'radial' in reference to the movement of the fractures, which are linear in plan, and not to that of the block as a whole.

Mr. R. B. Fermor asked several questions about transport in the mines. Mr. Fairmaid, General Manager of the Pahang Consolidated Co., Ltd., has given me the following information, which is not as complete as he would wish owing to loss of records by the Company during the Japanese occupation. General conditions of underground transport in the mines, before the occupation, were good, the track being maintained in good order and all trucks equipped with ball- or roller-bearings. Development ends were warm, but normal trucking from stopes was through ventilated drives and crosscuts. Diesel haulage was cheaper than electric haulage per ton mile, but no figures are available. The cost of haulage by diesel and electric locomotives was about 30 per cent of the cost of hand tramming, including the filling of trucks from chutes. The number of labourers on indirect haulage who were transferred to direct-production jobs when mechanical haulage was introduced is not known, but no labour was dismissed. The servicing of haulage locomotives by Asiatic labour, under European supervision, was good.

In conclusion, I should like to say once more how much I appreciate the interest taken in my paper and particularly the valuable and detailed comments made by Drs. Williams and Webb.

In the post-war period, during the unwatering of the main mines, production has been maintained by re-opening some of the outlying mines and I have now been able to examine them. The new evidence is not at variance with the ideas I have tentatively expressed and I hope, with the aid of the constructive criticism and comments passed on my paper, to be able to prepare an acceptable comprehensive account of the mines for publication in a memoir of the Geological Survey of the Federation of Malaya in the near future.

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NOTICE OF GENERAL MEETING

The Fifth Ordinary General Meeting of the Fifty-Eighth Session of the Institution of Mining and Metallurgy will be held, by kind permission, in the Apartments of the Geological Society, Burlington House, Piccadilly, London, W. 1, on **Thursday, 17th February, 1949**, at 5 p.m.

The paper entitled **Ground control—theory and practice**, by Mr. Jack Spalding, *Member*, which is published in this issue of the *Bulletin*, will be submitted for discussion, and will be introduced by the author.

Light refreshments will be provided at 4.30 p.m. for members and visitors attending the Meeting.

The Council invite written contributions to the discussion of papers from members who may be unable to be present at the Meetings of the Institution. The Council reserve the right to edit and condense such contributions.

INSTITUTION NOTES

Annual Dinner, 1949

The Annual Dinner of the Institution, which was not held last year, will take place on Thursday, 5th May, 1949, at the Savoy Hotel, Strand, London, W.C. 2. Full particulars will be published later.

Mond Nickel Fellowships

A number of Fellowships to the value of £650 to £850 each, are again being offered in 1949 by the Mond Nickel Fellowships Committee, to men and women of British nationality with a university degree or similar standard of education, with the object of providing additional training to make them more suitable for eventual employment in senior technical and administrative positions in British metallurgical and metal-using industries.

The Fellowships, which will normally be travelling Fellowships, will occupy one year; selected candidates will be required to undertake a programme of training in industrial establishments in the United Kingdom or elsewhere; awards for training at universities may be made in special circumstances.

Full particulars and application forms can be obtained from the Secretary, Mond Nickel Fellowships Committee, 4, Grosvenor Gardens, London, S.W. 1, to whom completed forms of application must be returned by 1st June, 1949.

Capper Pass Awards

In 1947, the Directors of Messrs. Capper Pass and Son, Ltd., Bristol, sharing the regret which had been expressed in many quarters at the dearth of papers on processes and plant used in extraction metallurgy in the *Transactions of the Institution of Mining and Metallurgy*, and of papers on processes and plant used in the fabrication of non-ferrous metals in the *Journal of the Institute of Metals*, offered to these Institutions the sum of £200 per annum for a period of seven years to be applied as follows:

(*a*) £100 per annum to be available for one or more Awards to the authors of papers on some aspect of non-ferrous extraction metallurgy;

(*b*) £100 per annum to be available for one or more Awards to the authors of papers relating to some process or plant used in the extraction or fabrication of non-ferrous metals,

contributed by persons engaged full time in industry or practice.

The Councils of the Institution of Mining and Metallurgy and of the Institute of Metals gratefully accepted this offer, and appointed a joint Adjudicating Committee. This Committee has power to make the awards on behalf of the two societies and may, at its discretion, make no award or awards of less than the money available if, in its opinion, the quality of papers submitted in any year fails to reach a suitable standard. Any sums not awarded will be carried forward to future years.

The two Councils hope that the generous offer made by Messrs. Capper Pass and Son, Ltd., will stimulate the writing of many papers of the types for which the Awards are to be made. Papers on extraction metallurgy should preferably be submitted to the Institution of Mining and Metallurgy, while those on processes and plant used in the fabrication of non-ferrous metals should preferably be offered to the Institute of Metals. Both societies are prepared to accept papers of suitable quality from non-members.

Authors should note that applications should not be addressed to the Adjudicating Committee requesting that their papers should be considered for an Award. All papers published by both societies will be examined by the Committee annually, and notices of the Awards will be published in the journals of the two societies and in the Press. The Committee will shortly consider all papers published by the two societies during 1948.

January General Meeting

The Fourth Ordinary General Meeting of the Session, held on Thursday, 20th January, 1949, at the Geological Society of London, was attended by about 65 members and visitors. The paper by Mr. Ralph Symons, *Member*, entitled 'Mining and milling antimony ore at Consolidated Murchison Goldfields, Transvaal,' was submitted for discussion and was introduced by Mr. J. B. Dennison in the absence of the author. A report of the discussion will be published in the March issue of the *Bulletin*.

Fifty-Eighth Session, 1948-49: Dates of Subsequent Meetings

The following are the dates fixed for General Meetings of the Institution during the remainder of the Session 1948–49:

17th March, 1949.
21st April, 1949.
19th May, 1949.

An additional General Meeting has been arranged for 16th June, 1949.

(These dates are the third Thursday of the month.)

Members from Abroad

The Council are always anxious to meet members who come to England after a long absence abroad, and ask such members to make themselves known to the Secretary when attending General Meetings of the Institution at Burlington House.

Institution Awards

'The Consolidated Gold Fields of South Africa, Limited', Gold Medal and Premium of Forty Guineas are awarded jointly or separately by the Council of the Institution for the paper or papers of highest merit contributed to the *Transactions* during each Session, or for researches on the occurrence, mining, or treatment of minerals. The Council shall be satisfied that the papers or researches are of sufficient merit to justify the award.

Two prizes of Ten Guineas each are offered annually for papers contributed to the *Transactions* by Students of the Institution, provided that the papers are, in the opinion of the Council, of sufficient merit to justify an award.

Papers for the consideration of the Publications Committee should be sent to the Secretary, if possible in duplicate, and should be prefaced by a summary of contents. It is understood that all papers submitted are original communications unless distinctly stated to be otherwise, in which event exact reference should be made to any previous publication. Figures illustrating papers should be drawn in ink, suitable for direct reproduction in a reduced size, and lettering on drawings should be in ordinary pencil. If there are photographic illustrations, prints on glossy paper should be sent; it is not necessary to send negatives.

Candidates for Admission

The Council welcome communications to assist them in deciding whether the qualifications of candidates for admission into the Institution fulfil the requirements of the By-Laws. The application forms of candidates for Membership or Associateship will be open for inspection at the office of the Institution for a period of at least two months from the date of the Bulletin *in which their applications are announced.*

The following have applied for transfer since 13th January, 1949:

To Membership—

Henry Thomas James Edward Barker (*Selukwe, Southern Rhodesia*).

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Walter Charles Hellyer (*Harrow, Middlesex*).

John Edward Howse Keylock (*Jos, Northern Nigeria*).

Russell Pascoe (*Prestea, Gold Coast*).

INSTITUTION NOTES

Annual Dinner, 1949

The Annual Dinner of the Institution, which was not held last year, will take place on Thursday, 5th May, 1949, at the Savoy Hotel, Strand, London, W.C. 2. Full particulars will be published later.

Mond Nickel Fellowships

A number of Fellowships to the value of £650 to £850 each, are again being offered in 1949 by the Mond Nickel Fellowships Committee, to men and women of British nationality with a university degree or similar standard of education, with the object of providing additional training to make them more suitable for eventual employment in senior technical and administrative positions in British metallurgical and metal-using industries.

The Fellowships, which will normally be travelling Fellowships, will occupy one year; selected candidates will be required to undertake a programme of training in industrial establishments in the United Kingdom or elsewhere; awards for training at universities may be made in special circumstances.

Full particulars and application forms can be obtained from the Secretary, Mond Nickel Fellowships Committee, 4, Grosvenor Gardens, London, S.W. 1, to whom completed forms of application must be returned by 1st June, 1949.

Capper Pass Awards

In 1947, the Directors of Messrs. Capper Pass and Son, Ltd., Bristol, sharing the regret which had been expressed in many quarters at the dearth of papers on processes and plant used in extraction metallurgy in the *Transactions of the Institution of Mining and Metallurgy*, and of papers on processes and plant used in the fabrication of non-ferrous metals in the *Journal of the Institute of Metals*, offered to these Institutions the sum of £200 per annum for a period of seven years to be applied as follows:

(*a*) £100 per annum to be available for one or more Awards to the authors of papers on some aspect of non-ferrous extraction metallurgy;

(*b*) £100 per annum to be available for one or more Awards to the authors of papers relating to some process or plant used in the extraction or fabrication of non-ferrous metals,

contributed by persons engaged full time in industry or practice.

The Councils of the Institution of Mining and Metallurgy and of the Institute of Metals gratefully accepted this offer, and appointed a joint Adjudicating Committee. This Committee has power to make the awards on behalf of the two societies and may, at its discretion, make no award or awards of less than the money available if, in its opinion, the quality of papers submitted in any year fails to reach a suitable standard. Any sums not awarded will be carried forward to future years.

The two Councils hope that the generous offer made by Messrs. Capper Pass and Son, Ltd., will stimulate the writing of many papers of the types for which the Awards are to be made. Papers on extraction metallurgy should preferably be submitted to the Institution of Mining and Metallurgy, while those on processes and plant used in the fabrication of non-ferrous metals should preferably be offered to the Institute of Metals. Both societies are prepared to accept papers of suitable quality from non-members.

Authors should note that applications should not be addressed to the Adjudicating Committee requesting that their papers should be considered for an Award. All papers published by both societies will be examined by the Committee annually, and notices of the Awards will be published in the journals of the two societies and in the Press. The Committee will shortly consider all papers published by the two societies during 1948.

January General Meeting

The Fourth Ordinary General Meeting of the Session, held on Thursday, 20th January, 1949, at the Geological Society of London, was attended by about 65 members and visitors. The paper by Mr. Ralph Symons, *Member*, entitled 'Mining and milling antimony ore at Consolidated Murchison Goldfields, Transvaal,' was submitted for discussion and was introduced by Mr. J. B. Dennison in the absence of the author. A report of the discussion will be published in the March issue of the *Bulletin*.

Fifty-Eighth Session, 1948-49: Dates of Subsequent Meetings

The following are the dates fixed for General Meetings of the Institution during the remainder of the Session 1948–49:

17th March, 1949.
21st April, 1949.
19th May, 1949.

An additional General Meeting has been arranged for 16th June, 1949.

(These dates are the third Thursday of the month.)

Members from Abroad

The Council are always anxious to meet members who come to England after a long absence abroad, and ask such members to make themselves known to the Secretary when attending General Meetings of the Institution at Burlington House.

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Walter Charles Hellyer (*Harrow, Middlesex*).

John Edward Howse Keylock (*Jos, Northern Nigeria*).

Russell Pascoe (*Prestea, Gold Coast*).

To Studentship—

John William Davies (*Barakin Ladi, Northern Nigeria*).

James Gurney Gateward Davis (*Camborne, Cornwall*).

Brian William Hester (*Ruislip, Middlesex*).

Jeffrey Kenyon (*London*).

William James Marshall (*Wallington, Surrey*).

Tadeusz Moskwa (*London*).

Alan Robert Dundas Orr (*Glasgow*).

Peter Leslie Vaughan (*Camborne, Cornwall*).

News of Members

Members, Associates and Students are invited to supply the Secretary with personal news for publication under this heading

Mr. L. D. Airey, *Associate*, has returned to Northern Rhodesia from England.

Mr. J. M. Alexander, *Associate*, has recently been appointed to the staff of Mount Lyall Mining and Railway Co., Ltd., Queenstown, Tasmania.

Mr. R. O. F. Barry, *Associate*, has left Malaya to join the staff of the Tavoy Tin Dredging Corporation, Ltd., Burma.

Mr. C. W. F. Bond, *Associate*, has been appointed Senior Inspector of Mines in the Gold Coast.

Mr. Leslie Bristowe, *Member*, has been appointed general representative in South West Africa of the South West Africa Co., Ltd.

Mr. C. C. Cave, *Student*, is at present employed on the survey staff of Taquah & Abosso Mines, Ltd., Gold Coast.

Mr. Alan Cawley, *Associate*, is in England on leave before taking up the appointment of mining geologist, Geological Survey Dept., Nigeria.

Mr. Annan Cook, *Associate*, has left Rhodesia for the United States.

Mr. A. H. Cretch, *Member*, is returning to England on leave from Malaya.

Mr. R. Duncan, *Associate*, has returned to England from Sierra Leone and is going to Wellington, New Zealand.

Mr. J. H. Ellis, *Associate*, has returned to India from England.

Mr. J. S. Everitt, *Associate*, is on leave in England from the Kolar Gold Field.

Mr. K. A. Fern, *Associate*, has been transferred to the staff of the associated organization of Messrs. Cyanamid Products, Ltd., in South Africa.

Mr. W. A. A. Freeman, *Student*, has left Australia to join the staff of Austral Malay Tin, Ltd., in Malaya.

Mr. B. L. Gardiner, *Member*, has been awarded the C.B.E. in the New Year's Honours List.

Mr. R. C. Howard Goldsmith, *Associate*, is leaving Nigeria for England in March.

Col. P. G. J. Gueterbock, *C.B., D.S.O., M.C., T.D., D.L., J.P., Member*, has been created a K.C.B. in the New Year's Honours List.

Mr. T. Haden, *Associate*, has been awarded the British Empire Medal.

Mr. J. H. Hohnen, *Associate*, has been appointed assistant general manager of New Guinea Goldfields, Ltd.

Mr. J. Hunter, *Associate*, has returned to England from the Gold Coast.

Mr. F. C. Jackson, *Associate*, has left the Union and has settled at Lusaka, Northern Rhodesia.

Mr. H. D. M. Jager, *Associate*, has relinquished his position of manager of the Orissa Minerals Development Co.'s manganese and iron mines at Barra Jamda and expects to return to the United Kingdom early in March.

Mr. A. M. Khan, *Student*, has joined the Geological Survey of Pakistan as a geologist.

Mr. D. Litt, *Student*, is returning to England from Burma.

Mr. G. B. Mackenzie, *Associate*, has returned to Sierra Leone.

Mr. Kenneth Mackenzie, *Associate*, has taken up a post with Gold Coast Banket Areas, Ltd., Aboso.

Mr. J. C. McManus, *Student*, has left Canada for an appointment in Venezuela.

Mr. D. M. Morgan, *Student*, has arrived in England on leave from the Kolar Gold Field.

Mr. P. I. A. Narayanan, *Associate*, has taken up the post of Assistant Director (ore-dressing) of the National Metallurgical Laboratory, India.

Mr. T. R. H. NELSON, *Associate*, is now Resident Engineer with Mianrai Teoranta, Co. Tipperary, Eire.

Mr. A. L. PARMA, *Associate*, has left England to take up an appointment with Frontino Gold Mines, Ltd., Colombia.

Mr. J. PENHALE, *Associate*, has arrived in England from Sierre Leone.

Mr. N. PRATT, *Associate*, has been appointed general mine manager, Endurance Tin Mining Co., N.L., Tasmania.

Mr. C. H. RICHARDS, *Member*, is returning to England from Tanganyika on a short visit.

Mr. F. G. SHARP, *Associate*, has left England on his return to Hyderabad.

Mr. H. LESLIE SWIFT, *Associate*, returned in December to Jos, Northern Nigeria.

Mr. E. H. TREGONING, *Associate*, is returning to England on leave from Burma.

Addresses Wanted

G. P. Anderson.
A. Armstrong.
D. S. Broadhurst.
J. B. Cocking.
E. Dickson.
A. I. Scott.
A. Sloss.

BOOK REVIEWS

British chemicals and their manufacturers, 1949 : the directory of the Association of British Chemical Manufacturers (Incorporated). London : The Association, 1949. 141 p.

In addition to listing some 200 member firms of the Association of British Chemical Manufacturers, the directory includes a 67-page list of chemicals with a key to their manufacturers and a list of proprietary and trade names. It forms a most useful reference book for any buyer of chemical products, who, upon application to the Association, can obtain a copy free of charge.

History of basic metals price control in World War II. By R. F. CAMPBELL. London : Geoffrey Cumberlege, 1948. (Columbia : University Press.) 264 p., demy 8vo. 18s.

This book is of interest to the metal economist. It is an authoritative account of the war-time application of price control by the U.S. Government to the vast and complex American metal industry for the purpose of increasing production.

The impact on the U.S.A. of the recent war created a sudden and immense demand for production. The rate of output of its industry at that time was far too low to meet this call and had to be built up. Not unnaturally a strong inflationary tendency at once appeared threatening to disrupt industry and retard—possibly prevent—the development to the full of its productive capacity.

The U.S. Government was alive to this danger and, to meet it, imposed on industry a number of controls. The most effective of these, and the one on which the Administration put increasing reliance, was the control over prices.

Dr. Campbell confines his study to the price control of metals and selects for close examination those he terms ' basic,' namely iron and steel, aluminium, copper, lead and zinc. He states clearly the arguments for and against the imposition of price controls and traces the changes in the form of such controls from the ' gentlemen's ' agreements of 1940 between industry and Government to the statutory measures taken during the next few years. Of much interest are the chapters devoted to the development and operation of the Premium Price Plan which was introduced in early 1942. This Plan allowed, in effect, producers of copper, lead and zinc, under certain conditions, to receive higher prices than the basic ones fixed by the Administration.

The author served during the war as an economist on the staff of the Government's Office of Price Administration and the book reveals his very real grasp of the many complex problems that Office had to face. Dr. Campbell's study is clear, reasonably concise and admirably objective.

To readers in this country the numerous references to Government publications are probably of little practical value but this in no way impairs the sense or detracts from the interest of the text. Of particular value are the excellent summaries appended to each chapter. The book concludes with some useful and well-compiled statistical tables and a good index. The only note of criticism that may be sounded is the rather free use of initials to denote branches of the Administration and certain important regulations. This at first is a little confusing but no doubt to the American reader, to whom the book is directed, it is quite clear.

A. R. O. WILLIAMS.

OBITUARY

Cecil Alfred Burne died on 23rd September, 1948, at the age of 71. After four years as an articled pupil at South Hetton Coal Co., Durham, he was appointed surveyor and assistant manager of Langley Park colliery, Durham, in 1897, and in 1899 left for Chile to take up the position of surveyor and assayer to Copiapó Mining Co. at Atacama. In 1903 he joined the Velardena Mining and Smelting Co. at Durango, Mexico, as engineer, and after six months transferred to the post of assistant to the general superintendent of Mexican Coal and Coke Co., Coahuila. From 1905 to 1907 he was employed by the San Carlos Copper Co. of Mexico, and then for two years mined on his own account at San Nicolas, Tamaulipas. He held the position of superintendent at San Gregorio mine, Sta. Eulalia, Chihuahua, and subsequently of New Sabinas Coal Co., and from 1911 to 1913 was again mining on his own account at San Nicolas.

Mr. Burne went to Portugal in 1913 to take over the managership of Anglo Portuguese Tin Co., at Belmonte, Beira Baixa, and in the following year was appointed manager of Asturiana Mines, Ltd., in Spain. He held this position for eight years and was then occupied on examination work in Portugal and Mexico for ten months before going to Sweden as manager of Lake Copper Proprietary Co. He returned to Spain and Portugal after two years and was manager and consultant to various mines there until 1932. From 1932 to 1941 he held the position of manager of San Finx Tin Mines (1933), Ltd., and consultant to Messrs. E. B. Ridsdel & Co., London. He remained in Spain until 1945, when he returned to England.

Mr. Burne was elected to Membership of the Institution in 1925.

Frederick Aubrey Grantham Maxwell died on 26th December, 1948, at the age of 74. He was born in Freemont, Nebraska, and was educated in Denver before entering the Colorado School of Mines, where he graduated E.M. in 1895. He went to South Africa in the same year and was employed as draughtsman to Eastleigh Gold Mining Co. at Klerksdorp, and a few months later as assayer and cyanide manager to New Ariston Gold Mining Co. From 1896 to 1898 he was cyanide manager to Porges Randfontein Gold Mining Co., Ltd., and held a similar position in 1899 at South Randfontein Gold Mining Co., Ltd., followed by a period of three years with Robinson Deep Gold Mining Co., Ltd., Johannesburg. In 1904 he was appointed metallurgist to Randfontein Estates Gold Mining Co. (Witwatersrand), Ltd., and after twelve years, in 1916, joined the General Mining and Finance Corporation, Ltd., as metallurgist, a position which he held until his death.

Mr. Maxwell was elected to Membership of the Institution in 1905.

The Council regret to report the death of **Henry Ewer Jones**, *Member*, on 14th December, 1948; **Reginald John Lemmon**, *Member*, on 26th January, 1948; **William John Smith**, *Member*, on 16th December, 1948; and **Frederick Harold Williams**, *Member*, on 18th January, 1948. Obituary notices will be published in a later issue of the *Bulletin*.

ADDITIONS TO JOINT LIBRARY OF THE INSTITUTION AND THE INSTITUTION OF MINING ENGINEERS

*Books (excluding periodicals and works marked *) may be borrowed by members personally or by post from the Librarian, 424, Salisbury House, London, E.C. 2.*

Books and Pamphlets:

*ASSOCIATION OF BRITISH CHEMICAL MANUFACTURERS. *British chemicals and their manufacturers. The directory of . . .* London: The Association, 1949. 141 p.

BRITISH OXYGEN CO. LTD. *Oxygen: its potentialities in iron and steel production.* London: The Company, 1948. 51 p. biblio. (*Presented by the Company.*)

CAMPBELL, Robert F. *The history of basic metals price control in world war* 2. N.Y.: Columbia University Press, 1948. 263 p., biblio. 18s.

EVRARD, René, and DESEY, Armand. *Histoire de l'usine des Vennes suivie de considerations sur les fontes anciennes.* Liège: Editions Solédi, 1948. 381 p., illus., diagrs., biblio.

FERMOR, Lewis L. *Thomas Henry Holland*, 1868–1947. Reprint from *Obituary Notices of Fellows of the Royal Society*, vol. 6, Nov. 1948, pp. 83–114. (*Presented by the Author.*)

INSTITUTE OF PHYSICS. *The measurement of stress and strain in solids.* London: The Institute, 1948. 114 p., illus., diagrs. biblio. (*Presented by the Institute.*)

INSTITUTO GEOLOGICO Y MINERO DE ESPANA. *Mapa geologico de Espana. Explicacio la hoja no.* 135 *—Sedano.* Madrid: El Instituto, 1946. 34 p., illus., maps, biblio.

INSTITUTO GEOLOGICO Y MINERO DE ESPANA. *Mapa geologico de Espana. Explicacion de la hoja no.* 243*—Calahorra.* Madrid: El Instituto, 1947. 36 p., illus., maps, biblio.

INSTITUTO GEOLOGICO Y MINERO DE ESPANA. *Mapa geologico de Espana. Explicacion de la hoja no.* 391*—Iqualada.* Madrid: El Instituto, 1947. 112 p., illus., maps, biblio.

MINWORTH METALS LTD. *Technical data on ferro alloys and special metals.* Birmingham: The Company, 1948. 29 p., illus.

ROSEN, George. *The history of miners' diseases. A medical and social interpretation.* N.Y.: Schuman, 1943. 490 p., illus. $8.50.

WARK, Ian W. *Australian research on the theory of flotation.* Presidential address to section B, chemistry, Australian and New Zealand Association for the Advancement of Science. Adelaide: The Association, 1946. 29 p., biblio. (*Presented by the Author.*)

Government Publications:

BRITISH GUIANA, DEPT. OF LANDS AND MINES. *Reports . . for the years* 1940–46 (inclusive). Georgetown: Govt. Printer, 1941–48. 11; 9; 6; 9; 17; 17; 30 p., illus., maps.

COLORADO, MINERAL RESOURCES BOARD. *Mineral resources of Colorado.* Prepared under the supervision of John W. Vanderwilt. Denver: The Board, 1947. 547 p., maps, diagrs., tabs. $2.50.

CZECHOSLOVAKIA, GEOLOGICAL SERVICE. *'Vestnik' du Service Geologique de la Republique Tchécoslovaque, vol.* 23, 1948. Prague: Geological Service, 1948. 320 p., illus.

GT. BRITAIN, GEOLOGICAL SURVEY OFFICE. *Geology of Southport and Formby*, by D. A. Wray and F. Wolverson Cope. (Memoir, explanation of 1 inch sheets 74 and 83, new series.) London: H.M.S.O., 1948. 54 p., illus., maps, biblio. 1s. 3d.

NEW ZEALAND, DEPT. OF MINES. *Mines statement (for the year ended 31st Dec.* 1947). Wellington: Govt. Printer, 1948. 75 p.

NEW ZEALAND, GEOLOGICAL SURVEY. *Reports . . . for the years* 1939–40 *to* 1945–6 (*inclusive*). Wellington: The Survey, 1940–8. 15; 12; 2; 2; 2; 2 p.

*U.S.A., Bureau of Mines. *Minerals yearbook*, 1946. Washington, D.C.: Govt. Printing Office, 1948. 1629 p. $3.75.

Western Australia, Dept. of Mines. *Report . . . for the year* 1946. Perth: Govt. Printer, 1948. 209 p., illus., maps.

Proceedings and Reports:

American Institute of Mining and Metallurgical Engineers. *Transactions, vol.* 172, *iron and steel division*, 1947. N.Y.: The Institute, 1948. 668 p., illus., biblio.

Gold Coast Chamber of Mines. 21*st annual report*, 1*st April* 1947 *to* 31*st March* 1948. Tarkwa: The Chamber, 1948. 39 p.

Transvaal Chamber of Mines. 54*th annual report, year* 1943. J'burg: The Chamber, 1948. 153 p.

Maps:

Calahorra. Geological map of Spain sheet no. 243 (and explanation). Scale: 1 : 50,000. Madrid: Institute Geologico y Minero de Espana, 1947.

Camsell River map-area, Northwest Territories. Geological survey, preliminary map 48–19 (and report). Scale: 1 in. = 2 ml. Ottawa: Dept. of Mines and Resources, 1948.

Coventry. Geological survey of England and Wales, sheet 169 (drift). Scale: 1 in. = 1 ml. Southampton: Ordnance Survey Office, 1948. 2s. 6d.

Duparquet sheet, Abitibi and Temiscamingue counties, Quebec. Geological survey, map 281A. Scale: 1 in. = 1 ml. Canada: Dept. of Mines, 1933.

England and Wales, geological map. Prepared by the Geological survey. Scale: 1 in. = 10 ml. Southampton: Ordnance Survey Office, 1948. 12s. 6d. (paper), 15s. (mounted and folded).

Ermelo coal field, south-eastern portion. Geological survey of the Union of South Africa, to accompany sheet 64 (and explanation). Scale: 1 : 75,000. Pretoria: Govt. Printer, 1947.

Ermelo, Transvaal. Geological survey of the Union of South Africa, sheet 64 (and explanation). Scale: 1 : 125,000. Pretoria: Govt. Printer, 1947.

Exeter. Geological survey of England and Wales, sheet 325. Scale: 1 in. = 1 ml. Southampton: Ordnance Survey Office, 1948. 2s. 6d.

Henley-on-Thames. Geological survey of England and Wales, sheet 254 (drift). Scale: 1 in. = 1 ml. Southampton: Ordnance Survey Office, 1948. 2s. 6d.

Igualada. Geological map of Spain, sheet no. 391 (and explanation). Scale: 1: 50,000. Madrid: Instituto Geologico y Minero de Espana, 1947.

Margaree and Cheticamp map-areas, Nova Scotia. Geological survey, preliminary maps 48–11A and 48–11B (and report). Scale: 2 in. = 1 ml. Ottawa: Dept. of Mines and Resources, 1948.

Mexico: carta geologica de la parte septentrional de la Republica Mexicana. Philip B. King. Scale: 1 in. = 39 ml. Cartas geologicas y mineras de la Republica Mexicana no. 3. Mexico, D.F.: Instituto de Geologia, Geofisica y Geodesia, Oct. 1947.

Michaud township, district of Cochrane, Ontario. Geological map 1947–3. Scale: 1 in. = 1,000 ft. Ontario: Dept. of Mines, 1947.

Opasatika sheet, Temiscamingue county, Quebec. Geological survey, map 240A. Scale: 1 in. = 1 ml. Canada: Dept. of Mines, 1929.

Pierres Greys Lakes map-area, Alberta. Geological survey, preliminary map 48–7 (and report). Scale: 2 in. = 1 ml. Ottawa: Dept. of Mines and Resources, 1948.

Ranji Lake map-area, Northwest territories. Geological survey, preliminary map 48–10 (and report). Scale: 2 in. = 1 ml. Ottawa: Dept. of Mines and Resources, 1948.

Rochdale. Geological survey of England and Wales, sheets 76 (drift and solid). Scale: 1 in. = 1 ml. Southampton: Ordnance Survey Office, 1948. 2s. 6d. ea.

Scotland and the north of England, geological map. Prepared by the Geological Survey. Scale: 1 in. = 10 ml. Southampton: Ordnance Survey Office, 1948. 12s. 6d. (paper), 15s. (mounted and folded).

Sedano. Geological map of Spain, sheet no. 135 (and explanation). Scale: 1 : 50,000. Madrid: Instituto Geologico y Minero de Espana, 1946.

Tavistock. Geological survey of England and Wales, sheet 337 (drift). Scale: 1 in. = 1 ml. Southampton: Ordnance Survey Office, 1948. 2s. 6d.

Whitehaven. Geological Survey of England and Wales, sheet 28 (drift). Scale: 1 in. = 1 ml. Southampton: Ordnance Survey Office, 1948. 2s. 6d.

INDEX OF RECENT ARTICLES

NOTE.—All Articles indexed are available for reference in the Library of the Institution. It is regretted, however, that unbound periodicals cannot be lent.

ANALYSIS AND CHEMISTRY.

ANALYSIS — PHOSPHATES. — Determination of phosphorus pentoxide in phosphate rock. James L. Kassner and others.—*Analyt. Chem.*, Easton, Pa., Vol. 20, Nov. 1948, pp. 1052–55, tabs., biblio. 50 cents.

ANALYSIS — PHOSPHATES. — Photometric analysis of phosphate rock. Charles J. Barton.—*Analyt. Chem.*, Easton, Pa., Vol. 20, Nov. 1948, pp. 1068–73, tabs., biblio. 50 cents.

ANALYSIS—RARE EARTHS.—Chemistry of thorium; quantitative estimation of thorium by a titrimetric iodate procedure. Therald Moeller and Nancy Downs Fritz.—*Analyt. Chem.*, Easton, Pa., Vol. 20, Nov. 1948, pp. 1055–58, tabs., biblio. 50 cents.

ANALYSIS — THORIUM. — Chemistry of thorium; quantitative estimation of thorium by precipitation with radioactive pyrophosphate. Therald Moeller and George K. Schweitzer.—*Analyt. Chem.*, Easton, Pa., Vol. 20, Dec. 1948, pp. 1201–4, tabs., biblio. 50 cents.

ANALYSIS — URANIUM. — Determination of rare earth elements and yttrium in uranium compounds. H. G. Short and W. L. Dutton.—*Analyt. Chem.*, Easton, Pa., Vol. 20, Nov. 1948, pp. 1073–76, tabs., biblio. 50 cents.

ANALYSIS — URANIUM. — Spectrographic determination of rare earth elements in uranium compounds. Robert O. Hirt and Norman H. Nachtrieb.—*Analyt. Chem.*, Easton, Pa., Vol. 20, Nov. 1948, pp. 1077–78, tabs. 50 cents.

ECONOMICS.

COSTS—SHAFT SINKING.—Shaft sinking costs at Viking, Yellowknife. (N.W.T.) Norman W. Byrne.—*Canad. Min. J.*, Gardenvale, P.Q., Vol. 69, Nov. 1948, pp. 58–61. 50 cents.

ECONOMICS—GOLD.—The outlook for gold in an inflationary era. Robert W. Bachelor. —*Min. J.*, Lond., Vol. 231, Dec. 18 1948, pp. 910–3. 8d.

ECONOMICS—*continued.*

LABOUR — WAGES — TRANSVAAL. — Conditions in the gold industry. The Gold Producers' Committee of the Transvaal Chamber of Mines on wage rates, hours of work, etc., of employees.—*S. Afr. Min. Engng. J.*, J'burg, Vol. 59, Pt. 2, 1948; Part 4—Nov. 27, pp. 355–7; Part 5—Dec. 4, pp. 391–3, tabs.; Part 6 (*concluded*)—Dec. 11, pp. 423–8, tabs. 6d. each.

MANAGEMENT. — Personnel management and works managers. A. K. Rice.—*Engineer*, Lond., Vol. 186, Dec. 31 1948, pp. 667–9. 1s. 6d.

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Subject to revision.] [*A Paper published on* 10*th February*, 1949, *to be submitted for discussion at a Meeting of the Institution of Mining and Metallurgy, to be held in the Apartments of the Geological Society of London, Burlington House, Piccadilly, London, W.* 1, *on* **Thursday, 17th February, 1949,** *at* 5 *o'clock p.m.*

Ground Control—Theory and Practice*

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IN 1937 the author published a paper entitled ' Theory and practice of ground control (the Kolar Gold Field) ' (1)‡. In the years that have elapsed since then further knowledge and experience have been gained, in the light of which the theory there propounded and its applicability to mining practice have been expanded. In the present paper the theory of rock pressure, originally produced in the earlier paper, has been brought up to date and from the conclusions reached the theoretical and practical aspects of ground control have been deduced.

Ground control may be defined as the control of the movements of the rock surrounding mining excavations, with the object that those movements shall neither hinder the operations nor endanger the men employed. Control is accomplished not only by the use of suitable supports, but, more important, by the proper design and sequence of mining operations.

Before this subject can be discussed it is necessary to have a clear view of the nature of the stresses likely to be encountered in the neighbourhood of mining operations, and the behaviour of rock under such stresses. The paper is therefore divided into two parts—Rock Pressure, and Ground Control.

PART I—ROCK PRESSURE

Since, as will be shown, the state of strain existing in a rock mass at depth is to some extent dependent on the general behaviour of rock under stress, it is necessary to consider the latter subject first.

THE BEHAVIOUR OF ROCK UNDER STRESS

In a number of articles (2) Dr. D. W. Phillips, of the National Coal Board, gives the results of an extensive series of experiments on the behaviour of rock specimens subjected to compressive,

*Paper received on 9th November, 1948.

†Consulting engineer.

‡Figures in parentheses refer to the bibliography given at the end of the paper.

bending, and torsional stresses. The results of the experiments, and the interpretation of them which he gives in these articles, are of great importance to the student of rock pressure and its effects. It is shown that many of the generally-held conceptions of the manner in which rock yields and fails under stress are false. Study of this work brings out the following conclusions :

(1) The behaviour of rocks under stress is not analogous to the behaviour of metals under similar stress—that is, strain is not proportional to stress up to a certain yield point, above which the material becomes plastic ; on the contrary, rocks exhibit a certain amount of plasticity from the beginning. In other words strains induced by stresses, no matter how small, increase with time.

(2) When a rock specimen is subjected to stress, a strain is produced. If the stress continues to be applied, the strain increases with time. The rate of increase becomes less as time progresses, so that the strain tends towards a certain limit. If the loading is at any time removed, part of the strain is recovered at once, but only part, and the specimen is found to have taken a 'set'. During a period of time immediately following the removal of the load the specimen 'creeps' back towards its original dimension, which dimension is, however, never reached. The specimen is thus found to have taken a 'permanent set'.

(3) The effect of these factors is that the so-called elastic 'constants' are not constant, but vary with the applied stress.

(4) The creep of a specimen after loading is called 'time-strain'. It is not true plastic flow, because normally it has a finite value.

(5) Although, as already stated, the time-strain or creep decreases with time, tending towards a certain limit, this is only true up to a certain value of the stress applied. If that stress is exceeded the time-strain progresses until failure occurs. Such failures may take place many days after the original application of stress. In one experiment time-strain was continuing after five months under load.

(6) The time effect is not always regular. Curves illustrating strain against time are usually of the form shown at A in Fig. 1. In some instances, however, and particularly when the specimen is proceeding to failure, the rate of increase of time-strain gradually decreases and then suddenly increases again. This may occur more than once, as shown at B. Therefore, the fact that the time-strain is decreasing is no criterion that equilibrium is being approached. (Incidentally this is borne out by experience underground—that is, the fact that a place 'goes quiet' is no criterion that a rockburst is not going to occur.)

(7) The general form of the time-strain curves was the same for all the rocks tested.

(8) It would appear from some experiments that failures which took place several days after the sudden application of a stress would not have occurred had the load been applied gradually.

Although Dr. Phillips's experiments were all carried out on rocks of the Coal Measures there is little doubt that the conclusions gained are more widely applicable. This is substantiated by the fact that experiments in building research have shown that concrete under stress exhibits the same phenomenon of time-strain or creep as do the Coal Measure rocks, although to a lesser extent (3). It is

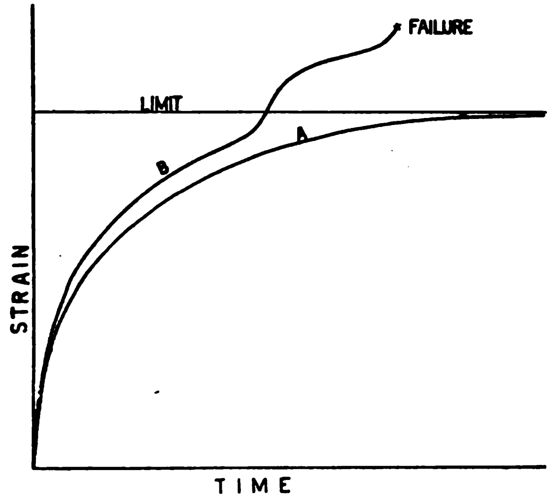

Fig. 1.

probable that time-strain in the harder rocks is also considerably less than in the softer types tested.

The Nature of Rock Pressure

It is not possible to calculate mathematically that the stress in any given rock mass at any given depth will be of a particular nature or value. So many variables affect the issue, most of them unknown and incalculable, that no definite formula can be proved applicable. However, by a process of reasoning a state of strain can be deduced which probably approximates to the actual conditions in the majority of cases.

In the earlier paper (1) the assumption was made that, in a rock mass at depth, in addition to the vertically-acting pressure due to the weight of superincumbent rock, there are horizontal components of stress of considerable magnitude. This assumption was based on observations of the effects of pressure underground in the mines of the Kolar Gold Field—for instance, from the magnitude of the forces exerted in the closing in of the walls of vertical stopes. It can now be shown, not only that horizontal stresses are universally present (and are not merely a special feature of the Kolar Gold

Field), but that they are of greater magnitude than was originally surmised.

(1) Consider an undisturbed rock mass at sufficient depth to be free from surface effects—such as weathering, contour of the ground, etc. In such a mass the vertically-acting pressure at any point will always be equal to the weight of superincumbent rock. Through the mass there will generally be a number of fissures—either partly open and water-bearing or cemented up with minerals such as calcite. The water in the open fissures will be at a hydrostatic pressure due to the depth. If the density of the rock is 2½, the static water pressure will be 0·4 times the vertically-acting pressure in the rock. This water pressure will act on the rock hydrostatically, so that horizontal components of stress are set up in the rock equal to 0·4 times the vertical component. Those fissures which are now cemented up were not always in that condition—at one time they were filled with liquid. Throughout the period of cementation the hydraulic stress was present, and when the process was complete the hydraulic stress remained.

(2) A particle of rock at depth, acted on by the weight of rock above, tends to expand sideways. As it is prevented from doing so by adjacent particles, additional lateral (horizontal) stresses are induced. They are equal to the difference between the vertical stress and the original horizontal stress due to the water in the fissures, multiplied by Poisson's Number* less one. This number for hard rocks under stress may be taken as lying between 4 and 5, which brings the total horizontal stress due to these causes up to about 0·7 of the vertical stress.

(3) In addition to the induced stress caused by the elastic properties of the rock there is a further induced stress due to pseudo-plastic deformation or time-strain. The value of this is incalculable at the present state of our knowledge, but there is little doubt that it is appreciable. Its effect will be still further to reduce the difference between the horizontal and vertical components of stress.

(4) In many cases the action of hydrothermal solutions or even moving ground-water promotes hydration of pre-existing rock-forming minerals. This process is accompanied by increase in volume, which, if unrelieved, will result in internal stresses within the rock, again increasing the horizontal stresses.

(5) In all rock masses which were once buried beneath an overburden since removed, the state of stress as deduced in the reasoning already given was once a function of the original depth. After denudation of the cover the vertical stress is, of course, reduced in proportion, but the horizontal stresses have not an equal opportunity for full relief and are therefore likely eventually to settle down to a value intermediate to that which might be expected from the original and present depths.

*Poisson's Number is the reciprocal of Poisson's Ratio (*sigma*), a figure dependent on the elastic constants of the material in question.

(6) Should some geological upheaval have increased the horizontal stresses to a greater value than the vertical stress, the elastic and pseudo-plastic deformations will then tend to equalize the stresses—that is, they will reduce the horizontal stress towards the value of the vertical stress, the surface of the ground expanding upwards and affording partial relief to the excessive horizontal stress.

Thus, there is strong reason to infer that, in a rock mass at depth and as yet undisturbed by mining operations, the horizontal stresses tend towards equality with the vertical stress and that in the majority of cases they will not be far removed from it in value.

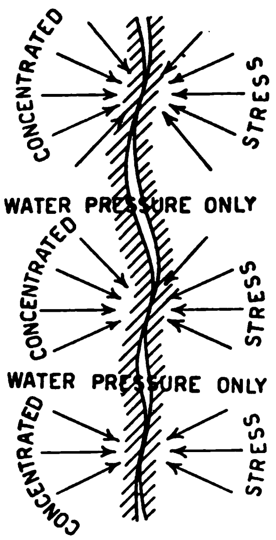

Fig. 2.

If, therefore, it is assumed that the two horizontal components of stress are equal to the vertical stress, the error will not be great. This gives a very simple basis to work from.

A system of stresses can be resolved into three components acting in any chosen directions mutually at right angles. In considering development work the vertical and strike directions can be used as components, with the third at right angles to them ; but in stoping it is more convenient to take components normal to the reef plane, on the dip and on the strike. In either case, as has already been shown, before mining commences the three components will generally approach equality.

There are three conditions, however, which may cause stresses underground which are widely differing in the three directions : (*a*), when there is some residual of stress resulting from geological movements ; (*b*), in ground adjacent to high mountains, deep canyons, or on coasts adjacent to deep seas ; and (*c*), when the ground is cut by an open fissure.

In condition (*c*) horizontal stresses both greater and less than the vertical component may locally be expected. This is because such fissures are not open from top to bottom—that is, the two sides of them are in contact in some areas and apart in others. If the fissure is filled with water the horizontal stress across the open parts must be equal to the water pressure at the point in question, whatever that may be. Therefore, any balance of stress, such as will occur when the water pressure is less than the rock pressure, will have to be carried through the areas where the sides are in contact (see Fig. 2).

In this context it should be noted that if the fissure is drained of water by mining operations the stress across the open parts will be reduced to zero, while that across the points of contact will be further increased to make up for it.

These special conditions are comparatively rare, and the majority of mines—at any rate the majority of *deep* mines—will be working in ground where the three components of stress approach equality.

The Theory of Rock Pressure

A theory of ring stress based on the assumption of equal components of stress was given in the author's previous paper (1). This theory has stood the test of time and is now generally accepted. It is briefly repeated here.

If, in a rock mass of infinite dimension subjected to equal components of stress, a spherical hole is made, the nature and intensity of the stress in the rock round the hole is radically altered. If the original stresses were Q, at a point in the rock distant r from the centre of the hole, the new stresses would be as shown in the following formulae* :

$$\text{Tangential stress} = \left(1+\frac{a^3}{2r^3}\right) Q$$

$$\text{Radial stress} = \left(1-\frac{a^3}{r^3}\right) Q$$

where a is the radius of the hole.

From these equations the following facts appear. At the side of the hole, in what may be termed the ' skin rock ' (where $r=a$), the total stress acting tangentially is $1\frac{1}{2}Q$, while that acting radially is zero ; at a distance into the rock equal to the radius of the hole ($r=2a$) the tangential stress falls to $1\frac{1}{16}Q$ and the radial stress rises to $\frac{7}{8}Q$; and at an infinite distance the stress in all directions remains unaltered at Q.

For the case of a cylindrical hole in similar circumstances, the formulae are :

$$\text{Tangential stress} = \left(1+\frac{a^2}{r^2}\right) Q$$

$$\text{Longitudinal stress} = Q$$

$$\text{Radial stress} = \left(1-\frac{a^2}{r^2}\right) Q$$

*These equations were originally produced by Mr. J. S. Jones, M.A., B.Sc., quondam head of Messrs. John Taylor and Sons' physical research department on the Kolar Gold Field (4).

Thus in the skin-rock ($r=a$) the tangential stress is $2Q$, the longitudinal stress is Q, and the radial stress is zero; while at a distance into the rock equal to the radius of the hole ($r=2a$) the tangential stress is $1\frac{1}{4}Q$, the longitudinal stress is Q, and the radial stress $\frac{3}{4}Q$.

The equations show that surrounding holes of spherical or cylindrical shape there is a zone or envelope of strain in the tangential direction greater than that pre-existing in the ground, that the extra stress is greatest in the immediate skin rock, and that it dies away fairly rapidly as the distance into the rock from the side of the hole increases. This extra stress has been variously called the 'ring stress' and the 'stress envelope'. It further appears that the maximum stress produced in the case of the spherical hole ($1\frac{1}{2}Q$) is less than that for the cylindrical hole ($2Q$), and also that it dies away more rapidly.

By far the most important points that emerge from a study of the equations, however, are:

(1) that when a hole is made in a rock mass the maximum intensity of stress which is caused in the surrounding rock is independent of the size of the hole; and

(2) that the larger the hole the greater the distance to which appreciable extra stresses extend into the rock—that is, the broader

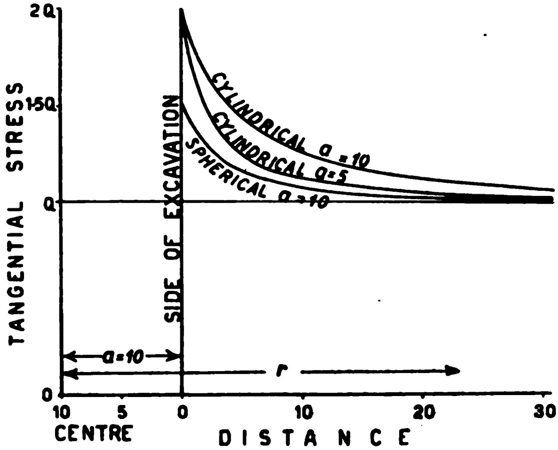

Fig. 3.

the belt of ring stress. The graphs in Fig. 3 are plotted from the above formulae and illustrate these points.

Natural Stress Relief

Mining engineers will have many objections to this theory. First, no excavations are spherical or cylindrical; secondly, the sides of excavations as blasted out are shapeless and irregular instead of being smooth; and, thirdly, those sides are often cracked or loose instead of being solid. How then, they will ask, can such cracked, irregular skin-rock of (for example) rectangular section contain stress amounting to double that due to the weight of rock above—a stress which at 2,500 ft. depth would amount to nearly 3 tons/sq. in. ?

They will say it is impossible, and they will be right, for such cracked, irregularly-shaped rock cannot transmit any stress at all. What happens is that, immediately on blasting, the tangential stress in the skin-rock is doubled, while the radial stress is reduced to zero. Consider a particle of rock in the immediate skin of such a newly-opened excavation. Subjected to high stress tangentially and with a newly-opened free face in front, it is free to expand

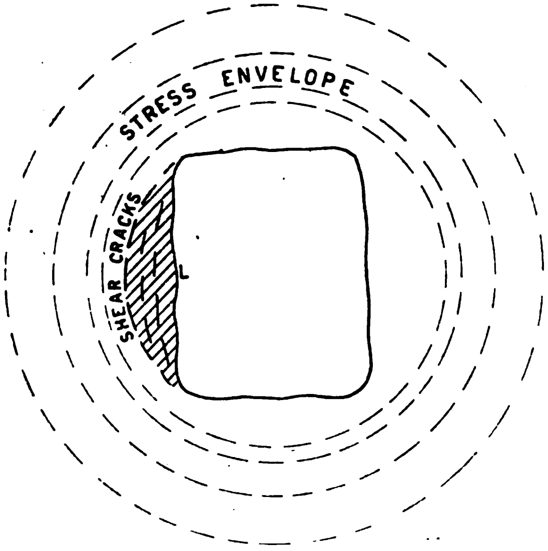

FIG. 4.

inwards. This permits a certain amount of tangential contraction, which, in turn, relieves the tangential stress. The amount of elastic relief afforded in this way depends on the difference between the two components of stress and Poisson's Ratio for the rock. Behind this particle are other particles, in which the tangential stress is less than in the skin and the radial stress is no longer zero. The difference between the two components of stress is therefore less than in the skin, and the elastic relief is also less. Thus the effect of natural stress relief dies away as the distance from the skin increases.

The action of elastic stress relief is assisted and increased by the occurrence of small shear cracks, pseudo-plastic flow (creep), or even by plastic flow at points where the stress is extra high due to irregularities in the rock or in the shape of the excavation. The strain avoided by the skin-rock in this way can only be taken up by other rock further back, so that actually the ring under maximum stress is no longer in the skin, but is removed some distance into the rock.

The total effect of natural stress relief is thus to surround all excavations with a 'ring' or 'envelope' under increased stress, between which and the skin of the excavation is a zone under decreased stress, which falls towards zero as the skin is approached.

When a tunnel is driven to a rectangular section there is often a tendency for it to 'arch out' to a circular section. This is explained as follows. Along the straight side of the drift is a lens-shaped piece of rock (L, Fig. 4) in which the stresses have been largely relieved—it has therefore expanded and is larger than it was before. This piece is held in place by its cohesion to the rock behind it, which, subjected to ring stress, is under heavier stress than before and is further compressed. Under differential stress, heavy in the stress ring and light in the lens-shaped piece, shear cracks tend to form parallel to this stress, dividing the stressed from the unstressed rock. Thus the lens of expanding rock is freed and forced out into the drift. Occasionally the action is violent.

In a homogeneous rock the tendency is for fretting to turn the excavation into the ideal cylindrical or spherical shape. Sometimes this action of arching is strongly evident and the loose rock comes away, leaving an excavation roughly circular in section; sometimes it is merely evidenced by a number of cracks and semi-loose rock, while in other cases there is no sign of it at all. It is merely a question of the magnitude of the stress in comparison with the physical characteristics of the rock.

The natural relief of stress in the skin, and the shifting of the ring under maximum stress to a zone further back, are illustrated in Fig. 5. No mathematical formulae for the value of the actual stress at any point are available (although it is probable that a solution could be found). In the figure, therefore, the curve of actual stress has only been estimated, but an idea of its true shape can be obtained from the known data. These data are:

(*a*) the stress acting in the skin is zero;

(*b*) it rises to a maximum a little way inside the rock;

(*c*) after this it tails away, asymptotic to the stress-equals-Q line;

(*d*) the two shaded areas shown must be equal, since the total stress on the ground is unaltered;

(*e*) since the theoretical peak of stress is a highly pointed one, and since this is very unlikely to be so in actuality, it is probable that the actual maximum stress will be considerably less in intensity than the theoretical peak.

The effect of natural stress relief, whether accompanied by active arching to a circular shape or not, is to surround every excavation

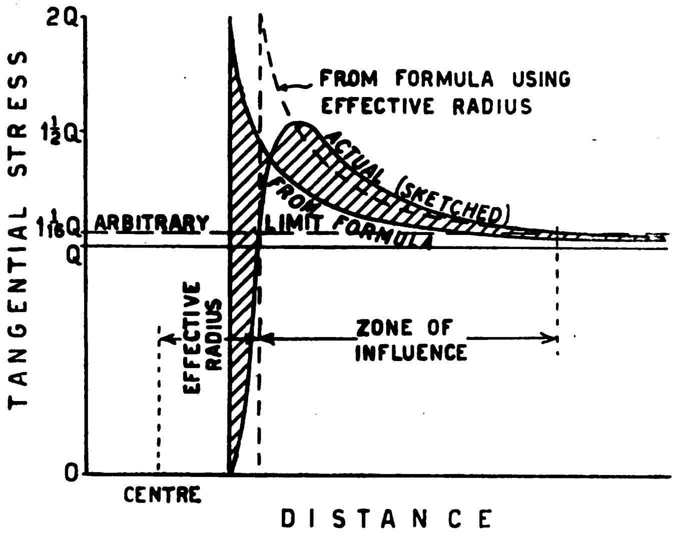

FIG. 5.

with a ring of de-stressed rock, which may or may not be loose and cracked, outside which is the zone under ring stress. The real effect then is to make each excavation effectively somewhat larger than it actually is. Granted this, then the principle of the theory of ring stress can be applied, not only to ideal cylindrical or spherical holes, but to holes of any shape whatsoever. All that is necessary is to imagine an arbitrary circular section that will include the smaller, irregularly-shaped actual excavation with its envelope of de-stressed rock, and to consider the ring stresses as surrounding this again. This imaginary excavation can be termed the ' effective excavation ', and its radius the ' effective radius '.

Mutual Effect of Adjacent Excavations

It is now possible to deduce from the theory matters of practical value to the mining engineer. Although, owing to natural stress relief, the maximum ring stress is probably less than $2Q$, while the peak of maximum stress is somewhat broader than that indicated on the theoretical curve, it will be convenient, in the absence of any formula for the actual stress curve, to use the theoretical formulae already given. These would give results with a considerable error when considering excavations which were within a few feet of each other, but at the distance at which adjacent excavations are normally kept apart the error will be trivial. The important matter is the relative values of the stresses caused in various circumstances, not their absolute value.

If some arbitrary figure is taken above which stresses are considered appreciable and below which they may be assumed to be negligible, it will be seen that each excavation is surrounded by a zone or envelope of increased stress of definite thickness (see Fig. 5), this thickness depending on the radius of the effective excavation. Suppose, for instance, that the arbitrary limit $1\frac{1}{16}Q$ be taken, then the thickness of the zone under appreciable increased stress surrounding a spherical excavation is equal to its radius, and that surrounding a cylindrical one is equal to three times its radius. These zones may be termed the 'zones of influence' of the excavation. If another excavation is made in the zone of influence of one already existing, the maximum stress in the skin of the new excavation will be a function not of the original stress in the rock but of the increased stress due to the first excavation.

For instance, consider a tunnel being driven past a large chamber, and suppose that it is at such a distance from the chamber that the ring stress in the ground is $1\frac{1}{4}Q$. Then the stress in the rock at the sides of the tunnel in the direction of the ring stress of the chamber will be given by $\left(1+\frac{a^2}{r^2}\right)1\frac{1}{4}Q$, with a maximum at $a=r$ of $2\frac{1}{2}Q$.

To take a concrete case, suppose the chamber is 36 ft. in diameter and assume that its effective radius is 24 ft. If the arbitrary limit taken is $1\frac{1}{16}Q$, the zone of influence of the chamber will extend

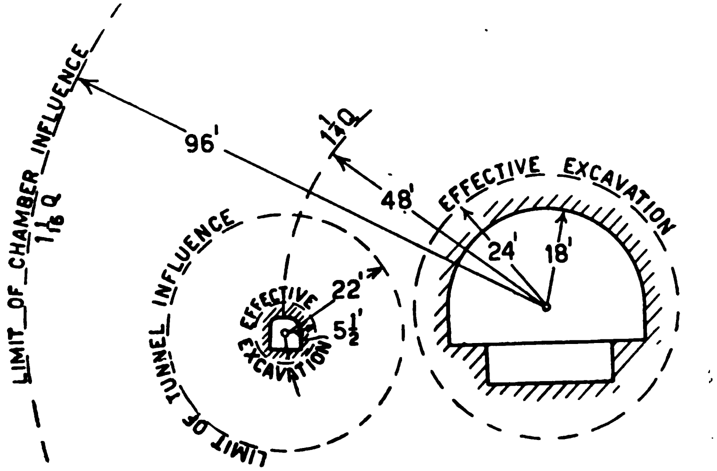

FIG. 6.

into the rock a distance equal to four times the effective radius—that is, 96 ft. from the centre of the chamber (see Fig. 6). The ring stress of $1\frac{1}{4}Q$ is met at a distance of 48 ft. from the centre of the chamber, and if a tunnel is driven at this point, say $5\frac{1}{2}$ ft. in effective radius as shown, the maximum stress in the rock of its sides will be twice $1\frac{1}{4}Q$, or $2\frac{1}{2}Q$. The most interesting point is that while the effect of the chamber is to raise the maximum stress in the neighbourhood of the tunnel by 25 per cent, the chamber is outside the zone of influence of the tunnel. The actual effect of the latter works out at $1{\cdot}03Q$ at the near side of the chamber and $1{\cdot}007Q$ at the far side.

In the driving of such a tunnel, therefore, it is far more likely that trouble with the ground will be experienced in the tunnel itself than in the chamber, but, in order to keep the effect on the chamber to a minimum, the tunnel should be kept as small as possible. It should also be lined, for, if unsupported, and if, owing to the increased stress and perhaps to some weaknesses in the rock, arching became active, the effective diameter of the drift would increase. This would probably be unnoticed by the miner, but, since the effect on the chamber would increase with the square of the effective radius, a small increase in the latter is important. Excavations in juxtaposition should, therefore, always be lined to prevent arching.

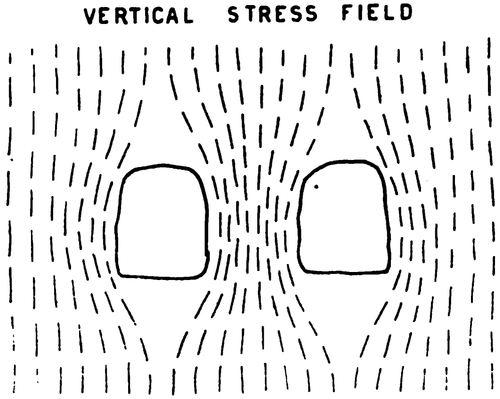

FIG. 7.

Working on similar lines, it is easy to assess the mutual effects of various other excavations. It will be sufficient to consider here three other cases :-

(1) Consider a drift being driven to hole to another one in the same straight line. The envelopes of stress embracing the ends of the two drifts will approach the hemispherical in shape. This being so the ring stress will be less in intensity and the zone of influence much less in thickness than in the case of cylindrical stress envelopes. The maximum stress will be something less than $1\frac{1}{2}Q$ and it will die away to $1\frac{1}{16}Q$ in 11 ft. from the centre of the end—say, 5 ft. into the solid rock. Therefore, when the approaching ends are more than 5 effective feet apart, their mutual influence is trivial. Suppose they are just 5 ft. apart: then in one more round the holing will be complete and the two ends will have merged into a single tunnel. This holing is therefore a perfectly innocuous one.

(2) Suppose now that the ends for some reason fail to hole and pass each other. In ground liable to rockbursts this is a dangerous state of affairs. At one moment the ends are approaching each other, the stress envelopes being of hemispherical shape. After a blast or two they are parallel excavations with cylindrical envelopes, of which the zones of influence will extend three times as far into the rock as the hemispherical ones. Thus each end will strongly influence the other.

A case like this is best visualized by drawing the drifts in cross-section and superimposing the field under vertical stress, exemplified by force lines as in a magnetic diagram (see Fig. 7). The concentration of stress in the bar of rock between the two may be many times the value of the original stress. As a result, if the

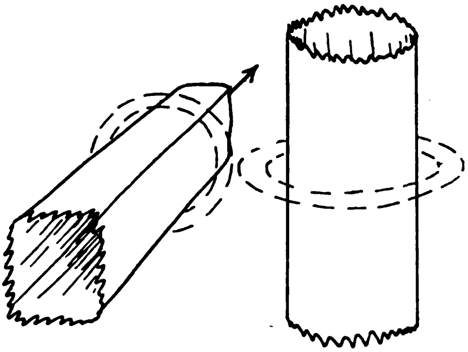

Fig. 8.

barrier is too thin it will fail, merging the two ends into one effective excavation, surrounded by a single envelope of stress.

(3) Consider finally a tunnel being driven to pass close by a vertical shaft without holing to it (see the perspective sketch in Fig. 8). It will be immediately obvious that the directions of ring stress are mutually at right angles, so that they do not interfere with each other. The drift may therefore be put out with safety much closer to the shaft than it could (say) to an excavation of similar size which was lying horizontally and parallel to it.

The Failure of Rock under Stress

When the three components of stress are equal—that is, when the stress is hydrostatic in character—there is no tendency to shear; but when the three components are unequal planes of shear are set up. Now brittle bodies subjected to compressive stress fail by shear only, so that when the stress is truly hydrostatic a body cannot fail, because there are no planes of shear. Once the stress system is unbalanced, however, as in the envelopes under stress caused by and surrounding mining excavations, planes of shear are set up, and the failure or not of the rock is dependent on the intensity of shear occasioned.

It has already been shown that the effect of driving a tunnel through a rock mass under hydrostatic stress is to unbalance the stresses, increasing them in the tangential direction and reducing them in the radial direction. Theory shows that when the three components of stress are unbalanced, so that $Q_1 > Q_2 > Q_3$, a shearing stress of $\frac{Q_1 - Q_3}{2}$ is set up on planes approximately at 45° to the direction of greatest stress Q_1[2]. Therefore, near the skin of a tunnel, shearing stresses are set up at 45° to the direction of the ring stress, which will approach in value $\frac{2Q - 0}{2}$ or Q, the original hydrostatic stress in the ground. Actually the angle of shear is not exactly 45°, being modified by certain characteristics of the rock and by any natural planes of weakness pre-existing in it.

The author has seen many examples of such failures underground. They are most obvious when the excavation has been provided with a smooth lining of concrete. Two of these examples are described here :

The first occurred in a steeply-dipping elliptical shaft at 8,000 ft. depth. The shaft measures 20 ft. by 10 ft. and had a monolithic

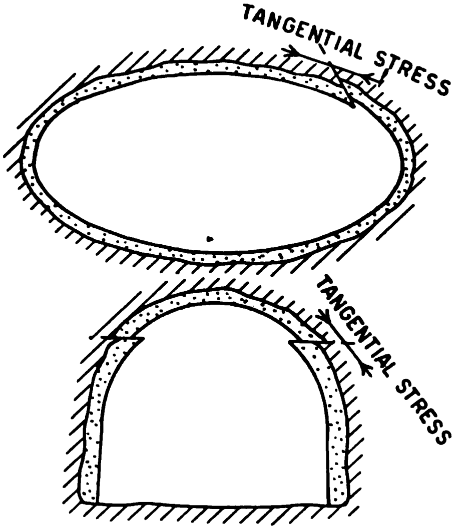

FIG. 9.

concrete lining which fractured as shown in Fig. 9 over a length of 100 ft., the overthrust being about 6 in. Incidentally, after the shear had once taken place—the action was quiet and was not noticed at the time of its occurrence—telltales showed no further movement over a period of many years.

The second example took place at 6,000 ft. depth in a shaft station, lined with a concrete arch 12 ft. wide, during a rockburst which affected a nearby stoping area. As shown in Fig. 9, fractures developed each side of the arch, on a horizontal plane of weakness caused, presumably, by some delay having occurred during the concreting of the arch.

PART II—GROUND CONTROL

The practice of ground control falls under two main headings—the support of development excavations, and the control of the walls in stoping. The practice in these two cases being essentially different, each is here treated separately.

The Support of Development Excavations

The rock through which development excavations are made is supported, when support is necessary, by a ' lining '. This term is used to differentiate between it and the supports used to control the walls in stoping. In this paper the term is used not only to denote an all-round covering as the name implies, but to part coverings also.

All development excavations can be placed in one of two classes—those on lode which will eventually be stoped and so will be subjected to the closure of the stope walls, and those which will not be affected in this way—namely, drifts on unpayable lode, drifts in the wall, crosscuts, shafts, stations, and chambers. These two classes require radically different treatment in lining.

Linings which will be Subjected to the Closure of Stope Walls

The object of a lining in this case is not always clearly understood and as a result linings are in use which are, to say the least, ill-designed. The essential duty of a lining is to keep the way open and safe for traffic in all circumstances, no matter what ground movements may occur ; it is *not* primarily to assist the stope supports in opposing closure, although if this can be achieved without sacrifice of the primary duty so much the better.

In order to allow for closure, and in those mines where rockbursts occur in order to resist shocks and live loads, the lining must be flexible. This can be achieved in a number of ways, of which the simplest is to use a rigid lining, say of timber sets, with a cushion of packing round it. By consolidation of this packing under squeeze a certain small amount of closure can be tolerated. In this context attention may be drawn to the fact that three- or four-piece sets of timber or steel were originally designed for use in flat lodes or seams, the ' weight ' which they were designed to withstand coming vertically downwards. They are ill-fitted for use in more steeply-dipping lodes, where the squeeze comes on them diagonally, a direction in which they have no strength whatever.

A more flexible but expensive lining can be made of steel rings or sets of circular or elliptical section, lagged with poles, planks, steel rods, or coarse wire-netting and surrounded by a cushion of packing. With these, closure is tolerated, first by consolidation of the packing and then by distortion of the set itself. When such a lining is used the original drift and the lining put in it should be made of such a size and shape that, after the maximum closure anticipated has occurred, there is still sufficient room for tramming or other purposes.

If the packing around a lining of circular steel sets is confined above and below by stope supports, closure of the stope walls will throw all the packing into compression. Since the packing is not a rigid body, but can flow under squeeze, the stress in it then becomes hydrostatic in character and presses on the steel set, not only opposite the hanging and foot, but all around. This throws

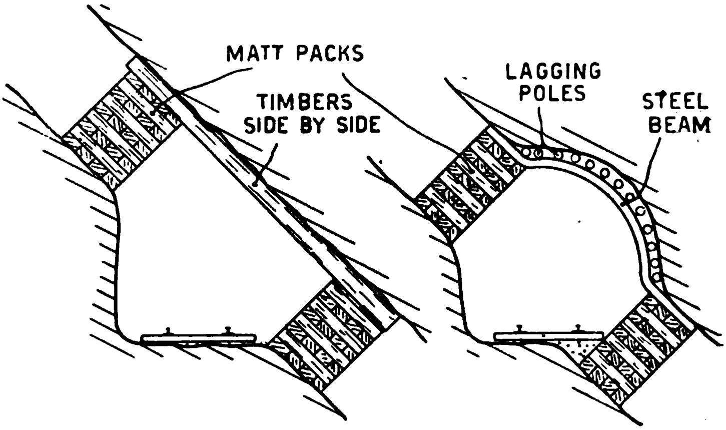

FIG. 10.

the steel set into ring compression without imposing very heavy bending stresses on it, a state in which it is immensely strong. Steel sets in this condition will remain efficient until so distorted that a re-entrant angle is set up.

Steel sets of curved shape will buckle under heavy load unless they are so designed that the section of which they are made is stiffer in the longitudinal direction than in the plane of the set. In British colliery practice steel sets are designed with their greatest strength (and stiffness) in the plane of the set, and in order to stop the buckling under excessive closure (roof sag), either elaborate bracing is used between sets or telescopic joints are fitted to the legs. If the set is designed with the weaker moment of inertia of the steel section in the plane of the set, the resistance of that set to bending stresses is, of course, less than if it were placed the other way. On the other hand there is no tendency to buckle, so that no bracing at all is required between sets. In any case the primary duty of the lining is not to resist closure, but to keep the heading open for traffic or ventilation, so that the resistance of the set to bending is a secondary consideration.

Steel sets, whether of inverted-U shape for use in flat lodes and seams, or of circular shape for other dips, can be rendered still more flexible by providing the pieces of which they are built up with overlapping instead of butt joints. Telescopic action at the joints then permits gradual reduction in size with a minimum of distortion of the steel. The resistance of the set to contraction can be controlled by regulating the friction between the sliding members at the joint by an adjustment of the nuts of the clamp bolts[7].

A great advantage of steel sets is that they can be used over and over again, re-shaping being done when necessary.

A third and very useful type of flexible lining can be made integral with the stope supports. It consists in fact of some sort of strap, which is held by the nearest stope supports above and below the drift and lies against the hanging-wall. Two examples of such a lining are shown in Fig. 10. It will be noted that, as the lining has no connection with the foot-wall, it is completely unaffected by the closure of the stope walls. When difficulty is

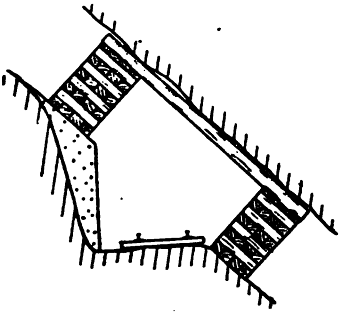

FIG. 11.

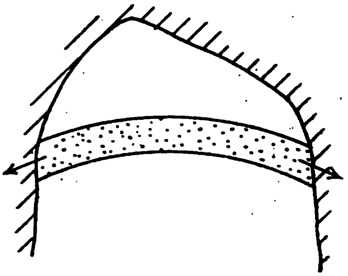

FIG. 12.

experienced in getting the two stope supports sufficiently close together owing to the fretting away of the foot-wall, the latter can be replaced by a building of waste rock masonry set in a weak mortar, or by a block of concrete (see Fig. 11).

Linings which will not be Subjected to Closure

When not subjected to closure all that is required of a lining is that it shall keep the way open and safe for traffic. Flexibility is not necessary, nor, as will appear later, is it even desirable. A further essential difference between the two classes of linings is that linings to be subjected to closure must be designed to suit the dip of the lode, whereas for those that will not, the dip of the lode does not affect the design in the slightest : nor do joints in the rock. All that the lining has to do is to prevent or restrict arching and to hold up loose ground. The problem is, therefore, a simpler one than in ways through stoping areas, and it can usually be solved by a cheaper lining.

A tendency of the back or sides of excavations to fret or arch up can be reduced by making them originally of a cross-section most suitable to contain the ring stresses. This then should be the first method of ground control in development. Since gravity greatly increases the action of arching in the back while decreasing that in the bottom the shape at the bottom is less important than the shape in the back. The most suitable sections are, therefore, the horseshoe or the inverted U. All development in ground liable to fretting in the back should therefore be driven with a rounded, and not with a flat, back, while, if trouble is experienced in the sides also, the horseshoe shape is better. When arching trouble is severe it is advisable to drive to a full circular section.

If the trouble is not cured by shaping the excavation it is then necessary to put in some sort of lining. In order to stop arching, this lining must be of a rigid nature capable of taking stress ; loose linings with packing around, although they protect travellers from falling rock, do not prevent the action of arching continuing behind the packing. Moreover, putting up a back covering and allowing the arching to continue above it is not a satisfactory solution. Not only has the covering to be cleared of debris from time to time, but the effective excavation, and therefore the zone of influence of the place, is constantly extending. For solitary excavations this has little effect, but where there is a number in juxtaposition it is a matter of importance.

There is one type of covering suitable for roofing drifts and crosscuts of widths up to 10 ft. which need not be built tight to the back. It consists of a thin arch of concrete sprung between the rock sides of the gallery. No sidewalls are required to hold up the arch and, unless the rock sides are particularly smooth and greasy, pegging is also unnecessary (see Fig. 12). By reason of its shape this arch throws stress on the ground in the direction of the arrows and this is found sufficient to stop the action of arching above it completely. It is, therefore, unnecessary to put packing on the roof.

Where the back of a large tunnel such as a main haulage needs permanent support the best section to use is a flat semi-ellipse. Mounted on sidewalls of stone, brick, or concrete a concrete roof of this shape is as good as one of semicircular section, while giving more headroom close to the sides.

In ground which is at all liable to fret, large chambers should not be designed with flat roofs. Such places have frequently ' arched up ' above the roof to double their original height, loading the roof with a great weight of debris and doubling the distance to which their zone of influence extends. When the amount of debris becomes too great it has to be cleared away and then supports are usually required to check the continuance of the action. These usually take the form of matt packs or pigsties built up from the flat roof. Such are, however, quite unsuitable as, being compressible, they yield if weight comes on them and permit the action *to continue.* Moreover, they transfer the stress to the flat roof, which is ill-designed to take it.

Flat roofs, then, should be avoided in all ground except that which it is certain will stand without any fretting at all. The best lining for a chamber is a rigid roof, arch-shape, built tight to the ground. If steel girders are to be used they can either be bent to the curve or built up to form a regular half-polygon. In the latter case the angles should be blocked as tightly to the ground as possible.

In soft ground, or in harder rock under very heavy stress—that is, in ground which is liable to fail all round—it is necessary to put in a complete lining, preferably of concrete. It should be of elliptical or circular section, so that as stress is transmitted to it by the yielding ground it is thrown into ring compression and takes the place of the smooth skin of the ideal cylindrical excavation of theory.

In using concrete for the support of mining excavations of all types the object of the engineer's design should always be to stress his concrete in compression only and not to subject it to bending stresses. Reinforcement is therefore usually unnecessary. The engineer should avoid flat surfaces and angles, and curve as many of his structures as possible. This is not so difficult nor so expensive as it may sound.

Some engineers, when they line an excavation with concrete, separate it from the ground by a cushion of packing in the mistaken idea of protecting it from stress. Behind that packing, arching is free to continue until packing and debris become so tightly jammed as to transfer the stress to the concrete. Since in the case of an arch of inverted-U shape the packing will be tightest at the bottom of the sidewalls and loose against the back, the concrete will be stressed unequally, the straight portions of the sidewalls receiving the heaviest pressure. By building concrete or brickwork tight to the ground a smaller excavation is needed for the same finished size, the construction of the lining is simpler, the cost and labour of packing is eliminated, and the lining is stressed evenly and in compression. Moreover, a gradual increase of the zone of influence due to arching is prohibited. In shaft sinking especially all these points are of the greatest importance.

From the ground-control point of view the rectangular shaft with its timber lining has nothing to recommend it. It owes its popularity solely to the facts that timber can be used for the lining, that rectangular compartments are formed with a minimum wastage of space, and that the lining can be built downwards closely following the sink. Otherwise, the straight sides of rectangular shafts are unsuitable for bad ground or rock under heavy stress, because considerable fretting can take place on the flat sides, unseen until the lagging planks are broken by the stresses transmitted through the debris.

The use of compressible supports, except in excavations which will be subjected to the squeeze of stope walls, is basically wrong. The resistance *offered by such supports* to ground movement is small at first, increasing as the support is compressed. Since the object of supports in such circumstances is to prohibit ground movement, materials which are only effective after considerable movement has taken place are therefore unsuitable. For instance, if a place has a flat roof and a support becomes necessary under the middle of the span, that support should be as rigid as possible, so that the moment the rock over it moves, the support is heavily stressed and resists the movement. Suitable supports in such cases are posts of steel, timber, or concrete ; pigsties or matt packs are not suitable. The differences in the stress distribution and the

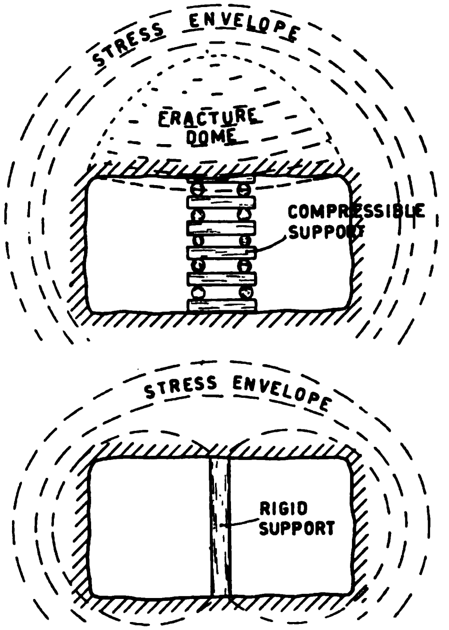

FIG. 13.

amount of arching permitted with compressible and rigid supports in such a case are illustrated in Fig. 13.

THE SUPPORT OF GROUND IN STOPING

Before considering the methods suitable for controlling the closure of stope walls it is necessary to study the manner in which that closure takes place and the effect which it has on the stresses in the ground.

The Closure of Stope Walls

So far all the excavations considered have been shafts, drifts, chambers, and similar places, but the same theories can be used equally in stoping, although the application of the theory is somewhat different owing to the very large extension of stopes in two dimensions compared with the third.

For a stoped area to arch itself out to a spheroidal shape, as postulated by Crowle in his theory of fracture doming ([5]), would mean in many cases an enormous amount of fracturing and movement, resulting in a very large effective excavation. In some cases the intensive fracture doming does take place, in others there is no sign of it, and the absence of any fracture doming on some fields threw the whole doming theory into question. Fracture doming is, however, entirely incidental; just as visible arching may or may not occur inside the stress zone enveloping a development heading, so may fracture domes form or not form inside the surrounding stress envelope.

The actual effects of opening up a new stoping area are somewhat as follows: As soon as a stope is started its walls, no longer held back by the lode matter, begin to close in. In the previous paper ([1]) this was accounted for by the expansion of the rock under relief from stress together with, in some cases, some fracture doming. The action was attributed to increasing pressure as the stope face advanced. When the stope was stopped but closure continued for a while it was attributed to increasing pressure on the immediate walls owing to rock fractures and movements behind them. It is now obvious that the inward closure of stope walls is only partly due to expansion under the immediate relief from stress, a considerable portion of the movement being due to time-strain, or creep.

Suppose, then, an area of lode is stoped out and left unsupported. If the walls are of strong uncracked rock they will stand; but they will be bent in towards each other in the centre of the area (see Fig. 14). The closure of the stope walls will take place, partly immediately on opening up the stope by expansion of the wall rock, and partly in time by pseudo-plastic deformation or creep. As is well known the latter movement may continue in practice for a considerable period of time after stoping is stopped; if the stope is big enough it may continue until failure of one of the walls occurs. Now the excavation, when formed, will be ringed around by a zone or envelope under increased stress, as shown in the figure. Within this zone will be a dome, in which expansive movement of the rock towards the open stope has relieved it of any part of the stress. This movement is indicated by the arrows. The movement, with its resultant stress relief, will be greatest in the immediate wall and will lessen as the distance from that wall increases, until, at the edge of the stress zone, the relief will be zero—that is, the stress will equal that originally in the rock. Beyond that point increased stresses will be found. The action is identical with that already described as the natural stress relief, which converts drifts of rectangular section into effective excavations of circular or elliptical shape. (When the stress at a certain point is mentioned, the highest of the three components of stress is meant.)

be stoped; thirdly, on the physical nature of the wallrocks; fourthly, on the materials available locally; and, finally, on the value of the ore, with which the cost of the supports is concerned.

When the area to be stoped is small and the rock is strong and the depth not great it is possible to mine without ground control. The walls of the stopes are sufficiently strong to withstand such bending and shearing stresses as are imposed on them, and the stopes are left unsupported until complete. Some Cornish tin mines may be cited as an example. At greater depths, however, and with weaker rocks, some sort of ground control is necessary. The purpose of this control is not to prohibit movement of the rock walls—that ideal is not possible—but so to control their movement that excessive closure does not take place in that part of the stope in which men are working—that is, along the advancing faces. If the stope face is steadily advancing while the walls steadily close in behind, the miners are all the time working between newly-placed and little-stressed supports. Should the face be halted for any reason the action of closure does not stop in synchronism, but catches up, so to speak, on the face advance. As a result of this the walls gradually become more acutely bent at the face and are liable to shear off along it.

One necessity for efficient ground control is, therefore, a steady and reasonably rapid advance of the face. A second is that the oreshoot should be mined in such a way that at no time in the operation is any pillar, remnant, or acute-angled single promontory of ground formed. This means advance by a series of stopes progressing in the same direction, of which the faces form roughly a straight, curved, or stepped line.

If these two requirements are observed, the walls will close in smoothly and regularly whatever kind of supports are used, and there is the greatest chance of the closure occurring entirely by flexure and not by the occurrence of major shear cracks. The important point is that the necessary movements of the rocks should, if possible, be made to take place without fracture. If this can be done perfect ground control is obtained. When shearing of the walls is common, however, the occurrence of the fractures cannot be controlled—they may occur anywhere and at any time, quietly or suddenly (rockburst).

The simplest form of ground control in stoping is to allow the walls to come together and control each other without the interposing of supports. In stoping narrow lodes where the rock of *both* walls is such that closure will take place by bending rather than by fracture, this practice can be followed. It is usually advisable to put in a few compressible supports near the face in order to restrain the closure from coming on too rapidly, and so to reduce the risk of one of the walls shearing off along the face. These supports may be left in to crush up to practically nothing, or they may be withdrawn behind the area in which miners are working, in order to allow the walls to close right in. As an alternative to compressible supports light stull timbers are sometimes used. These automatically break and fall out as closure progresses. Once the two walls are resting solidly together no further movement is possible, levels can be remade along their plane of contact, and excavations inside the walls will be out of danger of any further trouble. Since, however, the total closure permitted in this method is so great it is difficult to provide the original levels with a lining which will remain efficient until and after closure is complete.

It seems obvious that the less the closure permitted the easier it is to keep levels on lode open, the less the strain on the stope walls in bending in behind the face, and the less the penetration of the zones under stress into the walls. In fact, if it were possible to put in supports so strong and so close to the face that no closure at all were permitted, there would be no zone under increased stress. This is manifestly impossible, because the moment rock is blasted out movement of the walls commences, so that before a support can be got in considerable closure will have occurred. The movement which takes place between the original positions of the walls and those when a support has been placed can be termed the 'lost closure', because no matter how efficient that support is, it cannot check closure that has already occurred. The total closure which takes place in a stope depends, therefore, first on the lost closure and then on the rigidity of support. To allow considerable lost closure to take place and then to put in expensive supports is bad practice; the cost of such supports will be largely wasted, as much of the closure they are designed to check will have taken place before they are inserted.

For efficiency in supporting, therefore, the most important thing is to reduce the lost closure to a minimum—that is, to get the supports in as close to the face and as soon after the blast as possible.

Once a support has been interposed the amount of further closure that will occur depends on the stress on the rock—that is, on the depth, on the percentage control used, and on the rigidity of the supporting material. Since for any given place the depth is fixed, the closure of the stope walls can be regulated only by varying either the percentage of supports used or their rigidity. The less the percentage support—that is, the fewer the number of supports used—the greater will be the stress on each one, the more it will yield to that stress, and the greater will be the closure permitted. It is important that the percentage of ground supported should be sufficient to prevent failure of individual supports under excessive stress, except when their failure and full closure of the walls to mutual contact is part of the design. By increasing the rigidity of the supports used for any given percentage support, the amount of closure permitted is reduced. Since, in general, the more rigid the material the greater its cost for a given percentage support, very rigid supports will not be used except where local circumstances render them necessary.

Surface Interference with Doming

When an area to be stoped is very large and the lode has a fairly flat dip, as in coal mining and on some parts of the Rand, the stress zones eventually reach surface, whereafter they cease to exist as domes. All the weight of rock lying above the stoped area must then rest on the supports. At the same time the stress on the rock ahead of the stope faces (the abutments of the old stress dome) is reduced, as the weight of rock above the stress dome, previously carried by it and transferred to those abutments,

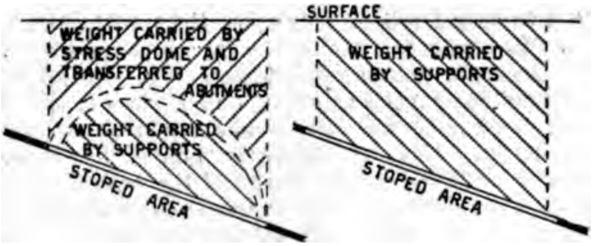

FIG. 15.

is now resting directly on the supports (see Fig. 15).

In any large stoped area there are two parts—a central area and an outer ring. In the central area equilibrium has been established, and hanging and foot are either in contact or held by fully-compressed supports. Further extension of stoping does not affect the rocks nor the supports in this area. Between this central portion and the surrounding ends of ground there is an annular zone in which the wallrocks are subjected to bending stresses and are in process of inward movement. In this annular zone are supports, newly put in, which are not fully compressed and so have not yet arrested the closure of the walls. Stresses coming on the walls in this area are taken largely by the rock in the face on which the walls rest as cantilevers, but partly also by the newly-put-in supports and partly by the fully-compressed supports on the fringe of the central area. In the figures this annular zone has been ignored, as it is very small compared with the whole.

When domes eventually reach the surface, the stress the supports will ultimately be called upon to bear is calculable. Suppose the depth is 2,500 ft., then the weight of superincumbent rock is 200 tons per sq. ft. of horizontal area. Since a rock-filled pigsty or cog of good quality cannot be expected to withstand a load much greater than this, and that only after 30 per cent compression ([6]), it follows that in a flat lode nearly 100 per cent support of such pigsties would be required to restrict the closure to 30 per cent. Clearly this is economically impossible ; it is also unnecessary, for if 100 per cent support is to be used and 30 per cent closure tolerated, sand or waste-rock filling would be equally effective and much cheaper.

It follows that when the area eventually to be stoped is very large it will be idle to attempt to keep the walls apart in the centre of the area other than by some sort of filling. Such supports as are used in the working stopes will be merely to ease the closing walls on to that filling, with which they will eventually become merged. Where no filling is used their purpose will be to ease the walls in until they are in contact with one another and in such cases a type of support should be chosen which will remain efficient until squeezed to a mere fraction of its original dimension. The best units for this purpose are some form of pigsty filled with waste, small mat packs, or waste-rock packs.

In this context it may be pointed out that if the foot-wall consists of a much softer rock than the hanging, so that practically all the immediate closure occurs by the upward expansion of the foot, it must not be supposed that there can never be any surface subsidence. If stoping is widely extended the weight of the overlying rock will eventually be transferred to the foot-wall, either by direct contact or through filling. Thus the foot-wall will again be subjected to the weight of the overlying strata and will be recompressed by it. In other words, the foot-wall, being weaker, closes in more rapidly and in advance of the hanging, but the latter, when it later comes down, forces the foot-wall back again. Thus normal subsidence of the surface can be expected in such cases.

In mining flat lodes when the area to be stoped is small compared to the depth, and when the dip is steep in areas of all sizes, part of the total stress which used to pass through the lode before it was mined out is diverted through the stress zone on to the abutments—that is, on to the lode ahead of the ends of ground. In such cases, therefore, the supports will not have to suffer such heavy squeeze as when the stress dome reaches and is broken by the surface. On the other hand, the rocks ahead of the faces will be more highly stressed than in that case.

This conception is illustrated in Figs. 16 and 17. In Fig. 16, which represents a large area of stoped ground, where the stress zone has been broken by the surface, the vertical stresses through the horizontal lode shown are, in the worked-out portion, of normal value, Q. In the annular area in which the walls are bending towards each other or to the filling, a little stress is taken through the partly-compressed supports, but the majority is thrown on to the rock ahead of the ends of ground. This is transmitted through the small annular dome under stress which bridges this span of lightly-supported walls. The breadth of the stress zone in the rock ahead of the faces (or, in other words, the zone of influence of the stope) is thus dependent on the breadth of the lightly-supported rock spanning between the faces and fully-compressed filling.

The reasoning given affords an explanation of why less trouble is likely to be experienced when filling is used than when the walls are allowed to come together. It is because in the former case the movement of the walls is arrested by the filling much sooner than it would be by their mutual contact were no filling used. The span of the lightly-supported walls is correspondingly much less, with the result that the breadth of the stress zone is also less. In other words, the less the total closure permitted the less the span of lightly-supported walls between fully-compressed supports and the face, and the less the breadth of the zone under appreciable stress ahead of the face. Therefore, the less the lost closure the greater the percentage support, and the greater the rigidity (or resistance to closure) of the supports the less will be the disturbance caused to the distribution of stress in the ground.

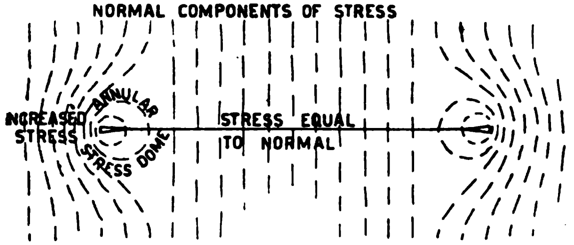

FIG. 16.

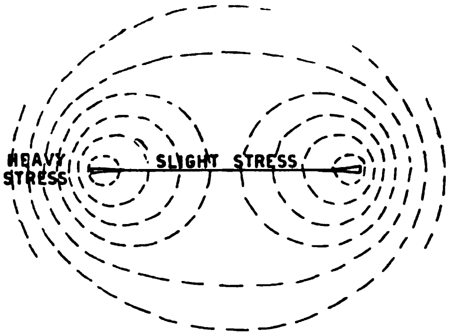

FIG. 17.

In Fig. 17, which represents a small area stoped on a horizontal lode, there is a stress dome covering the whole area and inside it an annular dome of stress bridging the span between the ends of ground and fully-compressed supports. The rock ahead of the faces has in this case to form the abutment not only of the small annular dome but also of the big dome covering the area. The breadth of the stress zone in it—that is, the zone of influence of the stope—is in this case much greater.

When a small stoped area as shown in Fig. 17 is gradually extended until the stress dome reaches surface, the distribution of stress changes to that shown in Fig. 16. The disturbance caused is considerable. The zone of influence of the stoped area of the rock ahead of the faces is reduced—that is, at any point (P, Fig. 18)

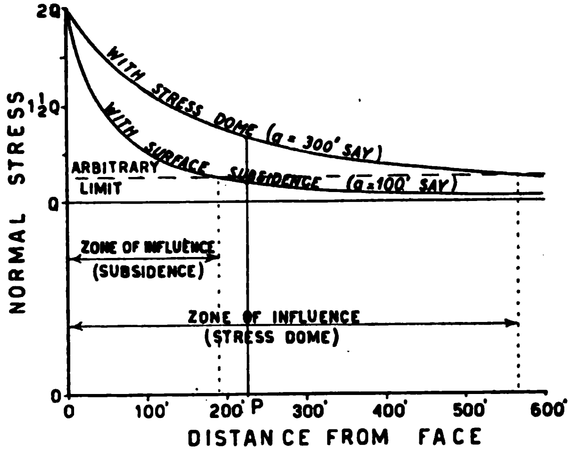

FIG. 18.

ahead of the face the stress is reduced. On the other hand, the stress on the supports in the middle of the area will be increased. This will cause a further compression of those supports which, since the area involved is necessarily a large one, will result in the movement of an enormous mass of rock. Should the action take place suddenly a series of earth tremors is to be expected which may cause rockbursts affecting not so much the working stopes as the worked-out areas and particularly excavations in the walls opposite those areas. Alternatively, apart from the heavy shocks, there may be no other effects.

The removal of a number of pillars which have hitherto prevented doming over a stoping area from reaching surface, will often cause sudden and violent readjustments of this nature. More commonly, however, the rock near surface is softer and much broken up, so that when the stress zone reaches it the change-over from the conditions shown in Fig. 17 to those of Fig. 16 takes place slowly and gradually and will often be imperceptible.

With regard to the possibility of rockbursts occurring as a result of the sudden movement of a large mass of hanging-wall rock as envisaged above, it should be noted that if no supports are used and hanging and foot have previously come together, the amount of movement permitted by the collapse of the stress dome will be much less. It will, in fact, be reduced to the further recompression of the rocks of hanging- and foot-wall, much of which recompression will take place slowly by creep. The practice of using no supports is therefore inherently safer from this point of view, but only if nothing has been left in the stopes to prevent total (100 per cent) closure, and only if the stope width is reasonably constant.

If the stope width varies considerably foot and hanging will come into contact only at points where the stope was narrowest. These points will then transmit very heavy stresses, while those where the stope was wider will transmit none. Excavations in the walls opposite the former will then be liable to suffer heavy punishment. The use of a regular system of supports will lessen this unevenness of stress distribution, especially if more rigid supports are used in the wide places than in the narrow.

When an area is stoped from a steeply-dipping lode, the stress distribution is similar to that shown in Fig. 17, whether the area is large or small, with the difference that, provided that the top of the area is still a considerable distance below surface, the stress domes are unlikely to reach surface if the area is extended.

Closure by Shear

As considered so far the object of ground control has been to control the closure of stope walls so that they bend evenly in without fracture until arrested by supports or by each other. In this context trivial breaks in the immediate wall which do not penetrate to any appreciable depth are not considered as fractures. In the examples given it has been assumed that the object has been achievable. In cases where the ratio of stress to rock strength is excessive, however, it may be impossible, even with the most rigid and close supporting, to prevent fracture of the walls. Fracturing in shear then takes place either regularly and immediately, or occasionally.

With regular shearing of either wall closure comes on very rapidly close to the face, and the wall affected becomes ragged and broken. The shearing takes place after every blast, either as a single crack or a multitude of parallel cracks (stepped shear). Should such shearing be held up by a patch of rock that is locally stronger than the rest, stresses may build up, so that when the failure eventually occurs it takes place suddenly and violently (rockburst). Since the violence of a rockburst depends on the mass of rock which is permitted to move by the occurrence of shear, the aim of ground control in such circumstances must be to reduce the mass of rock likely to be affected and the distance it may be permitted to move to a minimum. In order to achieve this object :

(*a*) supports should be as rigid as possible ;

(*b*) they should be of a type which does not fail under shocks and live loads ;

(*c*) the percentage control used should be high ; and

(*d*) lost closure should be reduced to a minimum by placing supports as close to the face and as quickly as possible. For the last-named requirement (*d*) it follows that the more rapid the advance of the face the better, because there is less time for closure by creep before supports are got in.

It follows that waste-rock filling is not a suitable medium for control in such cases because it usually permits 30 per cent or even 40 per cent closure before it becomes thoroughly compacted and brings the movement to a standstill. Sand-filling is usually compressible by 10–25 per cent, depending on the dip and method of stowage. When well stowed it is, therefore, a more suitable medium. The rigidity of waste-rock filling may be increased by the addition of just enough sand to fill the voids between the stones. The quantity used should not exceed this or there may be a tendency for the filling to flow under stress, causing confining difficulties.

Speed of Face Advance

While it has been reasoned that rapid advance of the face is advisable under conditions of heavy stress where closure by shear occurs regularly or where occasional shear cracks occur, this does not apply to other conditions. Where closure occurs by bending without risk of fracture, time is necessary to allow sufficient creep to take place. In such cases too rapid an advance of the face may mean that insufficient time is given, so that the breadth of walls spanning between the face and fully-compressed supports becomes excessive. This results in throwing excessive squeeze on the rock in the face and excessive bending stresses in the wall-rocks. For each set of circumstances there is, therefore, from the ground-control point of view, an optimum rate of face advance. This optimum rate can as yet only be determined by experience.

Special Methods of Ground Control

Caving

Under some conditions the closing-in of stope walls, while occurring evenly and without fracture, progresses too rapidly—so rapidly, in fact, that it commences ahead of the face. The effect of this is that the lode in the face is subjected to excessive squeeze, as between the jaws of a vice, with the result that it is constantly cracking up and spitting out, to the danger of the miners. This tendency could be reduced by increasing the speed of advance of the face or by putting in a greater percentage of a sufficiently rigid support close to it. In many cases, however, costs and other technical reasons will render these methods impossible. It is found that by deliberately breaking, or allowing to break, the hanging-wall of a stope at a safe distance behind the face, some of the bending stress is removed from that wall and the squeeze on the lode in the face is reduced. The effect is to convert closure by bending to closure by shear, the latter taking place, not at the face where it causes trouble, but under control further back at the caving break.

If the hanging-wall is considered as a heavily-loaded cantilever gripped between the face and the rock which lies above it, it will readily be seen that the effect of caving will be to shorten the overhanging length of the cantilever and therefore to reduce the stress on its fulcrum, the lode in the face (see Fig. 19). The method is

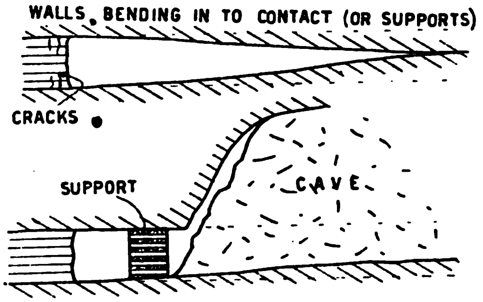

FIG. 19.

particularly suitable for wide lodes where, whatever kind of supports are used, a large amount of closure is bound to take place. As the area stoped is extended, the length of the cantilever and its loading continue to increase almost indefinitely. The method is also used successfully in narrow lodes—as, for example, in the Sub Nigel mine.

In most mines all that is necessary in order to provoke caving of the hanging-wall is to withdraw the supports at a fixed distance behind the working faces, whereupon the ground caves automatically. In some mines, however, the wall must be deliberately broken by blasting to start the action of caving, whereafter it usually continues automatically. Since broken rock occupies much more space than it did when solid, the debris from caving soon fills both stope and cave completely. Thereafter further caving stresses the debris, which acts from then on as filling.

Support by Pillars

When an oreshoot is worked outwards from the main artery, continually lengthening communications have to be kept open through or in the wall opposite stoped-out areas. The cost of keeping such ways open is usually high and there is always the risk of a collapse. Both expense and risk can be avoided if the oreshoot is mined by retreat from the boundary towards the main artery. In the majority of mines this practice is rendered difficult because production is required before the boundary is reached. Early production can, however, be attained without sacrifice of the principle of stoping by retreat, by mining over the areas twice—first outwards from the main artery, leaving behind a series of substantial pillars of ore in order to prevent general closure over the area ; and then in retreat, mining the pillars from the boundary backwards. In the second operation the stoped area is completely abandoned and allowed to cave in. Thus the expenditure on ground control is reduced to a minimum. The method is common practice in American coal mines.

Stepped Longwall

The longwall system of stoping has long been advocated as an ideal, because when all the stope faces are in an even line and progressing at a regular rate the greatest inducements are offered to the wallrocks to close in evenly and regularly.

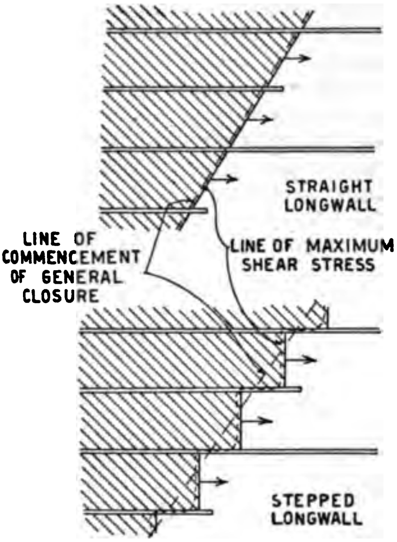

FIG. 20.

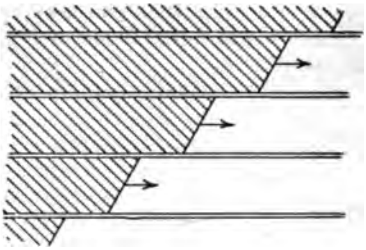

FIG. 21.

Unfortunately this practice also affords the rock of the walls the maximum opportunity to shear. Considering the walls behind the stope faces as long cantilevers more or less uniformly loaded, the maximum tendency to shear is along the stope face, the shear decreasing as the distance from the stope face increases. It follows that the shorter the length of the face between two points the shorter will be the length of the line of maximum shear and the greater will be the tendency to shear. Since a straight line 'is the shortest distance between two points', it follows that the straight longwall puts the greatest shearing stress on the wallrocks. Therefore, when the walls of stopes show a tendency to shear, the straight longwall method of stoping is not to be recommended. If used in such circumstances, heavy supporting with rigid materials is necessary to prevent the fractures from occurring along the working faces—for example, the steel props used in English colliery practice.

If, while keeping the faces roughly in line, individual stopes are arranged to form a series of bays and promontories the length of the total face is increased, so that for a given area of hanging-wall the length of the line of maximum shear stress is also increased. This reduces the possibility of shear occurring along the face. The two methods are illustrated in Fig. 20.

In stepped longwall the walls in the bays are controlled largely by the lode left in the promontories. Actually general closure will usually start in advance of the points of the promontories, so that the latter are crushed by the squeeze. Since the lode at these points has two free faces, violent spitting is unusual, and the effect is commonly a quiet yielding which facilitates breaking ground. Thus general closure of the walls is initiated by the failure of the lode and progresses slowly and gently. It is only on a line along the points of the promontories that there is any considerable tendency to shear, and the vast majority of men are working in advance of this line. In any case the shear stress along this line is much less than the maximum shear stress which is found at the faces, and so the likelihood of fracture of the wallrock is proportionately less.

This method, then, is inherently safer than the true longwall system, the only trouble likely to occur being by reason of the points of the promontories crushing excessively. This can usually be controlled by altering the size and proportion of the steps and increasing the angle of the promontories in the manner indicated in Fig. 21. The method is widely practised in the American collieries ([8]).

SUMMARY

Stress

(1) Save in exceptional circumstances the three components of stress in a rock mass at depth approach equality.

(2) All excavations, whatever their shape, are surrounded by an envelope under stress greater than that pre-existing in the rock.

(3) The shape of this envelope varies from the spherical, through all variations of the ellipsoid, to the cylindrical, according to the shape of the included excavation.

(4) Between the stress envelope and the excavation is a zone of rock in which the stresses are less than those pre-existing.

(5) The maximum stress on the rock approaches double that pre-existing when the shape of the stress envelope is cylindrical and one-and-a-half times that stress when the shape is spherical.

(6) For any given excavation an approximate estimation of the thickness of the stress envelope can be made, and its effect on other excavations gauged therefrom.

Rock Behaviour

(7) Rock subjected to a change in stress yields, part of the induced strain occurring immediately and part as time elapses.

(8) The time-strain tends to a limit below a certain critical load, but above this load it progresses towards ultimate failure.

Control of Rock in Development

(9) Linings which will eventually be subjected to the closure of the walls owing to stoping in the neighbourhood should either be so flexible

that they can tolerate this closure and still remain efficient or else they should be of such a design that they are only in contact with one wall.

(10) Linings which will never be subjected to closure should be of rigid materials built tight against the ground.

Control of Stope Walls

(11) Closure takes place by the inward expansive movements of both walls, partly immediately after blasting and partly by creep.

(12) The primary aim of ground control in stoping is to permit closure by bending while preventing the occurrence of shear.

(13) Ground control in stoping is achieved by the design and sequence of the operation and by the use of supports.

(14) In small stoped areas much of the stress previously taken by the lode is transmitted through the stress envelope to the lode ahead of the faces: the supports have, therefore, to take only a portion of the total load.

(15) The less the closure permitted the less the stress on the rock ahead of the faces and the more the stress on the supports.

(16) When the stress envelope is broken by the surface the supports have to take the full weight of superincumbent rock and the load on the ore ahead of the faces is reduced.

(17) In the latter case the squeeze at all but shallow depths is so great that no unit type of support can withstand it.

(18) When closure by shear is unavoidable the aim of supporting should be to reduce closure to the absolute minimum.

(19) There is an optimum rate of face advance for each individual set of circumstances.

(20) By caving the hanging-wall the squeeze on the ore in the immediate face is reduced.

(21) By stepping a longwall face the tendency of the rock to shear immediately along it is reduced.

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⁎ *Extra copies of this paper may be obtained at a cost of 2s. 6d. each at the office of the Institution, Salisbury House, Finsbury Circus, London, E.C. 2.*

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THE INSTITUTION OF MINING AND METALLURGY

SPECIAL GENERAL MEETING of Members and Associates held in the Rooms of the Geological Society of London, Burlington House, Piccadilly, London, W. 1, on Thursday, 16th December, 1948

Mr. S. E. TAYLOR, *President*, in the Chair.

The Secretary read the notice convening the Meeting.

The President said that the Special General Meeting had been called to consider and, if thought fit, to approve the new By-Laws. The notice calling the Meeting had been read and Members and Associates had received a copy of the new By-Laws, together with a memorandum pointing out the final amendments which had been made. He did not propose to repeat what was stated in the memorandum and would proceed with the formal business of the Meeting. He would propose the resolution and ask the President-Elect, Mr. W. A. C. Newman, to second it. Mr. Newman had been Chairman of the Committee which had dealt with the revision of the By-Laws during the past two and a half years and he and other members of that Committee had done a great deal of pains-taking work in connection with the revision of the By-Laws for which he would like to express the thanks of the Council and members of the Institution. After the resolution had been proposed and seconded, and unless any members wished to make any remarks, the resolution would be put to the vote.

He proposed the following resolution :

'That the By-Laws of the Institution of Mining and Metallurgy as revised and submitted herewith be and are hereby approved and ordered to be presented to His Majesty's Privy Council for confirmation and in substitution of the existing By-Laws of the Institution '.

Mr. Newman said that with the President he commended the new By-Laws for the approval of the members of the Institution. The Council had devoted much time and thought to the formulation of the new rules and hoped and believed that they would reinforce the stature, the dignity and the usefulness of the Institution. The President had referred in kindly terms to the Committee over which he had had the privilege and pleasure of presiding for a few meetings in the absence of Mr. Keith Allen, who was originally chosen to be Chairman of that Committee. He thanked the President on behalf of the members of the Committee, but he thought that a tribute should also be paid to the members of the two previous Committees presided over by Mr. Pryor in the first instance and Mr. Laycock in the second. On two occasions the work in connection with the rules was laid aside because of the incidence of the war and in order that ideas might mature, but to the members of those two Committees a great deal was owed because they did a great deal in digging into and bringing to the surface the problems with which the Institution was faced.

He had much pleasure in seconding the President's proposal.

The resolution was carried unanimously.

The President said that to comply with the By-Laws the Minutes of the Meeting would be read and signed in due course, and while the Secretary was preparing those Minutes there were one or two remarks which he had to make. Now that the By-Laws had been approved they could be presented to the Privy Council for confirmation and when that had been received the changes which had been made would take effect. As was pointed out in the memorandum of 12th April, by making the Institution an examining body and clearly defining the minimum standard required for membership the future status of the Institution would be enhanced.

The other important innovation was the creation of a new class of non-corporate members called 'Affiliates'. That class of Affiliates would include those candidates who would have qualified for Associateship under the old By-Laws by reason of their practical experience but who under the new By-Laws would not qualify for Associate Membership on account of their lack of technical training. In addition, Affiliateship would be open to members of other professions, such as doctors and mechanical and electrical engineers, whose work was closely associated with mining and metallurgy. It was also hoped to find other suitable applicants for Affiliateship amongst those whose work was closely associated with mining and metallurgy—for example, there might be some suitable applicants in mine offices who were engaged on such work as accountancy, costing, statistics, or in the survey offices.

He would suggest, therefore, that Members and Associate Members should bring to the notice of all those who were eligible the advantages of joining the Institution as Affiliates. He would also remind the Meeting that a substantial increase in the membership of the Institution would make it possible to improve the service which the Institution could give to its members and widen generally the scope of its activities.

The Secretary read the Minutes of the Meeting, which were signed as a correct record, and the business of the Special General Meeting was concluded.

THE INSTITUTION OF MINING AND METALLURGY

THIRD ORDINARY GENERAL MEETING of the 58TH SESSION held in the Rooms of the Geological Society of London, Burlington House, Piccadilly, London, W. 1, on Thursday, 16th December, 1948.

Mr. S. E. TAYLOR, *President*, in the Chair

DISCUSSION ON

Notes on the Treatment of Pyrites Cinders at the Plant of the Pyrites Co., Inc., Wilmington, Delaware*

By R. C. TRUMBULL, W. HARDIEK, and E. G. LAWFORD, *Member*

The President said that the paper was presented by three joint authors, Mr. Trumbull, the Plant Superintendent, Mr. Hardiek, Metallurgist, and Mr. Lawford, whom they all knew. Mr. Lawford had kindly undertaken on behalf of his co-authors to introduce the paper, which described the treatment of the Rio Tinto cinders, for the extraction and the recovery of the non-ferrous metals.

Mr. E. G. Lawford, introducing the paper, said that he very much regretted that neither of his co-authors was able to be present. Their duties at Wilmington made it impossible for them to attend the Meeting.

They were presenting the paper with some diffidence, for two reasons. First, owing to the change in conditions to which they referred in the introduction, they had had to go back rather more than ten years in order to find the typical years' operation of the main Henderson process, and they had had to go back even further in respect of the zinc and Glaubers salt recovery steps. It was, admittedly, more satisfactory when a paper described an operation that was actually being carried out rather than one which was, so to speak, in its prime ten years ago.

Secondly, it had to be admitted that the zinc and Glaubers salt recovery process had not proved capable of surviving in the economic blizzard of the early 1930s. After some thought, however, they had decided that the processes described were sufficiently interesting technically to justify the presentation of a paper and that they should face any criticism which it might attract on the grounds already mentioned. He wished to say, in regard to the zinc and Glaubers salt steps, that when he visited a number of cinder treatment works in Europe in 1935, four years after Wilmington operations had come to an end, he found many of the Glaubers salt works in a state of chronic re-vamping and he came to the conclusion that Wilmington operations were not, at the time, necessarily inferior to those being carried on in other

*Bull. 505, Dec. 1948.

places. The authors submitted that the work on the recovery of zinc and Glaubers salt could fairly be regarded as experimental work on a plant scale. In 1931 they had very little to guide them. Furthermore, they had decided to make an anhydrous sodium sulphate of high grade, such as could be sold in a particular market, and that in itself presented special problems which had to be solved.

They made no apologies for presenting the section of the paper dealing with the extraction of lead. They thought that the work at Wilmington was at least as good as, if not better than, that carried out anywhere else where attempts to recover lead from pyrites cinder had been made. With the present price of lead there was no doubt that even with the rise in costs which had taken place

TABLE IX

TYPICAL CHLORIDIZING DATA : AUGUST, 1931

Charge to furnaces	26,782 wet tons mixture.
Furnace hours run	6,467
Tons mixture per hour	4·14

	In Charge Per cent	*In Discharge Per cent*
Moisture	8·7	—
Total Cu	2·24	2·17
Water-soluble Cu	0·71	1·95
Acid-soluble Cu	0·71	0·18
Insoluble Cu	0·82	0·04
Copper solubility	63·00	98·00
Total S	3·89	3·49
Soluble S	1·81	3·44
Insoluble S	2·08	0·05
Pyrites in mixture	0·63	—
NaCl in mixture	11·032	7·7

Temperature (°F.) and Solubility

No. 1		*No. 6*	
2nd	654	2nd	505
4th	824	4th	715
6th	753	6th	610
No. 2		*No. 7*	
2nd	660	2nd	580
4th	804	4th	828
6th	711	6th	725
No. 3		*No. 8*	
2nd	611	2nd	617
4th	750	4th	747
6th	651	6th	680
No. 4		*No. 9*	
2nd	567	2nd	731
4th	756	4th	979
6th	642	6th	804
No. 5		*No. 10*	
2nd	516	2nd	624
4th	685	4th	805
6th	595	6th	731

since 1940, the value of lead recovered would show a reasonable profit over the cost of the lead extraction process.

Passing to the first section of the paper, there was little to be said concerning the operation of the main Henderson process. They had been bold enough to set out certain equations which represented in their view the reactions taking place in the chloridizing furnace, but it would be observed that they had not been dogmatic in their assertions regarding the relative importance of those reactions. He saw that there were present distinguished chemical engineers and he hoped that they would give their views on the chemistry of the process.

Table IX gave details of the chloridizing operation for the month of August, 1931, which could be regarded as fairly typical and which related to a period when zinc was being recovered from the cinders. When the cinders came to the process 63 per cent of the copper they already contained was soluble copper, so that the roasting of the pyrites at the acid works did produce a certain measure of solubility, but, as would be seen, that was increased by chloridizing, the soluble copper in the discharge being 98 per cent. Out of a total copper content of 2·17 per cent in the discharge 1·95 per cent was water-soluble copper and there only remained 0·198 per cent of acid-soluble copper. The soluble sulphur rose from 1·81 per cent in the discharge to 3·44, leaving practically no insoluble sulphur; that was the measure of the conversion of the sulphides to soluble sulphates. The fourth hearth temperatures were round the 800°F. mark. In the paper it was stated that the reaction zone temperatures for reasonable chloridizing had to be maintained between the limits of 800° and 900°F. That was probably a conservative statement and he did not think that a wider variation greatly affected the solubility obtained.

Salt usage was always a somewhat controversial matter. They had given their views in the text and it should be remembered that they were expressed solely with reference to the main Henderson process. No discussion of salt usage was complete without considering the bearing of Glaubers salt recovery and lead leaching. Glaubers salt recovery was on a different footing from the extraction of copper, zinc, silver, gold, etc., because in the one case they were dealing with the extraction of valuable metals from the ore, whereas Glaubers salt recovery was merely the recovery of some of the salt used in the chloridizing process. By using a high percentage of salt, and, if necessary, adding some pyrites to the furnace charge, the production of Glaubers salt could be greatly increased if desired. Their own view, however, was that, provided a plant was treating a cinder containing sufficient lead to justify lead recovery, then the correct technique was to cut down the usage of salt in the furnaces to the minimum required for good copper solubility. That might be as little as 5 per cent salt on cinders. In that case the silver would be recovered with the lead and comparatively little silver would appear in the copper precipitate.

However, it might be that 5 per cent salt would be too little to give a good extraction of zinc, but they had no direct experience on that question. He did know that in Germany present salt usage was still between 10 and 11 per cent on cinders, and it was possible that to get zinc down to 0.1 per cent in the leached cinders that amount of salt was necessary. They would understand, therefore, that the question of salt usage was closely related to such matters as whether or not zinc was being extracted and whether or not any proportion of the salt was being recovered as sodium sulphate, and he confessed that the optimum proportion was a matter on which opinions differed even among those with many years' experience of chloridizing.

It would be seen from the figures that the overall recovery of copper was very good and that virtually the whole of the copper taken into solution was recovered as precipitate. In the text it was stated that liquors were discharged to waste at less than 0·04 g./l. of copper and that statement was more than borne out in the figures given in the operating results for 1936–1937. In regard to overall recovery the 1937 figure was too high to be typical, but the 1936 figure could be regarded as quite normal.

Passing to the section of the paper dealing with zinc recovery, it had been pointed out that the major problem in the recovery of zinc by precipitation with milk of lime was how to avoid introducing iron into the liquor, or rather limiting it to the smallest possible amount. The co-precipitation of zinc when precipitating the iron was a major loss and the actual operation of oxidation and precipitation was a costly step in the process.

In Germany the precipitation of copper by copper precipitate was at present the method adopted and there was no longer any attempt to produce by selective leaching a zinc fraction with a very high zinc-copper ratio. The whole of the liquor from the copper leach was treated in zinc recovery. It was true that owing to the chloride content the precipitation of copper by cement copper was incomplete and there was a residuum of iron which had to be oxidized and precipitated, but there had undoubtedly been some increase in the zinc percentage recovery by treating the whole of the liquor, although the loss in the iron precipitate was still high.

The alternative method was to use zinc to precipitate the copper and so avoid the introduction of iron completely. That was the method they had adopted at Wilmington. It was obvious that as substantially all the zinc introduced could be recovered the cost of the precipitant was really only the difference between the market price of the scrap metal and the stage value of zinc in liquors prior to precipitation. The method would not be economical if applied to the whole volume of liquor, but where it was possible to produce a fraction containing a high proportion of the total zinc with a zinc copper ratio of 6 or 7 to 1, then it seemed to him that the method merited serious consideration.

Concerning the Glaubers salt and salt cake production there was little to say. They could not claim that the work done at Wilmington was satisfactory by modern standards. Great advances had been made in Germany in that branch of the process. It was interesting to note that in Germany, instead of treating the liquor from the Glaubers salt melters by evaporation, it was returned for a second cooling, yielding another crop of Glaubers salt; that was melted and a second lot of anhydrous salt obtained. Only a very small volume of liquor was eventually treated by evaporation. Centrifuges had entirely superseded filters and the whole operation seemed to be easy and trouble free. It must, however, be remembered that 17 years had elapsed since their struggles at Wilmington and there would be no attempt to-day to set up the process on the lines on which Wilmington was then working.

Passing to the section dealing with the extraction of lead, it was important to remember that at the time to which that section referred the price of lead was 7 cents per lb. The object of the process was to reduce the lead content of the cinders to a certain required limit and only to that limit, and in so doing to obtain the highest credit possible against process costs for recovered lead.

He thought they (the co-authors) had not perhaps sufficiently explained that the reason for precipitating the lead from the brine by milk of lime was two-fold. First, it prevented the build up of sulphates in the brine, and, secondly, it greatly reduced the bulk of the liquor going forward to lead precipitation proper. The lead content of the brine was only about 8 g./l. whereas the lead content of the solution going to the final precipitation step was about 100 g./l. As that solution had to be heated continuously during the precipitation step the absolute necessity for concentration would be appreciated. Members would see that no attempt was made to precipitate lead from the first tank effluent, notwithstanding the fact that it contained about 1 g./l. The reason was that, with lead at 7 cents, it simply did not pay to handle that weak liquor. Undoubtedly to-day that first effluent would be treated.

The effect of the SO_4 content on the solvent power of brine was shown in the curves (Fig. 15), but he would emphasize strongly that those curves were only a rough approximate guide. They did not represent in any way prolonged or deep research, but were put forward merely as a rough works experiment which indicated a trend.

It was the fact that that weak effluent was not treated that largely accounted for the poor recovery of the lead taken into solution and that recovery could undoubtedly be much improved to-day.

It would be realized that economical and efficient heating of solutions was a *sine qua non for the* lead process. The solution heater described in the Appendix was entirely the invention of his co-authors and he thought members present would agree that they deserved great credit for a very ingenious and effective piece of apparatus.

Finally he would like to express his personal indebtedness to Mr. C. T. Hill, whom he was very glad to see present. Mr. Hill was for many years closely connected with operations at Wilmington and he could truly say that without his help the paper would never have been presented.

Dr. S. I. Levy* said that although the Henderson process of chloridizing roasting had been known and practised for a very long time, very little detailed information had been published on it and as far as he knew no operating data had been disclosed up to the date of the paper. The authors, the Pyrites Co. and the Rio Tinto Co., were therefore much to be congratulated on having made available this big volume of operational information, and Mr. Lawford was particularly to be congratulated on the skill with which he had handled the great mass of detail contained in the paper.

Many years ago the speaker carried out for the Rio Tinto Co. a good deal of pure chemical research work on a long-term examination of questions connected with pyrites. Some of that work dealt with questions closely related to those which arose in dealing with chloridizing, and he had been, therefore, most interested in those parts of the paper which dealt particularly with the salt roasting process. He had examined the data which had been presented in the paper to see if they threw any light on the reactions which took place.

The cinders to be treated were in the main ferric oxide. They contained relatively small amounts of sulphur, present partly as sulphates and basic sulphates of zinc and copper, and partly as metal sulphides. He agreed that zinc and copper were more likely to be present as sulphides than as oxides, so that some at least of the sulphide sulphur was associated with the zinc and copper.

He had calculated from the typical analysis of the cinders which the authors gave in Table I the relative numbers of atoms of the elements present. He would not take the meeting through his calculations, but would give briefly a summary of the results. There were, very roughly, 4 atoms of zinc to 3 of copper. For his present purpose he took zinc and copper together as especially the non-ferrous metals which were to be brought into reaction. For every 15 atoms of zinc and copper together, there were 10 atoms of sulphur as sulphate sulphur and 14 atoms as sulphide. If there were no oxides of zinc or copper present there would be 10 molecules of zinc and copper sulphate, and 5 molecules of zinc and copper sulphide. That would leave 8 atoms of sulphide sulphur which must be combined with the iron. The typical analysis

*Consulting Chemical Engineer to the Rio Tinto Co., Ltd.

showed only 3·71 per cent of sulphur in all in the cinders, and that, as the paper made clear, was increased somewhat before chloridizing to something of the order of 5 per cent of sulphur by adding pyrites or pyrrhotite. The cinders after chloridizing still contained small quantities of sulphur and, allowing for that, the sulphur which was concerned in the reaction would seem to be in all about 4·7 per cent, of which 1·65 per cent was present as sulphates. Allowing for the additional sulphur which was added beyond the amount shown in the typical analysis, the sulphur combined with the iron increased from 8 atoms to, say, 14 atoms, and the total sulphur as sulphide increased from 13 atoms to 19 atoms—that is, all the metal sulphides present, zinc, copper and iron, were equivalent, on the basis he had adopted, to about 19 atoms of sulphur.

The practice in 1937, the year he took, was shown by the data given. Fourteen tons of salt was used to every 100 tons of cinders, which corresponded to about 48 molecules of salt for 5 molecules of zinc and copper sulphide, and 14 molecules of iron sulphide—that was, to the 19 atoms of sulphur as sulphide which he had calculated to be present. There were, therefore, 48 molecules of salt to 19 atoms of sulphur as sulphide. Every atom of sulphur as sulphide would require 2 molecules of salt for reaction, so that the 19 atoms of sulphur would require only 38 molecules of salt. Since 48 molecules were available it was obvious that there was an excess of salt present.

That established one very important factor for the reactions which had to occur. Another important factor could be established from the data which the authors had given on p. 6 of the paper as to the composition of the gases drawn from the furnaces. The tower acid obtained by scrubbing the gases with water had a strength of about 50 g./l. The ratio of hydrochloric to sulphuric acid was given as 5 to 7 parts HCl to 1 part H_2SO_4. He had assumed that to be parts by weight, and he had taken 6 parts of hydrochloric acid to 1 part of sulphuric acid. That corresponded to 16 molecules of hydrochloric acid for each molecule of sulphuric acid, and, on the basis of 900 lb. of water per minute, corresponded to a production of 3 tons of sulphuric and 19 tons of hydrochloric acid in the gases each day.

That was rather a surprising result, since if the 1937 rate of treatment were taken as given in the paper, say 650 tons of cinders per day, it meant that, whereas about one-third of the chlorine in the salt was removed from the furnaces in the gases, something less than 4 per cent of the sulphur which was brought into reaction was removed in that way. There again the calculations were approximate, and he did not know whether all the data he had used were obtained in the same period and under the same conditions, but he thought that the conclusions might be accepted as being not far from the true position in normal operation at Wilmington.

The operational data therefore appeared to disclose : (1) that a very considerable excess of salt over that required for reaction was present during roasting ; (2) that about one third of the chlorine in the salt was removed during the reaction and appeared as hydrochloric acid in the gas ; and (3) that substantially all the sulphur remained in the cinders, the amount driven off in the gas being less than 4 per cent.

The conclusions to be drawn from those facts seemed to be as follows : First, that substantially all the sulphide sulphur brought into reaction had to be converted to sulphate in the roasting, only a very small proportion of sulphur being driven off. It followed that hardly any direct combustion of sulphide in the air could occur, and there could only be slight production of sulphur dioxide. Thus equations (1), (2), (5) and (6) of the possibilities set out by Mr. Lawford could occur, if at all, only to a very small extent. Secondly, there must be direct reaction between the salt, the air and the sulphides to form sodium sulphate and chlorides of the metals which were originally present as sulphides, as indicated in Mr. Lawford's equation (7). That represented the chief course of the reaction, and must be the main reaction which occurred. That direct reaction must involve substantially all the sulphide sulphur which was brought into reaction, which was therefore converted to sodium sulphate, the zinc, copper and iron originally present as sulphides being converted to chlorides. Thirdly, since ferrous chloride reacted readily when heated in air, the ferrous chloride formed would be oxidized by the oxygen of the air to ferric oxide and ferric chloride. Ferric chloride was very volatile and was driven off as such in the gases, reacting in the vapour phase with the moisture of the gases to form ferric oxide and hydrogen chloride. That seemed the most probable explanation of the relatively large quantities of hydrogen chloride in the gases.

The alternative explanation of the formation of hydrochloride was that indicated by Mr. Lawford's equation (4). That required that direct reaction should take place between moisture, salt, oxygen and ferrous sulphide and it seemed to the speaker to be likely to occur, if at all, only to a small extent, since the moisture in the charge was driven off in the upper part of the furnace before the charge was hot enough for reaction to take place.

If, in fact, hydrogen chloride were formed by reaction between water vapour and ferric chloride in the gases, as suggested, there would be formed at the same time considerable quantities of finely divided ferric oxide which should partly settle out in the flues and partly be washed out in the towers and collect in the tower acid. If that explanation were correct, therefore, the tower acid should contain that finely divided material and one would expect to find in the flue dust some proportion of much finer material than would otherwise be present.

The course of the main reactions would, therefore, appear to be:

(a) In the furnaces—

(1) $(Zn, Cu, Fe) S + 2 NaCl + 2O_2 = (Zn, Cu, Fe) Cl_2 + Na_2 SO_4$.

(2) $6 Fe Cl_2 + 1\frac{1}{2} O_2 = 2 Fe_2Cl_6 + Fe_2O_3$.

(b) In the gases—

(3) $Fe_2Cl_6 + 3 H_2O = Fe_2O_3 + 6 HCl$.

There were one or two other matters on which he would like to comment. There were several features in the Wilmington practice as described in the paper which indicated considerable advances over practice in other works. The continuous precipitation of copper in revolving drums was unique. The Venturi solution heater was an interesting and very useful method of heating large quantities of liquor where direct steam had to be avoided. The method adopted for preparing pure anhydrous sodium sulphate was peculiarly interesting.

The lead extraction method differed from any other which he knew of, although he was puzzled by the statement that the chemistry of that method was not clear. He had always thought that lead must be present as sulphate during the chloridizing stage; in any event it would be present as sulphate before the the leaching. Lead sulphate was soluble in hot brine by conversion to lead chloride; that conversion was impossible, as indicated by the solubility curves in Fig. 15, if the brine contained any considerable quantity of soluble sulphate. The precipitation of lead by milk of lime, which at the same time removed sulphate from the solution, was therefore a very effective method of dealing with the pregnant lead brine, and in combination with the further steps of dissolving the lead hydroxide, precipitating the metal on scrap iron and treating the slurry in a cone classifier, constituted an elegant and effective method of recovering lead from pyrites cinders in an acceptable form.

Those responsible for operations at Wilmington—and he believed that part of the time they were under the control and supervision of Mr. C. T. Hill—had certainly introduced several novel and ingenious devices into the old Henderson process, and the paper was valuable for its description of these important improvements as well as for the presentation of such interesting process data.

Mr. F. D. L. Noakes felt that the authors should be congratulated on presenting a paper which was extremely complete and concise. There were, however, one or two points which he would like to raise.

It was stated that cinder crushing was done in two stages by a Jeffrey swing-hammer pulverizer. He had had experience of similar pulverizers for crushing coke and had found that they were subject to a considerable amount of wear. He thought at first that cinder would have similar properties to the coke, but he imagined that it was merely a conglomerate which required breaking up by the action of the hammers. He would like to know how those crushers behaved with the cinder.

The authors said that 50 to 70 per cent of the non-ferrous metals were present as sulphates. He had intended to ask how they arrived at that conclusion but he believed that Dr. Levy had already gone into it. The analysis of Rio Tinto cinders given in Table I showed 0·013 per cent cobalt, and he would be interested to know what happened to that cobalt. He noticed in the analysis of the precipitate copper a certain amount of nickel (·07 per cent) was shown but there was no mention of cobalt. He asked if it built up in the solution and, if so, what happened to it in the end.

On the question of the addition of sodium chloride, the paper said 'Whether the salt is added at the furnace or at the tanks, an increase . . .'; one would imagine that the salt was first required in the furnace for the chloridizing process and then later as brine for the leaching. He wondered if any attempt had been made to split those additions—that is, to add some of the sodium chloride in the furnaces in order to provide the chlorine for the chloridizing roast and then make a second addition by putting it in the leaching process to act as a brine leach.

The precipitation of copper was carried out by scrap iron in the usual way. He believed he was right in saying that at Spektakel, near the O'okiep mines in Namaqualand, tinned scrap was used and had been found to give very efficient precipitation of copper. He imagined that a certain amount of tin went into the precipitate, but it must be a very small amount and it did not affect the value of the copper. He wondered if anyone had any experience of the use of tinplate scrap instead of iron.

He noted that flat-bed vacuum filters were used, and wondered if that was a matter of availability or whether they were found to have any advantage over the rotary drum vacuum filter.

The precipitation with the zinc-copper alloy seemed to be very effective and he imagined that the silver was co-precipitated with the copper. He would like to know if it was a complete precipitation or whether some of the silver stayed in solution.

There were two small points on p. 19: the analysis of the de-copperized liquor showed for zinc '(*g./l.*) 1·0', which he believed to be a misprint for '91·0', and sodium sulphate was misprinted lower down.

Dr. Groves said he had noticed that, in their introduction, the authors explained that since 1940 Rio Tinto pyrites had been largely replaced by brimstone, and that in consequence the process they described at Wilmington now proceeded on a much reduced scale. A similar change had taken place in this country.

Before the last war, iron pyrites was the principal source of sulphur for sulphuric acid in Great Britain. It also gave rise to a particularly valuable iron ore, for, on burning, one ton of pyrites usually yielded between 14 and 15 cwt. of cinders containing 60–65 per cent Fe, and 3 to 5 per cent SiO_2. If the copper content was not much above 0·25 per cent, the material was sold to the iron and steel industry, which, after sintering it to achieve the dual purpose of removing the bulk of the 2 to 3 per cent of sulphur remaining and agglomerating the material to a suitable physical form, used it in the blast-furnace burden. Such material afforded the equivalent of a valuable low-phosphorus haematite and was used in the manufacture of haematite iron. Pyrites cinders that were too cupriferous were first submitted to the Henderson process, the resulting copper-free powdery material being called purple ore. At the outbreak of war there were three firms operating the process in the United Kingdom. Their combined capacity was about 180,000 tons of cinders annually ; a further 100,000 tons of cinders used to be sent every year to Germany for treatment. In 1938, the total purple ore and pyrites cinders consumed at blast-furnaces in the United Kingdom was 163,100 tons. In spite of a greatly increased steel production, the corresponding figure for 1947 was only 68,000 tons.

Thus, in the United Kingdom, as in America, much of such industrial arrangements had been swept aside as the result of the use of American sulphur in place of pyrites in sulphuric acid plants. Presumably there were good reasons for doing this, and in view of the dollar expenditure involved, one would think they must be extremely good reasons. He thought that it would be of general interest if they could be told something of the reasons for that important change.

Mr. Gordon S. Duncan asked, as a mining engineer and not as a metallurgical chemist, one question on the fourth full paragraph on p. 5 ; a fairly practical question. It had been his lot to have to sell his clients' production of pyrites in various parts of the world and subsequently to place their cupreous cinder as well, and some of the copper extractors handling the cinder rather liked to have a certain amount of sulphur left in the ore for the very reason stated by the authors. Others, mainly in Continental Europe, much preferred the low-sulphur cinder—that is, the calcine which was roasted nearly sweet—adding a controlled amount of sulphur in the form of pyrites to the cinder before chloridizing it.

He would be very glad if the authors could say whether there was any way of finding out which really was the more economic method to follow, (*a*) in respect of lump ore, and (*b*) in respect of fines. It seemed to him rather rule-of-thumb. At the I.C.I., Ltd., Widnes Works, in the old United Alkali Company's days, they used to leave 5 to 5½ per cent sulphur in the cinders, but in leaving a kernel of sulphide and sulphate surrounded by calcine they surely could not get such good reactions in a chloridizing furnace as they would by roasting it nearly sweet and then adding a controlled quantity of green ore afterwards.

Mr. Stanley Robson said he spoke with the reservation that he had no operating experience of that particular process, but he welcomed a paper of that type dealing with a metallurgical extraction process and considered it filled a definite gap in the Institution records. The paper was fully descriptive and as such was a very useful contribution. He would like to say how much he appreciated the manner in which Mr. Lawford introduced it.

He was also interested in Dr. Levy's analysis of the chemical reactions involved in the process, which was based on the various stoichiometric relationships. He thought a similar survey from the thermo-dynamic point of view with due consideration of free energies would throw light on the probable reactions which actually took place in the course of the operation and would at any rate eliminate some of the possibilities which stoichiometric considerations alone had suggested. That was a problem for the physical chemist, but such a survey might remove much of the obscurity which surrounded the analysis of the chemical problem at present. In the absence of the guidance which such an analysis would give he thought that the possible reactions between salt and SO_2 in the chloridizing furnace, to give a direct production of hydrochloric acid and sodium sulphate, should be considered. That reaction was actually used at one time in the well-known Hargreaves process.

He thought that the detailed description of the metallurgical procedure added much to the value of the paper, particularly as it was supplemented by details of the materials used in the construction of the plant and proved satisfactory by experience. The choice of materials to be used in the construction of such plants was often difficult, and experience was in many cases the only reliable guide. One of the great merits of the paper, therefore, was the complete statement of exactly what was used.

The addendum to the paper on the interesting use of a jet heater with a thermal efficiency of 80 per cent gave a neat and economical solution of the problem of heating. The use of the same device for mixing was also a very happy thought.

Mr. J. Jacobi said that he also was very interested to read about the Venturi solution heater. A similar device existed in the submerged flame burner, which gave exceedingly high efficiencies and was used not only for heating but also for evaporating solutions of inorganic salts even in multiple effects. The disadvantage of that type of heater was that it could only be used with town gas or some other clean gas, which was an exceedingly expensive form of fuel, in spite of the high heating efficiency; where steam could be used it was usually much cheaper.

He was particularly interested in the zinc extraction section of the paper; he thought that zinc chloride solutions of the type described could cause great difficulty—he had had some experience with them. Unfortunately there was not a market in Great Britain for such solutions with about 1 g. of iron per litre—the iron usually had to be of the order of 1 mg. or less; the market for zinc chloride solutions, as such, was limited but still considerable.

The authors did not say anything about the proposed method of recovering the zinc; a zinc oxide precipitation was feasible but beset with many difficulties and it was interesting to know that the I.G. Farben Industrie had developed a process of making a zinc oxide in that way, which, although not claimed to be suitable for colour purposes, was at any rate very good for impregnating rubber.

The authors mentioned tunnel burners for obtaining a suitable reducing flame in the roasting furnaces. He had made a series of experiments with different burners also on town gas for small-size furnaces, and a very useful form of burner had been developed, running round the periphery of the furnace, throwing the flame not tangentially on to the hearth, but upwards, so that it formed a dome over the arch of the furnace. It was also possible to alter the conditions to oxidizing or reducing as necessary. That had proved very successful in an application where, unfortunately, owing to the endothermic nature of the reaction, one needed not 800 ft. but 8,000 ft. of gas per ton treated.

He had often wondered about the maximum capacity of roasting furnaces on different materials. The authors mentioned a figure which gave an area of 21 sq. ft. per ton per 24 hours. No two calcines needed the same treatment, but he would like to mention that it had been possible to roast manganese ores at a very much higher rate, the endothermic reactions being carried out at the rate of 8 sq. ft. per ton per 24 hours, which he thought formed a record. It was gratifying to note that that was done not somewhere in America but within 10 miles of Burlington House.

Mr. Harry C. Lancaster said he agreed with the other speakers with regard to the great interest shown in the authors' descriptive practice of the Henderson process, as by careful and thorough technique on their part they had proved it was possible to recover not only the copper but also the gold, silver, lead and zinc present in varying quantities.

It would not be reasonable in present days of fluctuating markets and rapidly-changing conditions to pin the authors down too closely on the score of costs. Nevertheless they were of vital importance, and it should be noted that the lead in 1940–1941 cost about £36 per ton to recover, whereas the lead price during those years was pegged at £25 per ton, thus entailing a heavy loss. The authors might rightly claim that it was essential to meet the ironmasters' specification, and moreover lead to-day was £123 per ton—a very different story.

Many years ago he was asked to visit a derelict site in the Weardale district, where there had been originally three iron blast furnaces. There was nothing left but three enormous holes in the ground with from 80 to 90 tons of lead in the bottom of each. The iron ore they had smelted had evidently contained some lead which found its way to the bottom and had seeped through the brickwork. He mentioned that because it seemed to him that lead contents would be more cheaply and better recovered if the bottom of the blast furnaces could be constructed so that the lead could be recovered direct, instead of seeping into the ground.

It was a curious if not a striking fact that so much ingenuity was shown in removing both the copper and lead from those iron residues. The speaker could vouch that over 200 tons of lead had been added that year to iron and steel to speed up its turning qualities and in all probability much more was similarly used in the United States. Moreover, copper was being added to iron for many purposes, but doubtless the governing factor was when and how those additions were made.

Dr. Levy said that he believed, and had assumed for the purpose of calculation, that all the sulphate sulphur in the cinders was combined with zinc and copper, that the rest of the zinc and copper was present as sulphides combined with elementary sulphur, and the rest of the sulphur present was combined with the iron as sulphide. He agreed that zinc and copper were likely to be present as sulphides rather than as oxides.

The President expressed the congratulations and thanks of the Meeting to the three co-authors for their admirable paper, and particularly to Mr. Lawford for his able introduction. They would look forward with great interest to a reply from the three co-authors in due course.

THE INSTITUTION OF MINING AND METALLURGY

DISCUSSION ON

A Note on 'Steel' Galena*

By G. A. SCHNELLMANN, *Associate*

The paper was also presented for discussion, but in view of the lateness of the hour no discussion ensued and **the President** invited discussion by correspondence.

CONTRIBUTED REMARKS

Mr. T. Eastwood: I have never seen steel ore *in situ,* but Dr. Stanley Smith, whose description is quoted in the paper, is known to me as a careful observer. That steel ore has some connection with movement is obvious. While secondary growth replacing fault material is possible for galena and probably true of calcite and quartz, the explosive nature noticed by the old Derbyshire miners seems to indicate direct connection with movement, with the deformation of ordinary galena into steel ore. With such an origin it is not surprising that the composition of steel ore is variable, for ore constituents of the lode in the vicinity of the movement must have contributed to the resulting gouge.

Dr. Stanley Smith notes the effect of movement on fluor; has Dr. Schnellmann noted the effect on other minerals with good cleavage—such as calcite, barytes, blende—or on minerals devoid of cleavage—such as pyrite or quartz—and, if so, has he noted any strain phenomena under the microscope?

A high silver content is attributed to steel ore, although not shown in the analyses. I have noted a high silver content in ordinary galena when associated with barytes, in the Lake District, but have failed to find a clue to the reason, and I should welcome suggestions.

Dr. David Williams: Laboratory experiments carried out by Messrs. F. D. Adams, W. H. Newhouse and others have demonstrated that the softer ore minerals, including galena, can be deformed and made to flow under high differential pressure. Adams showed that a gneissic structure produced by the deformation of galena was accompanied by an increase in toughness of the mineral and that flowage was effected by internal gliding and twinning. At Sonora, Mexico, fine-granular 'steel' galena found alongside slips is attributed to the deformation and crushing of coarser, lamellar-twinned galena, the two types merging into each other. There the fine-grained galena is not abnormally rich in silver, whereas at the Halkyn mine, and at Coeur d'Alene, Idaho, 'steel' galena is appreciably more argentiferous than common galena.

*Bull. 505, Dec. 1948.

I have recently examined polished sections of 'steel' galena and an undeformed cube of galena from Halkyn. As no discrete silver-bearing minerals—such as argentite or tetrahedrite—could be detected even under high magnification, an attempt was made to ascertain the distribution of silver within the galena by the contact film method. Gelatin-coated paper saturated with an attacking reagent (1 : 10 HNO_3) and a specific reagent for silver (a mixture of 2 : 5 of 20 per cent formaldehyde and 10 per cent KOH), after being pressed on the polished surfaces, gave prints suggesting by their uniform colour that in both cases silver is present in solid solution in the galena. Moreover, the deeper tone of the print obtained from the 'steel' galena confirmed that it is richer in silver than the undeformed cubic variety, this fact being further corroborated by spectrographic analysis. It would be interesting to know whether there is any tendency for galena alongside the 'steely' type to be impoverished in silver, as if to suggest that the precious metal had migrated under pressure towards the crush zones.

Etching the polished surfaces with 1 : 5 HCl brought out no vectorial textures in the cube of galena, but part of the 'steel' galena specimen developed a series of lamellæ reminiscent of slip or glide planes due to plastic deformation of the sulphide.

What are the gangue minerals in the galena, said to be represented by the irregular black patches in the photomicrographs? Thin-sections might indicate that these minerals, presumably carbonates, exhibit signs of deformation and preferred orientation, and thus lend support to Dr. Schnellmann's justifiable contention that the texture of schistose and 'steel' galena at Halkyn is the result of pressure.

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BULLETIN OF THE INSTITUTION OF MINING AND METALLURGY

Principal Contents :

OFFICIAL NOTICES

GEOPHYSICS AND ECONOMIC GEOLOGY
By J. McG. BRUCKSHAW

AN IMPROVED TYPE OF KONIMETER
By J. P. REES, *Associate*, and S. R. RABSON

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NOTICE OF GENERAL MEETING

The Sixth Ordinary General Meeting of the Fifty-Eighth Session of the Institution of Mining and Metallurgy will be held, by kind permission, in the Apartments of the Geological Society, Burlington House, Piccadilly, London, W. 1, on **Thursday, 17th March, 1949**, at 5 p.m.

The paper entitled **Geophysics and economic geology**, by Dr. J. McG. Bruckshaw, which is published in this issue of the *Bulletin*, will be submitted for discussion. Dr. Bruckshaw, who is University Lecturer in Physics at the Imperial College of Science and Technology, will introduce his paper at the Meeting.

Light refreshments will be provided at 4.30 p.m. for members and visitors attending the Meeting.

The Council invite written contributions to the discussion of papers from members who may be unable to be present at the Meetings of the Institution. The Council reserve the right to edit and condense such contributions.

INSTITUTION NOTES

Malaria Control Course for Laymen

The 'lay' course in malaria control for planters and miners is to be re-established by the Ross Institute and will be held from 18th to 22nd July, 1949.

The following is an extract from a notice circulated by the Director of the Ross Institute of Tropical Hygiene:

'Before the war this course was very successful and greatly appreciated. It was the main agency which produced a generation of planters and miners who not only understood the gravity of malaria but had a considerable special knowledge of how it could be controlled.

'This generation is now growing older and is being replaced by one without this knowledge, which is as necessary for efficiency and economy in these days of D.D.T. and paludrine as it was before.

'Future programmes of instruction will depend very much on the success of this next course. Your active co-operation in securing attendance at this course will be of great help in ensuring their continuation, and is earnestly requested.

'It would be appreciated if agencies and firms would inform their managers and assistants that the course is being organized and encourage and assist them to attend. There is no fee.'

Notification of intention to attend the course should be sent as soon as possible to the Organizing Secretary, Ross Institute of Tropical Hygiene, Keppel Street, Gower Street, London, W.C. 1.

Index of Recent Articles

As members will see, the Universal Decimal Classification has been adopted, with the authority of the Council, for the Index of Recent Articles, and will be the system used in the Joint Library.

The pamphlet describing the library and information services includes an explanatory section on the new system of classification.

Annual Dinner, 1949

The Annual Dinner of the Institution, which was not held last year, will take place on Thursday, 5th May, 1949, at the Savoy Hotel, Strand, London, W.C. 2. Full particulars will be published later.

Fifty-Eighth Session, 1948-49: Dates of Subsequent Meetings

The following are the dates fixed for General Meetings of the Institution during the remainder of the Session 1948–49:

21st April, 1949.
19th May, 1949.
16th June, 1949.

(These dates are the third Thursday of the month.)

April General Meeting

The paper to be submitted for discussion at the General Meeting to be held on 21st April is entitled 'Recovery of sulphur from smelter gases by the Orkla process at Rio Tinto.' The authors are Messrs. H. R. Potts and E. G. Lawford, *Members*, and Mr. Lawford will introduce the paper at the Meeting.

February General Meeting

The Fifth Ordinary General Meeting of the Session took place at the Geological Society of London on Thursday, 17th February, when over 80 members and visitors were present. Mr. Jack Spalding, *Member*, introduced his paper entitled 'Ground control—theory and practice', and illustrated his remarks with lantern slides. The interesting discussion which took place will be reported in the April issue of the *Bulletin*.

Candidates for Admission

The Council welcome communications to assist them in deciding whether the qualifications of candidates for admission into the Institution fulfil the requirements of the By-Laws. The application forms of candidates for Membership or Associateship will be open for inspection at the office of the Institution for a period of at least two months from the date of the Bulletin *in which their applications are announced.*

The following have applied for transfer since 10th February, 1949:

To MEMBERSHIP—

Raoul Gustin Bergman (*Paagoumene, New Caledonia*).

John A'Court Bergne (*Great Missenden, Buckinghamshire*).

To Associateship—
Ronald Frederick Jarvis (*Marikuppam, South India*).
Lindsay Lee Shearer (*Mufulira, Northern Rhodesia*).

The following have applied for admission since 10th February, 1949:

To Membership—
Gordon Colvin Lindesay Clark (*Melbourne, Vic., Australia*).
John Henry Harris (*Dodoma, Tanganyika Territory*).

To Associateship—
Ian Scott Ferguson (*Jos, Northern Nigeria*).
Harry Raymond Miles (*Tarkwa, Gold Coast Colony*).
Robert Laurence Rodda (*Jos, Northern Nigeria*).

To Studentship—
John Trevor Hall (*London*).
Miles Holme Russell (*London*).
Roy Russell (*Hillingdon Heath, Middlesex*).

Transfers and Elections

The following were transferred (subject to confirmation in accordance with the conditions of the By-Laws) on 10th February, 1949:

To Membership—
Henry Albert Lavers (*Minas, Brazil*).

To Associateship—
Ivor Eric de Beer (*Que Que, Southern Rhodesia*).
Douglas Foakes Fairbairn (*Polis, Cyprus*).
Alfred Jose (*Redruth, Cornwall*).
Peter Frank Farnham Lancaster-Jones (*Springs, Transvaal*).
Hans Walter Pokorny (*London*).
William Victor Symes (*Milburn Grange, Westmorland*).

The following were elected (subject to confirmation in accordance with the conditions of the By-Laws) on 10th February, 1949:

To Membership—
Fernand Albert Jean Blondel (*Paris, France*).
Pierre Charles Alexis Legoux (*Paris, France*).
Henri Marie François Nicolas (*Tunis, Tunisia*).
James Haward Taylor (*London*).

To Associateship—
Wallace Reginald Coke (*Bulawayo, Southern Rhodesia*).
Alexander Forbes (*Sungei Besi, Malaya*).
James Galbraith (*Hangha, Sierra Leone*).
David Stanley Lawn (*Nsuta, Gold Coast Colony*).
Robert Murray Meikle (*Camborne, Cornwall*).
William McNeil Smith (*Scone, Scotland*).

To Studentship—
James Jamieson Bell (*Camborne, Cornwall*).
William Neil Blayney (*Camborne, Cornwall*).
Eric Charles Blunden (*Hayle, Cornwall*).
Norman Flexman Burrell (*Truro, Cornwall*).
Geoffrey Adam Daniels (*Gwithian Towans, Cornwall*).
James Gurney Gateward Davis (*Camborne, Cornwall*).
Thomas Foy Downs (*Manchester, Lancashire*).
Geoffrey Alan Dunthorne (*Seaford, Sussex*).
Brian William Hester (*Ruislip, Middlesex*).
Dale Seaton Hurrell (*Camborne, Cornwall*).
Alan Michael Jane (*Camborne, Cornwall*).
Robert Day Kennedy (*Camborne, Cornwall*).
Jeffrey Kenyon (*London*).
Patrick James Lindesay Lyons (*Camborne, Cornwall*).
William James Marshall (*Wallington, Surrey*).
Tadeusz Moskwa (*London*).
James Harold Newman (*Birmingham, Warwickshire*).
James Owen Nodder (*Plymouth, Devonshire*).
Stanley James Ramage (*Uxbridge, Middlesex*).
John Edgar Reed (*Plymouth, Devonshire*).
William Valentine Rickards (*Redruth, Cornwall*).
Everard James Ross (*Birmingham, Warwickshire*).
Dimbeswar Sarma (*Leeds, Yorkshire*).
Victor John Tilly (*Camborne, Cornwall*).

Peter Leslie Vaughan (*Camborne, Cornwall*).

News of Members

Members, Associates and Students are invited to supply the Secretary with personal news for publication under this heading.

Mr. R. J. AGNEW, *Member*, is travelling to Australia and will not be returning to England until the end of June.

Mr. D. G. ARMSTRONG, *Associate*, has been appointed metallurgist to the Colonial Development Corporation.

Mr. T. W. BENNETTS, *Associate*, has returned to Northern Nigeria after leave in England.

Mr. B. BERINGER, *Member*, assistant general manager to Randfontein Estates, has been transferred to the Palmiet Chrome Mines, Ltd., as manager.

Mr. G. B. BIXBY, *Associate*, has returned to Malaya from England.

Mr. J. C. BOLSOVER, *Associate*, has returned to Tarkwa, Gold Coast, after leave in England.

Mr. F. G. BRINSDEN, *Member*, has been awarded the Medal of the Australasian Institute of Mining and Metallurgy. He has taken up residence in Perth, having retired from the active management of North Kalgurli (1912), Ltd., and South Kalgurli Consolidated, Ltd.

Mr. ALAN CAWLEY, *Associate*, has left England for the Geological Survey at Jos, Northern Nigeria.

Mr. J. T. CHAPPEL, *Member*, is returning to England at the end of the month from Perak, Malaya.

Mr. ANNAN COOK, *Associate*, has joined the staff of Kennecott Copper Corporation, New York.

Mr. G. F. V. COOPER, *Associate*, is returning to Northern Nigeria.

Mr. E. EL-ZOGHBY, *Student*, has left the United Kingdom for Cairo.

Dr. W. DAVID EVANS, *Associate*, has left University College, Cardiff, on his appointment as Professor of Geology at Nottingham University. He has recently received the Lyell Award of the Geological Society of London.

Mr. J. GOODWIN, *Student*, took up the position, on 1st January, of manager of the manganese and iron mines of Orissa Minerals Development Co., Ltd., at Bara Jamda, India.

Mr. H. L. H. HARRISON, *Member*, is coming to England, where he will be on leave from Malaya until August.

Mr. R. W. HENDERSON, *Associate*, now holds the position of resident engineer in the Rhodesias for the Cementation Co. (Africa), Ltd.

Mr. J. O. HOWELLS, *Member*, is now on a professional visit in British Columbia, where he will remain for some time.

Mr. G. R. JONES, *Associate*, has relinquished his position as manager of the General Sandur Mining Co., India, and is returning to England.

Mr. O. McCULLOCH, *Associate*, has returned to Scotland on leave from Colombia, where he was chief surveyor with Frontino Gold Mines, Ltd., and will be returning to Colombia at the beginning of May as mine agent to the Silencio mine.

Mr. E. P. MEATON, *Associate*, has joined the staff of Randfontein Estates Gold Mining Co., Witwatersrand, Ltd., as senior study assistant.

Mr. G. C. PENGILLY, *Student*, is now in England, having left Konongo Gold Mines, Ltd., Gold Coast.

Mr. D. J. ROGERS, *Member*, has now returned to Springs, Transvaal, from Tanganyika.

Mr. K. B. SWAMY, *Associate*, has been appointed Professor of Mine Surveying at Banaras Hindu University, India.

Mr. N. B. VINSON, *Associate*, has been transferred from Vlakfontein Gold Mining Co., Ltd., to the post of reduction works foreman at Venterspost Gold Mining Co., Ltd.

Mr. MARTIN WATTS, *Student*, is in Sudbury, Ontario, on the engineering staff of the Stobie mine of International Nickel Co. of Canada.

Mr. H. CARLYON WEBB, *Member*, holds the position of manager for the North African Mining Corporation in Spanish Morocco.

Mr. JOHN WEEKLEY, *Member*, has returned to England on six months' leave from Malaya.

Mr. W. BROADHEAD WILLIAMS, *Associate*, expects to arrive in England at the beginning of this month on leave from Tanganyika.

Mr. C. J. W. WILSON, *Associate*, has joined the staff of the Imshi and Birkdale mines in Southern Rhodesia.

BOOK REVIEW

The economics of mining (non-ferrous metals): valuation, organization, management. 3rd ed. By THEODORE JESSE HOOVER. Stanford, California: Stanford University Press, 1948. 551 p., illus., diagrs. $7.50.

The publication of the third edition of this well-known standard American work upon the economics of mining is a noteworthy event although it could have been wished that it had been possible for the author to revise and bring his pages up to date in a more detailed manner. The changes which have taken place in world economics since 1939 have rendered invalid, or at best of academic interest only, many of the statements made and conclusions drawn.

However, it is encouraging to read his warnings against the waste of the precious, irreplaceable mineral resources of the world, a warning which cannot be too strongly emphasized in this machine age when, as he states, the 'repair of obsolescence in the area of Western civilization alone would place a terrific strain on our reserves'. It is somewhat difficult, however, to agree that 'the famine . . . for platinum . . . is already within sight'. If an assured economical price for the metals of this group could be guaranteed their production could, in the opinion of the reviewer, be greatly increased within a few years.

As specified in the sub-title the main divisions of the book are: (1) Mine Valuation, (2) Mine Organization and (3) Mine Management. Dealing with these subjects in this order, Mr. Hoover's remarks upon the duty of an engineer are apt and to the point and it could be wished that the non-technical public had a greater understanding of the problems involved. To quote: 'Indeed, the chief usefulness of the engineer in our social organization is that he is trained to make, and that he cheerfully undertakes, these tentative valuations from scattered and imperfect data; he relies upon his own past experience and the accumulated experience of others to give him a reasonable basis for his prophecy of the future'. The difficulty with which the engineer—who should always be hoping to be able to present a favourable report upon offered properties—is faced may be realized from the fact that 'records of several exploration companies show a total of 4,093 mines offered and eight purchased, or about one in five hundred'. The necessity for a high rate of return upon a mining investment is stressed and it is pointed out that it is only the lure of possible large profits that enables capital for mining ventures to be obtained, those investors seeking security and steady return having no business to risk their money in such a speculative undertaking. The determination of the Present Value of a mine is fraught with many pitfalls and Mr. Hoover's dictum that to be satisfied with less than 10 per cent annual return on a mining investment, after allowance for redemption of capital, would show a lack of acumen, combined with his implied suggestion that the estimated life of a mine should be based only upon the developed and 'probable' ore reserves, makes one sigh for the days when such was possible. Furthermore, the fallacy of using a double rate in Present Value calculations in these days of high taxation should be pointed out. That portion of the dividend which is set aside, at, suggests Mr. Hoover, 'a savings-bank interest of 4 per cent' in order to redeem the capital expended by the end of the life of the property, would not do so where 4 per cent before tax is equal to approximately 2 per cent after tax and, furthermore, interest of savings-bank safety is more likely to be 2½ or 3 per cent than 4 per cent. It is but rarely that shares in established, sound and flourishing mining companies could be obtained at, say, 8 per cent and 1½ per cent double rate valuations. For the sake of the younger engineer it is unfortunate that the author has not radically revised Chapter 8 and brought it into line with more modern conditions, the nomograph showing ranges of from 8 per cent

... should ... when dealing ... true enough ... at a time ... and a pennyweight ... terms, but ... it is necessary to ... whether one is using a dollar ... 1/35 or 1/20 of an ... any of which may be ... in many places of Canada: ... gold mines, it would be far preferable to deal only with the ... weight ... the Troy ... warning is sounded against ... for any increase in the ... mine through the ... newly invented ... lurgical equipment ... warning ... which the ... allow to ...

... ffecting Operating Cost' the heavy expense of central administration is commented upon and illustrated by the relatively small percentage of the operating profit of a mining company available for distribution to shareholders. The author emphasizes the point that Head Office charges are as much an operating cost as the cost at the mine, and ... even though ... the manager ...

... his chapter ... Mineral Production ... to a field ... the inexperienced, ... when dealing with ... concentrates to smelting ... whilst the paragraphs in ... chapter ... in dealing with 'The ... Gold Production,' though ... or before 1933, apply to-day with as much force as they did when the first edition was published. The same remark does not apply, however, to the whole of the chapter on 'Metal Prices' which is interesting but quotes prices only up to 1930; the prophecy regarding the Burma Corporation mine might well have been omitted from the third edition of the book.

Part Two, dealing with 'Mine Organization', is of an equally high standard and is of great interest—particularly so the discussion on 'No Par Value' shares and the advice to mining share investors. The chapter on 'Fakes and Fallacies' is amusing and instructive as showing the amazing fertility of the human mind in deception and in its gullibility, and would well repay study by a very wide public.

Mr. Hoover still talks, in his discussion of the principles of scientific management in mining, of the economy of abundance although that has passed, temporarily we hope, since 1939. The quotation from Stuart Chase on page 420 is as true now as when it was written but the status of the engineer has not noticeably improved during the intervening twenty or so years. With the greatly publicized full employment the influence of the Trade Unions in preventing a man working to his best ability has increased even further than is described by the author, and the handicaps under which any attempt to increase labour efficiency suffers prevent full advantage being taken of the advances in our knowledge of management and of human psychology. The chapters on research, industrial relationships, training and safety and health are all worthy of the same careful study as the remainder of this work.

F. E. KEEP.

OBITUARY

Adam Alexander Boyd died at Brisbane, Australia, on 16th December, 1948, at the age of 79. He was born in Scotland and from 1885 to 1888 took an engineering and mining course at the Technical College, Glasgow, after which he was articled to Messrs. Dixon and Marshall, civil and mining engineers of Glasgow. In 1890 he went to New South Wales as assistant mining manager, Bellambi Colliery, and then was assistant manager and surveyor to the Newcastle Wallsend Coal Co. for five years. In 1898 he was appointed mining manager to Broken Hill Proprietary Co., Ltd., where he remained for 13 years. In 1911 he was made manager of the Newcastle Wallsend Coal Co. and two years later took up the post of mining superintendent of The Mount Morgan Gold Mining Co., Ltd., at Mount Morgan, Queensland, four years later becoming general manager. He continued in this position until the company went into liquidation in 1927, but, being convinced that the deposit was still workable, Mr. Boyd was instrumental in forming a new company, Mount Morgan, Ltd., to take over the mine and some of the assets of the original company, and he became a director of the new company. In 1929 he was a member of the Royal Commission appointed to investigate the mining industry of Queensland and was subsequently appointed to a Special Commission to the U.S.A. and Canada to report on mining conditions in those ...es. He took over the general management of Mount Morgan, ... 1932 and remained in active charge until his retirement in 1935. ...med his association with the company, however, as chairman and ...ng director from 1938 to 1941, and retained his directorship of the ...any, acting as technical adviser to the board, until his death.

...r. Boyd was elected to Membership of the Institution in 1920. He ...l been a Member of the Australasian Institute of Mining and Metallurgy ...nce 1910 and a Member of Council from 1917 to 1945, and in 1941 was awarded the Institute Medal.

Henry Ewer Jones died on 14th December, 1948. He was over 77 years of age. He attended the Royal School of Mines from 1887 to 1890, graduating with the A.R.S.M. in Mining, and in 1891 went to South Africa to join Village Main Reef Gold Mining Co., Ltd., as assayer and underground manager. After three years he left to become assistant engineer to Rhodesia Exploration and Development Co., Ltd., and was appointed manager of their Ayrshire mine in Mashonaland in 1899. From 1901 to 1912 he held the position of chief engineer of the company and of the group of mines under their control. So far as is known, he was not subsequently engaged in professional work. Mr. Jones was elected to Associateship of the Institution in 1897 and to Membership in 1901.

Reginald John Lemmon died at Nenagh, Co. Tipperary, on 26th January, 1949, at the age of 70. He was originally trained as a pharmaceutical chemist but after some years he took a short course in metallurgy at Birkbeck College, London, and in 1911 went to the Gold Coast as chief assayer to Fanti Consolidated Mines, Ltd. In 1912 he joined Abbontiakoon Mines, Ltd., and served in various capacities up to reduction officer until 1919, when he was appointed to the consulting staff of Minerals Separation, Ltd. After working for this company in the Gold Coast and Korea he went to Brazil in 1924 as superintendent metallurgist to South American Gold Areas, Ltd. In 1926–27 he redesigned and operated the Alantana reduction works in Bolivia, and in the following two years was engaged in a similar capacity in Turkey, Russia and the Transvaal. In 1929 he was appointed senior field consulting metallurgist with Imperial Chemical Industries, Ltd., and travelled in many parts of the world until his retirement in 1940.

Mr. Lemmon then took up war work, first with the Ministry of Economic Warfare, then as production manager of a Royal Ordnance Factory, and finally for three years with the Inspectorate of Fighting Vehicles. In 1946 he became reduction superintendent to Marlu Gold Mining Areas, Ltd., and at the time of his death was employed as metallurgist by the Irish Exploration Co., Ltd.

Among his published articles is a contribution to the *Transactions* of the Institution: ' The concentration of gold-copper ores by froth flotation at Tul Mi Chung, the Seoul Mining Company, Korea ' (Vol. 33, 1923–24).

Mr. Lemmon was elected to Studentship of the Institution in 1913, and was transferred to Associateship in 1919 and to Membership in 1925.

William John Smith died on 16th December, 1948, at the age of 50. He began his career with The Glyncorrwg Colliery Co., South Wales, from 1916 to 1919, gaining practical experience, and was employed for a short time with the Imperial Navigation Colliery Co., Duffryn, before serving in various departments of Messrs. Baldwins, Ltd., of Swansea, from 1920 to 1923. During these early years he studied privately and at part-time lectures and classes organized by the Glamorganshire County Council, gaining an honours certificate of the City and Guilds of London in mine surveying, the mine deputy's certificate, and the Home Office Mine Surveyor's and Colliery Manager's certificates of competency. In 1923 he became chief surveyor to the Taff Rhondda Navigation Collieries, Ltd., at Nantgarw, and, after four years, foreman and assistant to the manager of Blaenclydach colliery of the Cambrian Consolidated Collieries, Ltd., Llwynypia, South Wales.

In August, 1928, Mr. Smith was appointed chief surveyor of the manganiferous iron ore mines of the Sinai Mining Co., Ltd., at Om Bogma, Egypt, and in 1931 was promoted to the position of general manager. This he held for fourteen years until March, 1945, when he took up the appointment of consulting mining engineer to the Northern Mercantile and Investment Corporation, Ltd., London.

Mr. Smith was elected to Associateship of the Institution in 1930 and was transferred to Membership in 1942.

Frederick Harold Williams died in Hove Hospital on 18th January, 1949, at the age of 69. He was born at Alexandra, Otago, New Zealand, and was educated at Alexandra High School. In 1897 he became a mining cadet with Manorburn Gold Dredging Co. at Otago, and after three years joined the staff of Little River Dredging Co. and Gungarline River Dredging Co. in New South Wales, Australia. He went to South America in 1905 as manager of Rio del Oro Gold Dredging Co. in Tierra del Fuego, and after four years left to travel in Sumatra and Central Borneo in charge of a prospecting expedition for the General Exploration Co. of The Hague from 1909 to 1911. In September, 1912, Mr. Williams was appointed general manager of Fuego Gold Syndicate, Tierra del Fuego, but returned to England in 1915 to serve with commissioned rank with the Royal Engineers in France, and was mentioned in despatches. He was demobilized in 1919, and returned to South America as assistant general manager and, later, general manager of Fuego Development Co. He left in 1924 to go to Siberia as general manager of Vint Eastern Siberian Goldfields, Ltd., but resigned after a year and was employed from 1925 to 1928 as mining representative in Malaya and Siam of the Borneo Co., Ltd., subsequently becoming acting general manager of Tavoy Tin Dredging Corporation. He returned to England in 1929 and was engaged on consulting work for the South Andes Syndicate and the Colombis Syndicate. In 1932 he joined the staff of the West African Gold Corporation and on the formation in 1937 of the Bremang Gold Dredging Co. he was appointed general manager, and held this position until his resignation in 1942 on account of ill health. His services were retained in a consultative capacity up to the time of his death, and during 1946–47 he was consultant to the Ministry of Supply on the rehabilitation of the Malayan tinfields.

Mr. Williams was elected to Associateship of the Institution in 1930 and was transferred to Membership in 1943.

The Council regret to report the death of **David Cinamon,** *Associate*, on 12th January, 1949 ; **Robert Henry Jeffrey,** *Member*, on 31st January, 1949 ; and **Walter George Woolston,** *Associate*, on 12th February, 1949. Obituary notices will be published in a later issue of the *Bulletin*.

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3 ECONOMICS

336.2 Taxation

336.2
Mining taxation. (Memorandum by the British Overseas Mining Association.)—*Min. World,* Lond., **156**, Jan. 29 1949, 59. 6d.

336.2(72)
Mining taxes in Mexico. J. Kellogg Burnham.—*Miner. Tr. Notes,* Wash., D.C., Spec. supp. 25, **26**, April 1948, 19 p.

53 PHYSICS

55 (...) Regional

537.533.72
The electron microscope. A. D. Merriman. —*Metallurgia,* Manchr., **39**, Jan. 1949, 139–43. 2s.

55 GEOLOGY

55(569.4)
The geology and mineral resources of Palestine. S. H. Shaw.—*Bull. Imp. Inst.,* Lond., **46**, no. 1, 1948, 87–103, illus., maps, tabs. 2s. 6d.

55(831.7)
Geology and ground water of the Casablanca basin, Chile. George C. Taylor.—*Econ. Geol.,* Lancaster, Pa., **43**, Dec. 1948, 661–74. $1.

55(85)
Geological reconnaissance of the Contamana region, Peru. Bernhard Kummel.—*Bull. Geol. Soc. Amer.,* Baltimore, Md., **59**. Dec. 1948. 1217–66, maps, biblio.

550.8 Prospecting

550.8(679)
Mineral prospecting in Moçambique. *Min. J.,* Lond., **232**, Jan. 29 1949, 79. 8d.

550.8 : 778.35
Photogeology—a tool for mineral development. (Geological interpretation of aerial photographs for prospecting.) Robert C. Stephenson and Richard M. Foose.—*Min. Congr. J.,* Wash., D.C., **34**, Nov. 1948, 39–42; 96, diagrs. 30 cents.

550.837.3
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550.84
A sensitive field test for heavy metals in water. (Semiquantitative colorimetric analytical method using dithizone.) Lyman C. Huff.—*Econ. Geol.,* Lancaster, Pa., **43**, Dec. 1948, 675–84. $1.

553 ECONOMIC GEOLOGY

553 (...) Regional

553(569.4)
The geology and mineral resources of Palestine. S. H. Shaw.—*Bull. Imp. Inst.,* Lond., **46**, no. 1, 1948, 87–103, illus., maps, tabs. 2s. 6d.

553(672/69)
The richest part of the earth's crust; a belt from Congo to mouth of Orange river. (Location of minerals, production and value figures.)—*Min. Industr. Mag.,* J'burg., **38**, Nov. 1948, 597–601, map. 1s.

553.2 Ore deposition

553.2 : 553.492.1
Formation of bauxite from basaltic rocks of Oregon. Victor T. Allen.—*Econ. Geol.,* Lancaster, Pa., **43**, Dec. 1948, 619–26. $1.

553.21
Transport and deposition of the non-sulphide vein minerals: 3-phase relations at the pegmatitic stage. F. Gordon Smith.—*Econ. Geo.,* Lancaster, Pa., **43**. Nov. 1948, 535–46, biblio. $1.

553.21
The direction of flow of mineralizing solutions. A. E. J. Engel.—*Econ. Geol.,* Lancaster, Pa., **43**, Dec. 1948, 655–60, biblio. $1.

553.216.7

The decrepitation method applied to minerals with fluid inclusions. (The escape of fluid intrusions, liberated by heating, is recorded by its audible rather than visible effects.) H. S. Scott.—*Econ. Geol.*, Lancaster, Pa., 43, Dec. 1948, 637–54. $1.

553.232(713.712 McIntyre)

The ore deposition temperature and pressure at the McIntyre mine, Ontario. F. Gordon Smith.—*Econ. Geol.*, Lancaster, Pa., 43, Dec. 1948, 627–36. $1.

553.3/.4 Metals

553.31 : 553.464

Vanadium in iron ores consumed by the British iron and steel industry. T. Deans.—*Bull. Imp. Inst.*, Lond., 46, no. 1, 1948, 104–10. 2s. 6d.

553.311 : 553.2

Titanic iron ore formed by dissociation of silicates in granulite facies. Hans Ramberg.—*Econ. Geol.*, Lancaster, Pa., 43, Nov. 1948, 553–70, biblio. $1.

553.311 : 553.2

The supposed oxidation of Fe_3O_4 by carbon dioxide. J. B. Hawley and S. C. Robinson.—*Econ. Geol.*, Lancaster, Pa., 43, Nov. 1948, 603–9, biblio. $1.

553.32(94)

Manganese (Australia). C. L. Knight.—*Miner. Res. Aust. Sum. Rep.* no. 7, Canberra, Feb. 1947, 35 p., maps, tabs. biblio.

553.411(711.15 Cariboo Gold Quartz)

Geology of the Cariboo Gold Quartz mine, Wells, B.C. A. C. Skerl.—*Econ. Geol.*, Lancaster, Pa., 43, Nov. 1948, 553–70. $1.

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Lead (Australia). N. H. Fisher.—*Miner. Res. Aust. Sum. Rep.* no. 23, Canberra, Jan. 1947, 44 p., maps, tabs., biblio.

553.446(94)

Zinc (Australia). N. H. Fisher.—*Miner. Res. Aust. Sum. Rep.*, no. 33, Canberra, Feb. 1947, 30 p., maps, tabs., biblio.

553.446.3(781)

Observations on sphalerite in southwestern Kansas. David LeCount Evans.—*Econ. Geol.*, Lancaster, Pa., 43, Dec. 1948, 685–7. $1.

553.449(94)

Cadmium (Australia). C. J. Sullivan.—*Miner. Res. Aust. Sum. Rep.*, no. 21, Canberra, Oct. 1947, 44 p., maps, tabs., biblio.

553.464 : 553.31

Vanadium in iron ores consumed by the British iron and steel industry. T. Deans.—*Bull. Imp. Inst.*, Lond., 46, no. 1, 1948, 104–10. 2s. 6d.

553.487(94)

Cobalt. P. B. Nye.—*Miner. Res. Aust. Sum. Rep.*, no. 32, Canberra, March 1947, 21 p., maps, tabs., biblio.

553.492.1 : 553.2

Formation of bauxite from basaltic rocks of Oregon. Victor T. Allen.—*Econ. Geol.*, Lancaster, Pa., 43, Dec. 1948, 619–26. $1.

553.497(825.6 Cochinoca)

An unusual antimony deposit in Argentina. (Cochinoca area, Province of Jujuy.) Federico Ahlfeld.—*Econ. Geol.*, Lancaster, Pa., 43, Nov. 1948, 598–602. $1.

553.6 Non-metals

553.64(94)

Phosphates (Australia). T. D. Dimmick.—*Miner. Res. Aust. Sum. Rep.*, no. 29, Canberra, Feb. 1947, 34 p., maps, tabs., biblio.

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622.006.2

The role of research in the development of labor-saving equipment and method. E. R. Borcherdt.—*Min. Congr. J.*, Wash., D.C., 34, Nov. 1948, 32–5; Dec. 1948, 49–52.

622.1 Topography, surveying

622.1 : 526.88

Tri-dimensional representation; the anaglyph.—*Univ. Durh. King's Coll. Min. Bull.*, *Durh.*, 2, no. 5 (Ser.: Surv., no. 2), 2 p., illus.

622.2 Development, exploitation

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The mechanical movement of muck; a symposium conducted by the Porcupine Branch. (Shovel loader and mechanical scraper operations. Tabulated data of operations and performance.)—*Canad. Min. Metall. Bull.* 439, Montreal, Nov. 1948, 629–35, tabs. $1.

622.23 Machine mining, drilling

622.235.41

Delayed-action detonators in the driving of stone headings at Thoresby colliery. H. Langford Holt.—*Trans. Instn. Min. Engrs.*, Lond., 108, Pt. 1, Oct. 1948, 37–46, diagrs. 10s.

622.25 Shafts, sinking

622.25

Grundlagen der Schachtberechnung. Fritz Schultz-Grunow.—*Gluckauf*, Essen, 84, Dec. 4 1948, 835–44, diagrs. 4 DM.

622.27 Mining methods

622.271.3(776.72)

Dragline and conveyor stripping at the South Agnew mine; stripping glacial drift ranging in depth from 80 to 175 ft. and containing large boulders. Earl S. Mollard.—*Min. Congr. J.*, Wash., D.C., 34, Dec. 1948, 31–5, illus., diagrs., tabs. 30 cents.

622.273.1

Vergleichende Erfahrungen mit Strebbau und Pfeilerbruchbau beim Abbau eines Doggererz-Flözes. Hans Bornitz.—*Gluckauf*, Essen, 84, Dec. 4 1948, 845–8, diagrs. 4 DM.

622.273.23

Vergleichende Erfahrungen mit Strebbau und Pfeilerbruchbau beim Abbau eines Doggererz-Flözes. Hans Bornitz.—*Gluckauf*, Essen, 84, Dec. 4 1948, 845–8, diagrs. 4 DM.

622.276 : 622.33

Oil and gas wells drilled through workable coal seams. Arch J. Alexander.—(*Amer. Inst. Min. Engrs. Tech. Pub.* 2430) *Coal Technol.*, N.Y., 3, Nov. 1948, 4 p. $1.65.

622.34 Metalliferous mining

622.341.1(435.3)

Der Aufschluss und Abbau der Salzgitterer Erze nach neuzeitlichen Gesichtspunkten. Heinz Prause.—*Z. Erzberg. Metallh.*, Stuttgart, **1**, May 1948, 33–42, diagrs., tabs. 3 RM.

622.343(791.2 San Manuel)

Exploration and development of the San Manuel orebody; churn drilling disclosed extensive copper orebody minable by block caving. Wesley P. Goss.—*Min. Congr. J.*, Wash., D.C., **34**, Dec. 1948, 22–6, illus. 30 cents.

622.349.46(94)

Australia's lithium deposits await development; mineral has wide uses in glass making. Charles Lynch.—*Min. J.*, Lond., **232**, Jan. 29 1949, 78–9. 8d.

622.349.72(682.6 Murchison)

Mining and milling antimony ore at Consolidated Murchison Goldfields, Transvaal. Ralph Symons.—*Bull. Instn. Min. Metall.*, Lond., no. 506, Jan. 1949, 1–36. (Paper 2s. 6d.)

622.37 Precious stones

622.37 : 549.211(688)

The Orangemouth diamond workings; developments at the C.D.M. (Consolidated Diamond Mines) property.—*S. Afr. Min. Engng. J.*, J'burg, **59** (Pt. 2), Dec. 18 1948, 453–5. 6d.

622.41/6 Ventilation

622.41 : 622.016.8

Ventilation; a limiting factor in deep mining. J. M. Beskine.—*Mine & Quarry Engng.*, Lond., **15**, Feb. 1949, 51–6, biblio. 1s. 6d.

622.457.5

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Subject to revision.] [*A Paper published on the* 10*th March*, 1949, *to be submitted for discussion at a Meeting of the Institution of Mining and Metallurgy, to be held in the Apartments of the Geological Society of London, Burlington House, Piccadilly, London, W.* 1, *on* **Thursday, 17th March, 1949,** *at* 5 *o'clock p.m.*

Geophysics and Economic Geology*

By J. McG. Bruckshaw,† Ph.D., M.Sc., D.I.C.

Introduction

In a recent Symposium on Mining Geophysics (1) in America it was pointed out that about 100 metallic minerals and 100 non-metallic minerals were of commercial importance. Although it is customary to divide geophysical prospecting into four main methods, each method involves several techniques. Thus the subject under discussion is one of considerable breadth and variety and to examine at length all the different aspects is clearly impracticable. Further, it is impossible to formulate a set of infallible rules governing the application of geophysical processes to the problems of mining geology. Indeed, each particular problem must be examined in its own environment to determine which, if any, of the field techniques is suited to assist in its solution. Accordingly, an attempt is made to outline certain general guiding principles controlling the use of these field techniques, while considering the limiting factors involved.

It is well known that the main sphere in which geophysical methods have been used is that of oil prospecting; here many substantial and outstanding successes have been obtained. Elsewhere these methods have been used on a much smaller scale and, although much useful information has been produced, spectacular results are rare. The reason for this is not far to seek. Fundamentally, the search for oil is reduced to the location of a few different types of geological structure, usually in sedimentary rocks, and although these may be deep seated, they have, in general, a considerable lateral extent. In mining problems the geology is considerably more complex, involving rocks of all types, and the ratio of size to depth of the economic deposit is usually small. In general it has been possible to standardize in oil prospecting, with seismic investigation for detailed studies, gravity measurements as a reconnaissance, and magnetic surveying as an auxiliary. The changing nature of the problems of economic

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(1)See list of references at end of paper.

geology, in its wider sense, precludes such an approach. Finally, the cost involved in geophysical prospecting for oil would be prohibitive in any other sphere.

The Geophysical Methods

It will be convenient to summarize the essential factors involved in the geophysical prospecting methods before discussing their application. In the first place it is necessary to appreciate that a geophysical survey involves the measurement, usually at different points on the surface of the earth, of certain physical quantities. The particular physical quantities so observed are influenced in some way by changes in one physical property of the rocks and minerals below the surface. Only four such physical properties have been found to be of general use in prospecting, and these give rise to the four better-known methods. The variation in the density of the rocks leads to gravitational methods, in the magnetic properties to magnetic methods, in the electrical conducting properties to the electrical group, and, finally, the variation in the speed of propagation of shock waves through the rocks, dependent on the individual densities and elastic properties, to seismic methods. Seismic methods are not well adapted to the type of problem involved here, and will not be considered at any length.

The gravitational method of prospecting depends upon the universal Newtonian attraction between all matter. The force of gravity, which is the attraction between the earth and unit mass at the surface, is obviously dependent on the distribution of mass within the earth. If, in particular, there exists in the earth's crust near the surface an excess mass, due to the presence of a rock or mineral of greater density than the surroundings, then at the surface the value of gravity will be high. A deficiency of density, on the other hand, will give a low value of gravity. The local changes in gravity from this cause are small, at best a few parts in a million, but nevertheless they may be examined by the gravity meter or the Eötvös torsion balance. The former can be used to measure the disturbed gravitational field directly by measurement of gravity differences, while the latter measures how rapidly gravity varies with position, the measurements being shown on maps by arrows pointing from regions of low towards regions of high gravity (see Fig. 6, p. 14). The gravity distributions, depicted by the use of these instruments, are directly related to the size and depth of the disturbing body and to the density difference between it and the surrounding body. Before the observations can be used to make deductions concerning the local geology, however, certain corrections have to be applied, especially those for elevation and topography. One limitation of the Eötvös torsion balance arises here, since the correction for rugged topography cannot be estimated with sufficient accuracy. As will be shown, the direct measurement with the gravity meter does not suffer so seriously from this defect.

Nearly all materials have the property of acquiring temporary magnetic characteristics, to a greater or less extent, when placed in a magnetic field. In addition to this induced magnetism, many possess permanent magnetism. Most rocks exhibit these characteristics which, when pronounced, are usually related to the magnetite content of the rock. Rocks, accordingly, are buried magnets, often with complex polarity arising from the combined permanent and induced magnetism. As such they possess their own magnetic field, which is superimposed on the earth's normal field. In their vicinity then anomalous magnetic fields exist and these, if sufficiently great, can be observed at the surface by a variety of magnetometers, including airborne instruments. Once again, there is a direct relation between the geometry of the body, its magnetic properties, and the field observed at the surface. Here, however, the magnetism is not a characteristic of the body, for the induced part depends on the interaction between it and the earth's field.

The electrical methods are more complex, and include the study of natural currents set up in the vicinity of deposits undergoing active oxidation with the production of electro-chemical activity, the investigation of the distribution of artificial currents passed conductively through the ground or of currents induced in the ground. Further, the distribution can be investigated by measurements of voltages at the surface or of the magnetic field accompanying the currents. In spite of this apparent multiplicity of methods, the principles involved are relatively simple and, apart from the study of the natural currents, all depend on the electrical conductivities of the rocks. The artificial currents always tend to concentrate in good-conducting regions and avoid badly-conducting regions. When the method of exciting the ground is fixed, then the resulting current distribution depends only on the rock resistivities.

General Considerations

A closer examination of these methods demonstrates that they all suffer from one serious disadvantage. Although from a given density distribution, from the magnetic properties, or from the electrical resistivities, it is possible to predict (at least in theory) the precise nature of the field observed at the surface, the reverse process is far from true. For every gravity anomaly, magnetic anomaly, etc., it is possible to produce an infinite number of distributions of the physical property which are capable of fitting exactly the observed values at the surface. Further, the picture obtained is one of differences of gravity, differences of magnetic property, or of resistivity ratios. Except in rare cases it is impossible to identify a mineral or rock from a knowledge of density, magnetic property, or electrical resistivity, so there can be no hope of direct identification of the rocks involved from data supplied by the ambiguous interpretation of a geophysical survey.

It is apparent that the geophysical data alone are insufficient to allow the solution of any particular problem; they must be supplemented by other evidence, in the form of all the relevant geological information available. Any solution which satisfies both geophysical and geological observations must be regarded as reasonable, but even here there may be more than one. Frequently, on the basis of such tentative solutions, further exploratory work may be undertaken. The additional evidence allows some solutions to be discarded, while others may be modified or adapted to fit. At no stage can the geophysical work be regarded as an independent investigation; for the best advantage to be taken it must form an integral part of the total survey, proceeding in step with other methods of attack.

The identification of a rock or mineral from geophysical data is not possible, although a knowledge of the geology of an area may substantially reduce the possibilities. In rare cases more than one method of prospecting can be used to assist in this problem. The question of separating good-conducting metal sulphides from other conductors, of no economic importance, by using their high density was discussed before this Institution in 1938. ([2]) Their electrochemical activity is another possible test, or a magnetic investigation on associated magnetite. Although a combination of methods is an advance, the identification is not positive. Graphitic schists and shales may be conducting, but the former may give electrochemical activity and the latter may be appreciably magnetic.

The questions set by the mining engineer are: Where, how big, how deep, and what is the quality of the deposit? The geophysicist, however, cannot give precise replies to these questions and no attempt can be made to answer the last of them for reasons already given. Further, because of the ambiguity in the interpretation, the answers to the others are necessarily vague, but may become better defined when allied to the relevant geological data. Many minerals occurring in sufficient concentration can be detected because of their intrinsic physical properties. Some base metal sulphides, oxides, and others are of high density in comparison with the surrounding rocks, a number possess outstanding electrical properties, while magnetite, ilmenite, some types of pyrrhotite, manganese ores, etc., have sufficient magnetic influence to allow direct detection. In other problems the mineral of economic importance may be in small concentration, or may not possess any outstanding characteristic by which its presence may be revealed. In certain cases, however, associated minerals may allow an indirect attack. Even a small percentage of magnetite produces a marked magnetic effect, or the presence of pyrite may give conducting properties or evidence of electrochemical activity, while, in the case of certain disseminated minerals, the gangue material may be of importance.

Yet another approach, even more indirect, is by using geological structures. It is well known that the deposition of many ore minerals is controlled by the geology of the region, and hence a delineation of the structure of the region is an essential step in their location. Mineralization is frequently associated with faults, or contacts between different formations, or may be structurally related to igneous bodies. On the other hand, the ore minerals may be confined to one geological horizon which is folded and faulted. In the case of placer deposits the examination of an older buried topography is significant. This mode of attack, although it gives no specific information concerning the presence of the mineral, only indicating the positions most favourable for its occurrence, is possibly the most promising. Indeed, it is this particular approach which has been found most successful in the search for oil. It should be noted here that, with this procedure, the relevant geological data are of paramount importance.

These three methods of attack do not exhaust all the relevant geological conditions of ore occurrence. Bodies are known whose attitudes bear little relation to the local geological structure and, if they do not possess with their associated minerals any noteworthy physical characteristic, then they are outside the scope of the geophysical methods at the moment.

In assessing the possibilities of any proposed geophysical survey three main issues have to be considered. The first is magnitude of the response from the feature to be investigated. Secondly, and of equal importance, is the possibility of any other body producing an effect of equal magnitude, which might confuse the interpretation. Finally, there is the question of the general background against which the observations must be made. Although, in any area, there may be only one or two major contrasts in any selected physical property, there will be many smaller ones which will contribute to the measurements and produce a varying background. The magnitude of these usually erratic fluctuations imposes a limit to the magnitude of response which can be detected. Any signal smaller than the fluctuations will be confused with the apparently random variations encountered. The background is a special case of the second factor raised, for here a large number of different contrasts are producing roughly equal anomalies and there is no method of separating them and assigning them to their origin.

Many of the points which have been raised in this general discussion are best appreciated in terms of examples, and a number of surveys has been selected with this object and described briefly in what follows. In addition a number of factors, peculiar to each method, are discussed.

The Magnetic Methods

The magnetic anomaly shown in Fig. 1 forms a good illustration of the type of procedure used in interpretation. The anomaly, situated in Estonia, is shown by lines of equal vertical component and these take the form of a series of roughly elliptical closed

rdingly, the geophysical interpretation results in the depth of pper pole and the horizontal length over which it is distri-d. It was found that a line pole 345 m. deep and 960 m. with a pole strength of 4,140 c.g.s. units per cm., was required produce the observations. Small differences between the lated anomaly and the observed anomaly along a line perpen-lar to the strike suggested that the dip was about 15° from the ical, conforming with banding observed on core specimens.
he length of the pole is obviously related closely to the strike th of the magnetic bed, but as the pole of a magnet is usually in the body and not exactly at its end, the depth is an over-mate. The width of the body requires a knowledge of the gnetic properties of the deposit. The bore-hole specimens were nd to be permanently magnetized and of variable intensity, based on an average intensity a body of width 40 m. was uired to fit the surface observations. The combined evidence ds to the section shown in the sketch, with the magnetite ountered in the bore-hole insufficient to account for the anomaly. extension of the drill-hole was justified and, on samples tained between 264 m. and 383 m., further measurements of gnetic intensity were obtained and the magnetite content deter-ned. These were sufficient to account for about half of anomaly. As the bore-hole would intersect the mid-plane the bed at 382 m. this appears satisfactory. The iron content, fortunately, was low, varying from 10 to 45 per cent, but there s no reason to believe that the tenor in the rest of the body uld improve materially.

This example of a simple nature has been discussed at some gth to illustrate how the final picture is built up not by geo-ysics alone but by combining all the data, both geophysical and logical. The initial values of the magnetic field are of little mediate use except to select a bore-hole site. In the case of a re complex magnetic field even this would require careful nsideration, taking into account all the known geology.

This magnetic example is outstanding in its magnitude, but y much smaller anomalies have been examined as, for example, the search for haematite in Cumberland (³) just before and ring the early years of the War, in which maximum anomalies 20 to 50 *gammas* were contemplated. With small anomalies of s size, the background field may be of great importance, since er causes, mainly geological in origin, may mask the influence the orebody or the feature sought. Thus, a basic igneous rock h an irregular upper surface covered by a variable thickness of, , drift might well lead to an erratic magnetic field. A second se is due to the permanent magnetism of some igneous rocks, ich, for some unknown reason, may change its direction in a st erratic manner. The surface of such a rock will show an parently irregular distribution of poles, each producing its own al anomaly, and the aggregate may mask any feature below.

curves. The magnitude of the anomaly is great, changing by 19,400 *gammas* from the normal field (47,300 *gammas*) to the maximum, and covers in all an area of several square kilometres. The presence of intensely magnetized rocks cannot be disputed and this is one of the rare cases where the cause can be assigned to magnetite as the only natural mineral occurring in sufficient concentrations to yield a disturbance of this magnitude. Further progress in the useful interpretation is impossible without geological data, since there are many distributions which would yield this particular anomaly.

This information was provided by a drill-hole to a depth of 264 m. situated about 10 m. to the north of the position of the maximum anomaly. To a depth of 238 m. this passed through Silurian limestones, Cambrian sandstones, and clays; followed by a hard skarn rock, which, 20 m. deeper, showed about 30 per cent of iron as magnetite. A detailed examination revealed this to be

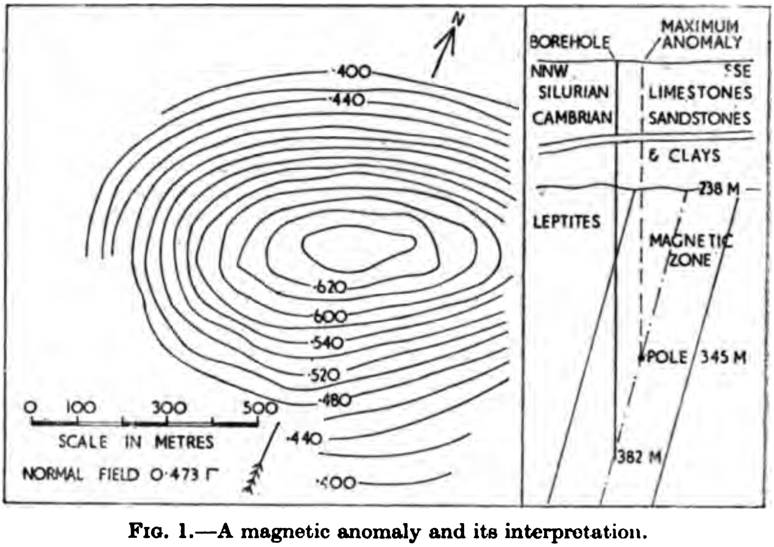

Fig. 1.—A magnetic anomaly and its interpretation.

a highly-metamorphosed chemical sediment interbedded with leptites, the recrystallization having occurred at considerable depth owing to regional metamorphism. Further, the stratification appeared to be an original characteristic.

On this basis it appeared reasonable to assume that the magnetite formation took the form of a dipping sheet of finite strike length. If this is correct then the strike direction is given by the major axis of the iso-anomaly curves—i.e., NNE.–SSW.—and it appears to be steeply dipping. Further, the field changes more rapidly towards the NNW., which is the probable direction of dip. As the geophysical survey does not show any negative anomaly the lower magnetic pole of the sheet appears to be at a great depth.

Accordingly, the geophysical interpretation results in the depth of the upper pole and the horizontal length over which it is distributed. It was found that a line pole 345 m. deep and 960 m. long, with a pole strength of 4,140 c.g.s. units per cm., was required to reproduce the observations. Small differences between the calculated anomaly and the observed anomaly along a line perpendicular to the strike suggested that the dip was about 15° from the vertical, conforming with banding observed on core specimens.

The length of the pole is obviously related closely to the strike length of the magnetic bed, but as the pole of a magnet is usually within the body and not exactly at its end, the depth is an overestimate. The width of the body requires a knowledge of the magnetic properties of the deposit. The bore-hole specimens were found to be permanently magnetized and of variable intensity, but based on an average intensity a body of width 40 m. was required to fit the surface observations. The combined evidence leads to the section shown in the sketch, with the magnetite encountered in the bore-hole insufficient to account for the anomaly. An extension of the drill-hole was justified and, on samples obtained between 264 m. and 383 m., further measurements of magnetic intensity were obtained and the magnetite content determined. These were sufficient to account for about half of the anomaly. As the bore-hole would intersect the mid-plane of the bed at 382 m. this appears satisfactory. The iron content, unfortunately, was low, varying from 10 to 45 per cent, but there was no reason to believe that the tenor in the rest of the body would improve materially.

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In general it is theoretically possible to locate and trace basalt-filled deep leads below a flow, but irregular magnetization on a suitable scale would render the observed field too confused to be interpreted satisfactorily. Measurements on orientated specimens may reveal the possibility of such effects, but, unless they are collected systematically at the surface and in depth, their influence is difficult to estimate. Many cases of altered permanent magnetization are due to lightning, but these give a purely local influence which may be removed from the general picture. Accordingly, although the existence of such conditions may be suspected, it is difficult to forecast their importance, which can only be revealed by preliminary observations on the site. In simple cases a great diversity in magnetic intensity and direction is not a barrier to magnetic prospecting for, in tracing a magnetic body in non-magnetic rocks, any anomalous observations would be indicative of its presence. The magnetic profiles in Fig. 2, one

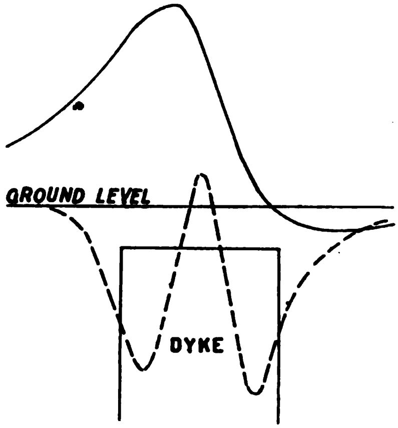

FIG. 2.—A comparison of the shapes of the calculated magnetic profile across a dyke (continuous curve) and the measured profile (broken curve).

estimated for a known dyke and the other the observed values over it, illustrate this point.

The number of common minerals having any significant magnetic properties is small, but in sufficient concentration they can be located directly if the ratio of size to depth of deposit is satisfactory. Frequently, the association of magnetite or pyrrhotite with the ore may permit a direct examination of the body as in the case of the Sudbury nickel deposits. [4] More generally, however, the method is best suited to the examination of igneous rocks and their contacts with non-magnetic rocks, a problem of considerable importance in many mineral deposits. Here the approach is very indirect and leads to a better appreciation of the local geology.

As an example of the indirect use of the magnetic method, its application to the estimation of the position of the Main Reef in the Witwatersrand area may be cited. The reef itself has no magnetic properties, and its approximate position is obtained by locating a magnetic shale whose attitude is conformable with it (Fig. 3). The origin of this method of approach is due to Krahmann, who demonstrated the feasibility of tracing the shale through a considerable thickness of younger sediments resting unconformably on the Witwatersrand Series. From a knowledge of the sub-outcrop of the marker bed, and its dip as determined from the magnetic profile at the surface, the position of the reef was inferred. This method has been discussed at length by Weiss ([5]), with special reference to the errors arising in it. The zone of magnetic shales is thick and the magnetite content, both across and along its strike, by no means uniform. The variable magnetization leads to errors in its estimated dip and in its precise location, and the

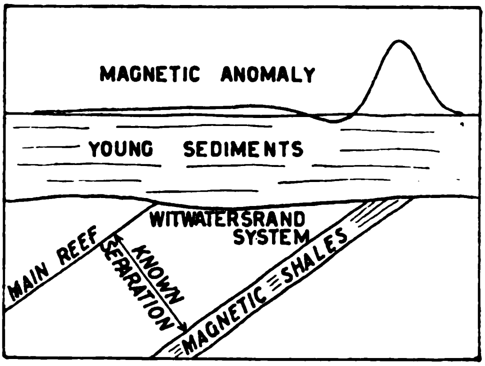

Fig. 3.—Sketch illustrating the indirect method of locating the Main Reef.

picture may be confused by any erosional feature at the top of the Witwatersrand Series which will displace the sub-outcrop. It is obvious that the importance of these errors must be assessed in the interpretation. It is now a matter of history that, in spite of these difficulties, very valuable information was obtained by work of this nature.

Summarizing, the magnetic method is mainly indirect in its application. As far as depth of investigation is concerned this is entirely a question of ratio of size to depth, together with the difference in the magnetic properties. Basement uplifts at depths of two miles have been detected and examined by the method.

It should be noted that the interpretation of magnetic anomalies is more difficult than that of gravity anomalies, to which they may be related. A gravity anomaly is a characteristic of the buried feature, but, as already noted, the magnetic anomaly depends upon the interaction of the body and the earth's magnetizing field. As a simple example, a vertical dyke at the earth's magnetic north pole would give a simple symmetrical peak if magnetized by induction only. The same dyke running E.–W. at the magnetic equator would yield a positive anomaly to the south and a negative anomaly to the north, while with a N.–S. strike it would give no anomaly whatsoever. Superimposed on these features would be the influence of any permanent magnetism, which sometimes dominates the picture, changing entirely the characteristics of the observed field (Fig. 8).

Magnetic surveying is one of the simplest and quickest methods of geophysical prospecting. The instruments are cheap and skill in their manipulation is readily acquired. Without the necessary experience, however, the interpretation may not prove satisfactory. Over small areas the older form of vertical or horizontal variometer is adequate, but for extensive reconnaissance surveys the airborne magnetometer is incomparably quicker and, ultimately, cheaper if the area involved is large enough. Another advantage of air surveying is that the background irregularities, usually due to very near surface features, are reduced. If the feature sought is below these irregularities its anomaly, although also reduced in magnitude, bears a greater ratio to the background noise and is more readily appreciated. In addition, field measurements at two different levels may assist the interpretation, particularly in relation to depth estimates.

Gravity Measurements

When gravity prospecting was first introduced, some attempts were made to utilize the method in other spheres than the search for oil. In general, however, it was not found very satisfactory, since the Eötvös torsion balance, at that time the only sufficiently sensitive instrument, was slow in operation and very sensitive to the topography of the area. As a result the effects of the hidden geology were masked completely by the irregular background produced by surface level differences and it was impossible to compute their influence with sufficient accuracy. This type of instrument is, for most practical purposes, obsolete and has been replaced by the gravity meter, which examines the anomaly in a more direct manner than the torsion balance. As these instruments measure gravity differences in place of the rate of increase of gravity, they are very much less susceptible to the nearby surface irregularities, and modern instruments only require a few minutes at any station to make the necessary measurement. The latest instruments also have a sufficiently high sensitivity to detect anomalies which are of interest to the mining geologist, and so the gravitational method of prospecting has changed from an interesting, but generally impracticable, tool to one of potential importance.

Gravity anomalies have one advantage over other types of anomalies because they are more directly related to the mass of the orebody. It is not, however, related to the total mass, but only the excess mass—i.e., the volume of the body multiplied by the excess (or deficiency) of density. The best of modern instruments can just detect 0·02 *milligal** change in gravity, and hence an anomaly of 0·1 *milligal* represents about the limit which can be investigated. On the basis of this the two graphs in Fig. 4 show

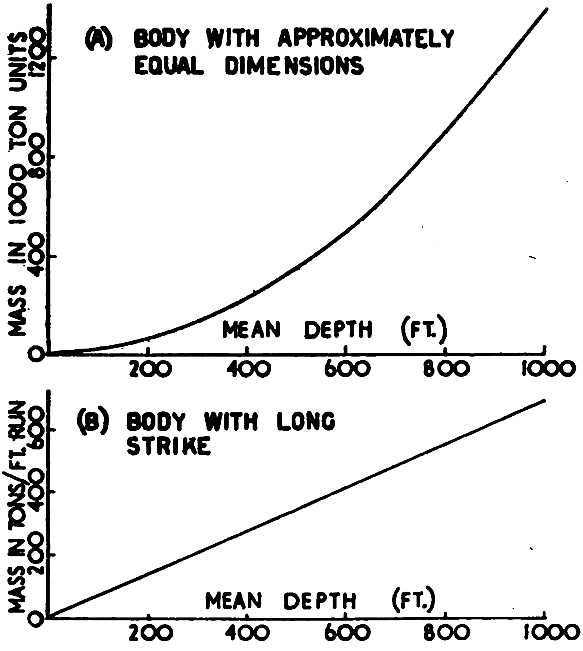

FIG. 4.—Graphs showing the excess mass at different depths required to produce an anomaly of 0·1 *milligal.*

the excess mass which can just be examined at different depths. The gravity anomaly is, however, not only dependent upon the mass, but also upon its distribution, so that the values given can only be taken as a general guide. In one case the figures refer to a body in which the dimensions are nearly equal, when the maximum anomaly is obtained for a given mass, and the second refers to a body with a long strike length in comparison with its cross-sectional dimensions and depth.

One of the first direct gravity meter surveys for ore was carried out during 1941 in the search for chromite in Cuba. (6) The chromite, of density 4, contrasts well with the associated serpentine (density 2·5) and even disseminated chromite left by weathering of the serpentine gave a density difference of 0·9. There were, however, other significant density contrasts between the basic igneous rocks, their weathered products, and the surface soils.

*A change of a *milligal* in gravity is equivalent to a change of 0·001 cm./sec.^{2} in the total value of about 981 cm./sec.^{2}.

The gravity observations were made to the nearest 0·01 *milligal* at intervals of 20 m., and the necessary surveying of topographical detail was carried out with great care. Every precaution was taken to obtain the highest accuracy.

Initial tests over a known orebody were encouraging and gave an unmistakable anomaly with a maximum of 0·35 *milligal*. Part of the results of the extended survey are depicted in Fig. 5, which

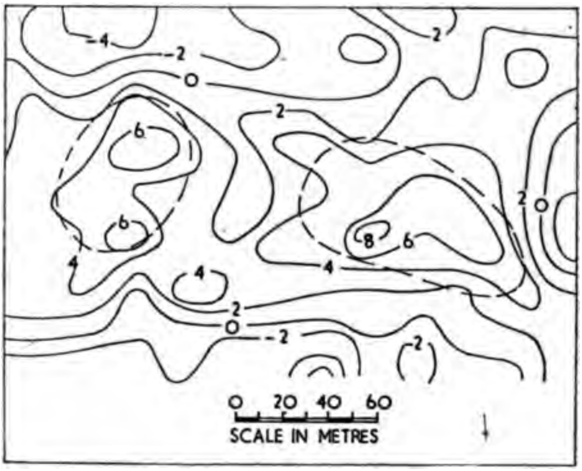

FIG. 5.—Small gravity anomalies over chromite deposits in Cuba. (After Hammer, Nettleton and Hastings, *Geophysics*, **10**, 1945.)

shows the residual gravity values after all corrections, including the removal of a large regional gradient, had been made. In addition to some negative anomalies and a number of small positive ones, the plan reveals two with maxima of 0·07 and 0·08 *milligal*, respectively, and of reasonable lateral extent. Subsequent drilling proved that these were associated with the same chromite deposit having a thin section joining the major parts, which exist immediately below these anomalies. Other anomalies, larger in areal extent and gravity value, were also located, but they proved to be due to other density contrasts. One, for example, proved to be a compact anorthosite encountered at a depth of 5 ft. and another was due to gabbro. A small exposure of gabbro occurred in the area of the latter anomaly and suggested its cause. The gravity meter is thus capable of detecting shallow bodies of this nature, but is incapable of distinguishing them from anomalies produced by other unwanted bodies. Nevertheless, there is no point in shallow drilling in regions where high gravity values do not occur.

More recently a paper by Romberg, Mathis and Barnes was read before the 18th International Geological Congress, held in 1948; it described a similar survey in the search for zinc-lead deposits in the Pinos Altos mining district of New Mexico. The survey involved orebodies of the replacement type in limestone, the densities of the rocks averaging about 2·7 against the 3·8 of the ore. One feature of this area is the ruggedness of the topography with average slopes of 20°. The gravity survey revealed three positive anomalies, after suitable reduction of the measurements. One of these was disregarded, as it bore a close resemblance to the topography and was probably due to local errors in the correction for elevation. Drilling on the second anomaly did not encounter any mineralization, but this was considered inconclusive, as the holes were not correctly sited. On the third gravity high, of two inclined holes, one encountered 10 ft. of ore followed by mineralized limestone and the other a greater thickness of ore. This survey is of considerable importance, since it demonstrates that even under conditions of rugged topography successful gravity surveys are possible with quite small gravity anomalies.

The torsion balance survey made in the Orange Free State (7) will serve as an illustration of the indirect approach. In this area, the Witwatersrand quartzites and conglomerates lie below the Ventersdorp System, which in turn is covered by Karoo sediments. Drill-holes had revealed thicknesses of 5,000–7,000 ft. of the Ventersdorp beds, with correspondingly great depths of the reefs. The problem was reduced to the discovery of areas in which the last-named beds would be nearer the surface, a condition which involved a thinning of the Ventersdorp System. In this sequence the basic lavas have an appreciably greater density than the others, and also of the quartzites. Further tests had shown that high gravity values existed when the lavas were thick and the reefs correspondingly deep. It appeared reasonable to search for regions of low gravity value, where the lavas might well be thin and the quartzites at a shallower depth. The conditions have been deliberately simplified for the sake of brevity, for other density contrasts exist—in particular between the Witwatersrand quartzites and shales. It was possible to distinguish this last contrast, however, by its magnetic influence.

An example of the type of result obtained is shown in Fig. 6, in which a well-defined trough is revealed by the lines of equal gravity (broken curves) and the gradient vectors pointing to the regions of high gravity. A bore-hole, sited on the basis of this gravity anomaly, entered Upper Witwatersrand quartzites at a depth of 991 ft.

It is worth noting that the surface topography in this case favoured a torsion balance survey. A modern gravity meter would have revealed the structure of the low gravity region with possibly better accuracy than the torsion balance and at a far greater speed. It was found, however, that, in the analysis of the results, the 'Horizontal Directing Tendency', which is measured by the torsion balance but not revealed by the gravity meter, was also of value in the interpretation.

Essentially the gravitational method requires a density contrast than the perfect insulator. This is not always true and, by applying the current in the most appropriate manner, the measurements can, as a rule, be made to reveal either type to the best advantage. In general, the current flowing parallel to the strike of a good conductor gives conditions most suited to its detection, whereas for a high resistivity feature the current perpendicular to the strike is best.

This diagram reveals many of the advantages and disadvantages of the surface potential methods. Quite small resistivity ratios

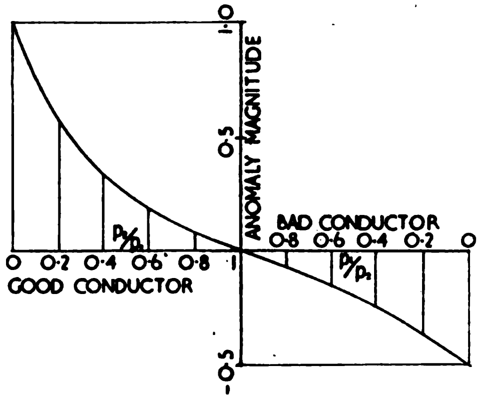

FIG. 7.—The effect of the resistivity ratio on the distortion of an electric field.

give appreciable disturbances, when compared with the maximum possible effect, and hence very large ratios are not necessary. A second point is that there is only a small percentage change in the magnitude of the anomaly (actually 5 per cent for a good conductor in the case shown) as the ratio decreases from $\frac{1}{30}$ down to zero. Now the range of rock resistivities is large, from a few ohm.cm. for a massive galena up to 1,000,000 ohm.cm. for certain crystalline rocks, but there is no corresponding change in magnitude of the anomaly. The significance of this is best illustrated by an example. In a massive limestone of resistivity, say 100,000 ohm.cm., there is little change in the magnitude of the anomaly if the body resistivity lies between 3,000 and 0 ohm.cm. In this range will lie the values of good conducting metal sulphides, but in addition bands of clay, graphitic schists, etc., may fall within it. They are detected with equal ease and no adaptation of this system will distinguish them. Under suitable conditions, other geophysical methods may assist here, since gravity observations may select the dense deposits, spontaneous polarization may reveal oxidizing sulphides, or associated magnetite may permit a magnetic distinction.

The factors controlling rock resistivities are water content and its salinity. The water content may vary rapidly, particularly above the water table, where its retention against gravity depends on grain size, etc. Thus, in the surface layers, resistivity changes are likely and they will give an irregular background masking the influence of deeper contrasts. Here the large resistivity range is a disadvantage, since it increases the chances of such contrasts. The diagram also reveals, in a general way, the influence of a surface layer of either high or low resistivity. A surface layer covering a fault, with the resistivities in the ratio 1 : 10 : 100, gives a ratio change from $\frac{1}{10}$ to $\frac{1}{100}$ as the fault is crossed, whether the surface layer has the highest or the lowest resistivity. Accordingly, only a small percentage change in the measurement can be anticipated as the fault is crossed. With the good conducting surface layer the current remains in it, no matter what the lower resistivity, and, with a high resistivity surface layer, the current is dragged down into the better conductors, that left to influence the surface measurement being practically independent of the lower resistivity. By suitable steps the importance of these conditions can be reduced, but the effect cannot be eliminated entirely.

The interpretation of electrical surveys involves a knowledge of the way in which a current flows through, and distributes itself in, the ground, and how this flow responds to modifications in the resistivity distribution. This subject is complex and a detailed mathematical analysis is impossible, except in simple cases, although the general principles are well known. The calculation of the magnitude of the effects in any particular case is difficult and often impossible, although model experiments may be used for this purpose when the experimental conditions are more ideal than those encountered in practice. Quantitative interpretations are accordingly rare and the numerical checking of an interpretation difficult. The practice, where possible, of making observations over known conditions, similar to those anticipated in the extended survey, is the most satisfactory method of establishing a basis for interpretation.

The major defects of the electrical methods can be summarized as a lack of discrimination, a limited depth of penetration, and a susceptibility to near-surface features which give an irregular background. This last condition is by no means universal, and many interesting and useful surveys have been made. It is unfortunate, however, that although some judgement can be made on the feasibility of any particular problem, it is often difficult to forecast the magnitude of the background. A preliminary survey gives the only definite answer.

The electrical methods can be used with advantage in a large number of different problems. In structural studies it has been used to trace faults and dykes, to determine depths of contacts between rocks and (an extension of the last) to the determination of the attitude of a contact—i.e., its dip and strike. In a sequence of rocks, a bed of outstanding electrical property can frequently be traced and used to detect shallow anticlines, faults, etc. The determination of sub-surface topography has been successfully accomplished. These are examples of the indirect use.

A simple direct problem has been discussed by de Magnee at the 18th International Geological Congress, where it took the form of a pipe of diamond-bearing kimberlite in Precambrian limestones but hidden by a thick layer of sands. The yellow ground proved to be a much better conductor than the limestone, but, in addition to the resistivity magnitudes, the values over the limestone were erratic, owing to irregular changes in its depth, while the kimberlite gave much smoother values.

In these cases the irregularities in the background, although substantial, did not prove a barrier to the use of the method, because of the very large change produced by the near surface features. In fact, the change in nature of the background can be of diagnostic value. If, however, the observations with position vary as in curve B, Fig. 8, with rapid changes by a factor or two

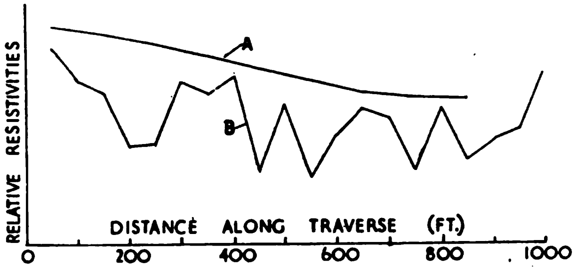

FIG. 8.—Resistivity measurements along lines showing (A) a suitable background and (B) an unsatisfactory background for the detection of small effects.

in very short distances, it is obvious that quite large effects from a wanted body would pass unnoticed. On the other hand, quite a small effect would be detected on A in spite of the fact that it is varying, because it does so smoothly. The former values were obtained in an area of sands, clays, and gravels with an irregular distribution, the other over chalk with a gradually changing thickness of soil. To some extent these factors can be taken into account by observations with different current penetration, but ultimately such lateral changes limit the application of the electrical methods, particularly to deeper seated disturbances.

GENERAL

The electrical methods have been discussed mainly in relation to the surface potential methods, but similar considerations apply to those methods which measure the magnetic field to determine the current distribution. In the present form of the geophysical methods there appears little advantage in increasing the accuracy of the measuring equipment. In general such a step would be to increase the accuracy with which both signal and background is known without permitting any better discrimination, and all methods—gravity, magnetic and electrical—have developed to the state when the general background is recorded. The electrical inductive methods, however, have not been exploited to the limit, and it may be that greater precision here, leading to a slower field survey because of the increased accuracy, might give some advance. Again efforts have been made to utilize natural earth currents, the so-called telluric currents, for the investigation of regional geology. These currents, distinct from those produced by electro-chemical activity, are more uniformly distributed than artificial currents and should give an increase in the ratio of signal to background. At the moment such investigations are on a large scale and technical difficulties arise in their application to small-scale problems, in particular the small voltages to be measured.

Of the other methods, the radioactive method has been found of use as an aid to geological mapping through a surface cover, as long as this is not too thick. Its use, however, is very limited. Again, efforts have been made to adapt a seismic technique to the problems of mining geology and with costs suited to this aspect. There is also the technique of making observations in bore-holes, thereby, to some extent, enlarging the effective region examined. Another potential field consists of observations within mines, a field in which, at present, there is but limited experience. Of necessity, such observations are restricted to a set of lines within the mine workings, but under suitable conditions valuable information may be obtained.

So far, the question of cost has not been touched. It is difficult to make any reliable estimate since it depends on so many factors. The usual magnetometers or electrical equipment cost a few hundred pounds, but the price of a gravity meter is two to three thousand pounds. Although, with any of these instruments, the time taken at any station is small, the surface topography and vegetation influence not only the ease of transport of equipment from one station to the next, but also the cost of surveying in the stations. In an electrical survey such conditions will impede the manipulation of the long cables leading to the ground contacts. The ease of actually making ground contact must be considered. Again the rate of covering the ground depends to a great extent on the nature of the problem, for this controls the interval between observations. In open country some 20 to 30 magnetic stations at close intervals (25–50 ft.) can be made per hour by a skilled man, or a mile of traverse can be laid out and measured by the resistivity method in a day, but adverse conditions may reduce this by a factor of three or four.

It is unfortunate that, in a paper of this nature, a large part must of necessity be devoted to the limitations of the methods. As a result the picture is distorted from its true perspective and is painted in sombre tones. Nevertheless, these disadvantages must be fully appreciated in order to take the best advantage of the available techniques and to prevent wasted effort. The author is convinced that the various methods in their present form are capable, in the right environment, of making an important contribution to solving the existing prospecting problems, and there is no reason to doubt that future advances will extend their usefulness to a wider range of problems.

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An Improved Type of Konimeter*

By J. P. Rees, *Associate,* and S. R. Rabson

The konimeter is a dust-sampling instrument which is extensively used in routine dust control in mines on the Witwatersrand and other parts of the world. It was first introduced on the Witwatersrand by Kotzé in 1916.† Later the design was modified by the Transvaal Chamber of Mines who developed the circular konimeter.‡ An instrument of similar type was also produced by the firm of Zeiss in Germany.

Recently a sub-committee was appointed by the Chamber of Mines to investigate improvements in dust-sampling technique. The members consisted of officials connected with dust-sampling on Witwatersrand gold mines, under the Chairmanship of Mr. J. P. Rees (Chamber of Mines), the other members being Prof. C. W. Biccard Jeppe (University of the Witwatersrand), Mr. J. de V. Lambrechts (Anglo American Corporation), Mr. C. T. Hardy (Silicosis Research Committee, Government Mines Department), Dr. J. T. McIntyre (New Consolidated Gold Fields), Mr. R. A. H. Flugge-de Smidt (Union Corporation), Mr. E. C. Polkinghorne (Government G.M. Areas), Dr. R. A. Pelletier (New Consolidated Gold Fields), Mr. J. Buist (Chamber of Mines), Mr. E. C. Whittaker (Robinson Deep, Ltd.), and Mr. S. R. Rabson (Chamber of Mines). The sub-committee developed an improved konimeter incorporating the best features of the previous types in addition to several new features.

The konimeter collects a sample of airborne dust by causing a known volume of air to impinge through a jet on to an adhesive-coated glass slide, whereby the dust particles are deposited on the slide in the form of a ' spot ' capable of being enumerated under the microscope after suitable treatment.

The new instrument, which has been named the Witwatersrand Konimeter, was designed with a view to ease of handling. Its shape shows a certain degree of streamlining. The body is made from hardened aluminium or other light alloy. The overall length is 6¾ in. A feature of the instrument is the use of a square slide

*Published in the *Journal of the Chemical, Metallurgical and Mining Society of South Africa,* Sept. 1948, and reproduced by kind permission of the Society.

†Final Report of the Miners' Phthisis Prevention Committee, p. 10, 1919.

‡Boyd, J. The estimation of dust in mine air on the Witwatersrand. *J. S. Afr. Instn. Engrs.,* Vol. 26, 1928, p. 142.

$1\frac{1}{2}$ in. by $1\frac{1}{2}$ in. in size, cut from a standard 3-in. by $1\frac{1}{2}$-in. microscope slide, on which a circle of up to 58 dust spots can be taken.

The instrument is essentially a small suction pump with a capacity of 5 c.c. It consists of a nearly hemispherical head recessed to contain the glass slide connected to a barrel containing in its upper section the cylinder and piston. The cylinder is made of brass and is threaded into the top of the barrel. The piston is a precision sliding fit in the cylinder, and is made of steel. It is provided with oil-retaining grooves to ensure air-tightness. It has a bore of $\frac{5}{8}$ in. and a stroke of $1\frac{1}{32}$ in. Attached to the piston by means of a piston rod is a rectangular guide which slides along the inner sides of the middle section of the barrel. The rod extends through the guide to a disc by means of which the piston is pushed up to the top of the cylinder by a finger inserted through a hole in the bottom of the barrel. The extension of the piston rod passes through a felt washer held in a baffle plate across the barrel at a point between the guide and the finger disc. This baffle serves to keep out dirt or sludge from the upper part of the barrel.

The piston is pushed up against the pressure of a coiled brass spring containing 15 coils. The coils of the spring go over the brass cylinder and stretch from near the top of the cylinder to the back of the guide. The spring is made as long as possible to assist in giving an even speed over the whole stroke of the piston. The strength of the spring is such as to give a time of stroke of between $\frac{1}{4}$ and $\frac{1}{3}$ sec. When the piston is pushed up, it is kept in position by the engagement of a catch over the guide. The catch is attached to the lower end of a lever which is pivoted at the centre, the upper end being attached to the trigger which protrudes through a hole in the side of the barrel just under the head. To disengage the catch, the trigger must be depressed against the pressure of a small spring. When the trigger is released, the main coiled spring forces the piston to the end of its stroke, thereby producing a partial vacuum in the cylinder and causing air to be drawn in. The position of the trigger is convenient to the thumb when the instrument is held round the barrel with the left hand.

The cylinder is in communication with the space under the glass slide in the head through a channel leading from the top of the cylinder. The glass slide rests on a rubber ring, diameter $1\frac{3}{8}$ in., partially recessed into the floor of the well in the head, and is pressed against it by a spring on the back of the cover closing the head. An air-tight chamber is thus formed under the slide, so that the vacuum caused by the displacement of the piston enables 5 c.c. of air to be drawn in through a jet or nozzle leading into this space from the outside air. The jet is made of brass 3·6 cm. long and is threaded externally along half its length for screwing into the head. It is tapered from a diameter of 1·75 mm. at the inlet end to a diameter of 0·5 mm. at the outlet facing the slide and is placed at right angles to the glass slide, the outlet being at a distance of 0·5 mm. from it. The air impinges at high velocity from the jet on the slide, to which a thin film of adhesive has previously been applied, and the dust particles adhere firmly to the slide in the form of a spot.

The tapered jet gives a better distribution of the particles in the spot than a parallel-sided jet. The inlet of the jet is flush outside with the surface of the head, a sunken lock nut being used to keep it in position and another detachable sunken nut is provided which, when removed, exposes several threads of the jet, and enables an adaptor to be fitted which can be connected to a U-tube for testing the capacity of the instrument. The air is exhausted from the air-tight space through the exhaust channel communicating with the cylinder, the exhaust being placed circumferentially round the jet.

The $1\frac{1}{2}$-in. by $1\frac{1}{2}$-in. glass slide is held in position in a square recess cut in a toothed brass ring, no screws being required to fasten the slide to the ring. The teeth of the ring engage with a pinion connected to an external winding pin by means of which the slide can be advanced in a circle after taking each spot. The result is a circle of spots round the slide at a diameter of $1\frac{1}{16}$ in. The winding pin is provided with a notch, which enables an exact number of half turns to be given between spots, and also prevents any unintentional movement of the winding pin. Usually three half-turns are given between spots, and with this spacing a maximum of 58 spots can be taken on one slide. The brass ring is numbered from 1 to 29, and the numbers may be viewed through a hole provided in the cover closing the head, to enable the position of the slide to be judged.

The cover, or lid, closing the recess in the head fits on with a small turning movement, three studs under the edge of the cover fitting into corresponding slots in the head. A thumb screw is provided to prevent accidental loosening of the cover.

The brass toothed ring is an integral part of the instrument. The glass slides are kept loose, preferably in a box provided with slots. When required for use a slide is cleaned, the adhesive film is applied to one side, and the slide inserted into the recess of the ring in the konimeter, with the adhesive side downwards facing the jet.

A new method of applying the adhesive has recently been developed on the Witwatersrand. A film of petroleum jelly (e.g., vaseline) is formed on the slide by applying a solution containing 1 per cent of the jelly dissolved in xylol and allowing the xylol to evaporate.

After sampling, the slide can be handled by first lifting an edge with tweezers or any suitable appliance, e.g., a large pin, and then seizing the slide with the fingers held at opposite edges.

Slides are generally treated by immersion in acid and ignition in order to remove soluble and carbonaceous matter, and are then examined under the microscope using a 16 mm. objective with dark-field illumination. An 18° sector eye-piece graticule is used to enable the particle count to be made. A special circular stage is used to advance the slide, with a toothed brass ring similar to the one in the konimeter, but numbered in the opposite direction since the slide is counted face upwards.

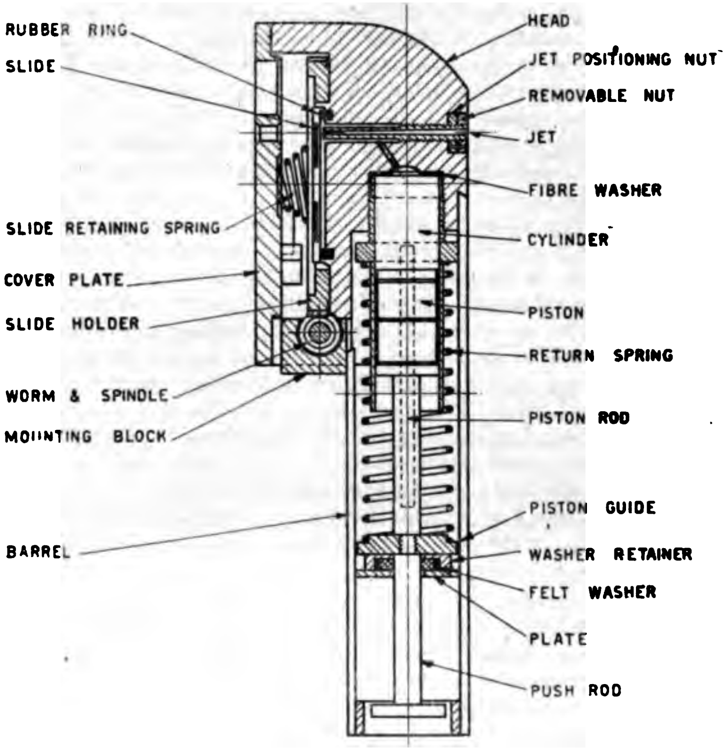

FIG. 2.

Figs. 1 (Plate I) and 2 show two views of the Witwatersrand Konimeter. Detailed working drawings for the instrument have been prepared, and copies may be obtained from the Transvaal Chamber of Mines on application by interested bodies.

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J. P. Rees and S. R. Rabson: *An Improved Type of Konimeter.* *Plate I.*

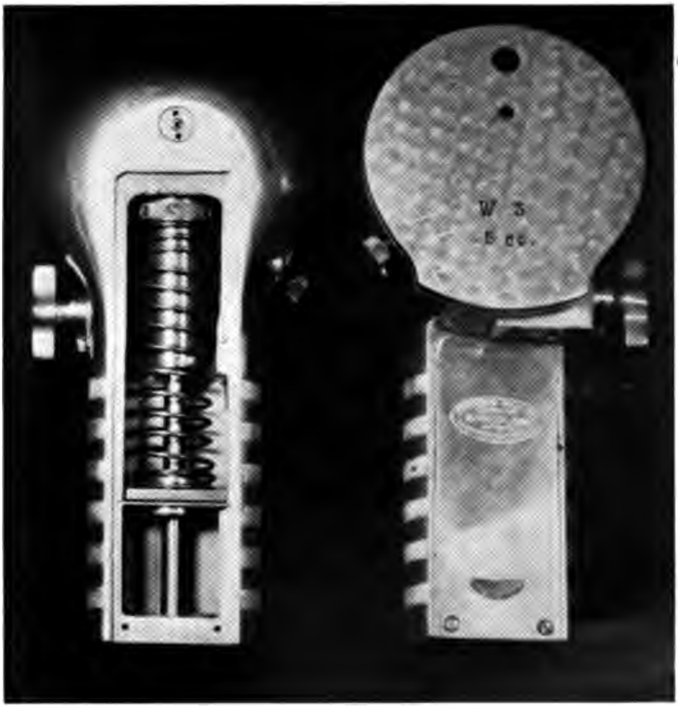

Fig. 1.—Front view and back view with cover plate removed.

THE INSTITUTION OF MINING AND METALLURGY

FOURTH ORDINARY GENERAL MEETING of the 58TH SESSION held in the Rooms of the Geological Society of London, Burlington House, Piccadilly, London, W. 1, on Thursday, 20th January, 1949.

Mr. S. E. TAYLOR, *President*, in the Chair

DISCUSSION ON

Mining and Milling Antimony Ore at Consolidated Murchison Goldfields, Transvaal*

By RALPH SYMONS, *Member*

The President, in opening the Meeting, said that since the author was unable to be present to introduce his paper, Mr. J. B. Dennison had kindly undertaken the task. It was an interesting paper to which the author had given a very modest title, because it was clearly a comprehensive report on what amounted to a group of small mines. It covered nearly every department in addition to mining and milling.

Mr. J. B. Dennison said that he was sorry that Mr. Symons was not present to introduce his paper because, needless to say, his knowledge of the inside of Murchison was much greater than the speaker's own. He had been at the mine and had been underground and had vivid recollections of much of it, but that was not the same thing as the ordinary day-to-day battle of the man on the job. He thought it would be agreed that the author had given good measure and food for thought in the paper, and members would be grateful to him. When he undertook to introduce the paper he wrote to the author to find out if there was anything he wished to add and he had been good enough to send the following up-to-date information :

'We have made one or two improvements in our practice since I wrote the paper.

Method of mining: In order to overcome the difficulty mentioned in the paper in connection with weak side walls we are now working all stopes on the cut-and-fill method. So far this has only been applied where the antimony reef is narrow, and the waste is blasted from the side walls so that it might be called resuing. The reef is carried on one side of the stope back and the waste is blasted down first with light holes and the reef blasted separately later. Grid ore-boxes are carried with sawn-off laggings to prevent the creeping against the compressive action of the ore. This is working

*Bull. 506, Jan. 1949.

very well and, although a certain amount of fines is lost, the drawback of pollution is overcome and a clean product obtained.

Breaking ground: Tungsten carbide steel was being tried when I wrote the paper, but the experiment was in too early a stage to mention. The result has been so successful that it has been decided to equip the whole mine with this type of steel '.

Later in the letter Mr. Symons gave the results obtained by the use of carbide-tipped steel. One drill apparently did 151 ft. (sharpened twice, ground exceptionally hard) ; another steel did 556 ft. (sharpened three times) ; another one 2,755 ft. (sharpened four times) ; another one 1,189 ft. (sharpened four times) ; and another 928 ft. : the record was 4,468 ft., sharpened four times. With regard to No. 1 steel which did not do so well the author said that it was used in an extremely hard quartzite in which it was impossible to obtain a normal round with any steel. On occasions it took four or five jumpers to drill one hole. In other cases where ordinary steel was replaced by tungsten carbide tipped steel one jumper boy per machine crew was saved. The author added : ' In view of these results I have decided to introduce this type of steel for all development work and I am also making trials in stoping work '. With results like that one could understand his coming to that conclusion. Some of the advantages of the carbide tipped steel were the saving of transport in the mine (which was an important factor in view of vertical travelling ways) and the saving of transport from the shafts to the central drill sharpening shop—a very important factor when the work was so widely scattered. There was also the saving of one jumper boy on each machine gang, the increase of tonnage drilled per machine shift, which resulted in a better breaking efficiency and the saving of complete machine gangs, as well as the economy in compressing or drilling equipment.

Another point on which the author touched was underground lighting. He said : ' It has been decided to equip the whole mine with electric cap lamps in place of the present carbide lamps. It is felt that the introduction of this type of lamp will prevent many accidents, because persons will be able to use both hands on the ladders. By improvement of working on the face it is hoped that better efficiency will result, although the cost will be the same or a little higher '. He also added : ' It has been decided to rebuild the compound completely with single and married quarters of the Rondavel type, built of brick with felt roofs, with kitchens, shower baths, latrines, and a new and larger native hospital. The compound will continue to be of the unenclosed type '.

Most of the electric cap lamps which the speaker had seen had an alkali battery, but a friend who was a member of the Institution had told him that alkali batteries were rather old-fashioned and that the latest thing was to have acid batteries which could not spill. He was shown a photograph of a battery lamp with three batteries and a top shaped like a small electric heater, which he illustrated on the blackboard. Inside were two glass tubes exactly like the fluorescent lighting tubes with which one was familiar. It was claimed that with these three 12-V. batteries it was possible to obtain a light which would light up the whole of the development end. He wondered who would want to carry such a battery, but it had a handle which would accommodate only one hand and his friend had said that it could be carried thus by one man. He was old-fashioned enough to feel, however, that he would very much like to have a carbide lamp somewhere handy!

At the end of the Surveying and Sampling section the author stated that 'The assay plan factor for 1947 was 82·75 per cent for antimony and 107·9 per cent for gold', which interested him and doubtless others. Assay plan factors in his experience were usually something under 100 per cent; how common it was to find it over 100 per cent he did not know, but thinking about it he came to the conclusion that one of the many considerations that must affect the assay plan factor was dip. There were many others: friability, the nature of the foot-wall, and whether it was smooth or rough. In many mines on the Rand and other places the foot-wall was sufficiently rough and the ore sufficiently valuable to have people going about with wire brushes on their hands and knees scrubbing the foot-wall, and that was why the question of the dip particularly interested him. On reference to one of the cross-sections (Fig. 3) it was to all intents and purposes vertical; obviously the nearer the deposit was to the vertical the less likelihood there was of loose ore catching in ridges in the foot-wall. One could say that the worst possible case for assay plan factor was the horizontal point and the best case of all would be when the reef was vertical (as in this case) for the simple reason that the ore shoots straight into the mine cars and a very great deal would not touch the side-walls at all.

There were many other considerations affecting assay plan factor. He imagined when Mr. Symons first thought his assay plan factor was 107 per cent he felt very much as the speaker himself would have felt on receipt of an unexpected large rebate from the income tax! One obtained 82·7 per cent antimony, but, if what he thought was right, if it was a much flatter orebody one would expect an assay plan factor of 60 or 70 per cent.

A non-technical point, but a point of interest, was that the ore was most satisfactory to mine. Those who had to mine large deposits of galena in limestone would appreciate what he meant. It really looked worth mining, not like so many mines on the Rand, where men had been mining for 20 or 30 years and had never seen the metal for which they had been mining.

There were other points of interest to which he would like to refer, but he had dealt with a few of the branches of the paper and would leave it to others to do likewise.

Dr. F. E. Keep said he regretted that although he had visited many small workings in southern Africa he had not had an opportunity of visiting the mines of the Murchison Range. The paper gave an interesting account of those gold-antimony deposits, which were among those first known to Europeans in South Africa, but which had always in the past proved to be a snare and a delusion to the individuals and small companies who attempted to work them. The refractory nature of the ore, the technical knowledge and large capital expenditure necessary to obtain satisfactory gold recoveries, and the unhealthy nature of the locality, combined with the erratic and patchy occurrence of the gold values, were the major factors responsible for the failures of the early workers. In fact it might be fair to emphasize that a successful mining industry had not yet been built up on the gold occurrences, but that the greatly increased value of the antimony alone had resulted in its being possible to operate the mines profitably.

It was noteworthy that nowhere in the paper was any idea given of the average tenor of the ore, either in gold or antimony content, although the author stated that the value of the output of the former was only a fraction of the value of the latter. Was the fact that sample lengths were not permitted to exceed 12 in. due to very erratic antimony values as well as to patchy gold values? Also, had the author any simple explanation of the discrepancy between assay plan factors of 107·9 per cent for gold and only 82·7 per cent for antimony? Might that be due to the fact that the gold values were often visible and always erratic, resulting in a rigorous reduction of actual sample values when ore reserve estimates were being prepared—a reduction which proved in fact to be too rigorous? He did not believe that Mr. Dennison's suggestion regarding the dip really explained the discrepancy: it might be one factor, but only one. He would himself prefer not to have to plot his assay plan factor against the angle of dip and to be expected to keep to that curve.

Having had considerable experience of shrinkage stoping in steeply-dipping veins he could say that in his opinion it was a poor method, its only redeeming feature being its cheapness in first cost. However, it might, and often did, prove very expensive in the long run, owing to the large unsupported cavities left, to dilution of the ore from weak walls (a particular danger, it would seem, in the Murchison orebodies), and from loss of ore due to large wall slabs hanging up above stope-boxes or from the collapse of cribbed manways and the consequent mixture of broken timbers with the ore. To travel down with cleaners as the ore was finally drawn was too dangerous except under particularly good wall conditions, when, in fact, the cleaners were of least use. It was surprising to read that cribbed manways as pictured in the drawings did not collapse as the stopes were being finally drawn. He was pleased to hear that cut-and-fill stoping was to be used, and he believed that if any considerable orebodies were encountered they would be very thankful a few years hence that they had changed their methods of stoping to cut-and-fill. Mr. Dennison had mentioned the loss of fines in that method of mining, but he did not consider that to be necessary. He had seen a lot of cut-and-fill stoping in high-grade orebodies, particularly in Canada, and it was customary to cover over the fill with planks, often overlaid with old filter canvas from the sulphite paper mills, before breaking down the ore. That practice, besides preventing loss of fines in the fill, assisted the shovellers in getting the ore to the 'mill-holes' or passes by providing a level surface from which to shovel. He realized that timber was much more expensive in South Africa than in Canada and probably old canvas was unobtainable, but some type of fairly inexpensive covering for the fill might be procured.

The extraction of sill pillars below drives was an expensive proceeding, particularly so when a large open stope existed below. No wonder the author stated that those pillars were left 'to be partly extracted later'. What percentage of the payable ore in those sills was actually recovered? The foot-wall haulage system, combined with crosscuts to the reef, and no reef drives, had the advantage of obviating the necessity for any sill pillars being finally left, the lower stope being able to continue right through to the base of the stope above. That advantage might, combined with the cheaper cost of a foot-wall drive on line as compared with a reef drive which had to follow all the irregularities of the orebody, and combined with the fact that the shovelling of the silled spoil into cars on the level was not necessary, result in an actual saving of cost as compared with the reef drive method of development.

The author had stated that crosscuts or diamond drill-holes were put out from the reef drives to examine for values in the walls. He stated earlier in the paper, however, that owing to the sporadic mineralization no great reliance was placed upon assays from individual diamond drill-holes from surface. Under those circumstances the value of diamond drilling the walls of the drives appeared to be controversial.

It was noted that 50 per cent dynamite was used for stoping. Presumably the use of the cheaper 40 per cent explosive had proved less economical. In the speaker's experience it was rarely necessary in fairly wide flat-back stopes to use more powerful explosive than 40 per cent dynamite, but at Consolidated Murchison tests had presumably proved its use inadvisable. Information on that point would be welcome.

Although he had asked for further information and perhaps had criticized minor points in the paper, he thought that the author was to be congratulated upon a very interesting dissertation. It suffered from the lack of certain particulars, but perhaps they had been omitted for reasons of policy, the base-metal producer being in a competitive business. He could, however, pass on any increased costs to the consumer, a thing the less fortunate gold miner was unable to do.

Mr. D. G. Armstrong said that Dr. Keep had commented on the rather surprising absence of assay figures, particularly mill heads, and he wondered what the gold figure was for the mill heads. There was a surprising omission in the gold flotation section, where there was no mention of reagents, and he wondered if it was a trade secret and deliberately omitted. In the crushing section it was stated that the ore bin was covered by a 14-in. by 14-in. grizzly, and he wondered whether it should be 14 ft. If 14 in. was correct, the ore in the bin would be *minus* 14 in. with an occasional piece nearly 14 in. in dia. That had to go on to the 22-in. conveyor set at a slope of 17 in. and was fed into a 16-in. by 9-in. jaw crusher, so he wondered if that was correct. The jaw crusher discharge chute was set at 32°. He wondered if that was by design or necessity as he imagined it would choke up rather easily.

In the milling section it was stated that there were two ball-mills. One of them was 5 ft. by 16 ft., and he thought it should more properly be called a tube-mill. One found later on that a helpful factor in roasting the antimony was coarse particles. Was there not too much sliming in the long mill ? The lining was interesting—white iron bricks. He imagined they were quite small as they were called bricks and it would be interesting to know how they were fixed in the shell. He took it that the third dimension given for the mill size was the size of feed that the mill would take. There was also a third dimension for the strake tables and he supposed it was depth, but it was hardly a relevant figure.

In the gold flotation section the author stated 'Talcy gangue will float in preference to antimony at this stage'—perhaps a gangue depressant would be useful at that stage, but they might not want a higher grade gold concentrate as it might cause trouble later in the roasting process.

In the section on roasting it was stated that temperature was determined by means of a Cambridge electrical pyrometer and he would like to know how it was done. He presumed a thermocouple was meant, but he wondered where they put it, because it was the temperature of the ore on the hearth rather than the atmosphere in the furnace which was important. With the sweeping rabble arms there was not much space on the hearth for a thermocouple. Calcine was discharged direct on to a conveyor without cooling and then in to a bucket elevator. He thought that would be a very dusty operation and the elevator joints would need to be very tight.

The double treatment with cyanide was interesting. There seemed to be extensive washing before filtration, which was rather surprising. Assay figures to show the extraction by the two cyanide treatments would be interesting. In tabling the antimony flotation concentrates, he would expect some difficulty with frothing on the table and he wondered whether any precautions were taken to prevent the products from floating over the table. Precipitation zinc was purchased ready cut. Some people attached considerable importance to the freshness of the surfaces and would prefer it to be cut at site. Apparently no trouble was experienced at Consolidated Murchison.

Dr. A. W. Groves said that the author stated at the end of the History section that that was the only mine producing antimony in the Union of South Africa. He thought that was an unduly modest statement and the fact might have been brought out that it was the principal producer of antimony in the Empire. The 1947 statistics showed that the Union of South Africa was third on the list of world producers of antimony; Bolivia came first, Mexico next, and South Africa third.

Mr. J. A'C. Bergne said that one point had not been mentioned on which the author deserved congratulation and that was his handling of the enormous and indifferent collection of equipment—probably old, possibly 'junk'—which no doubt he had inherited from the little properties now amalgamated.

If one looked carefully at the makers' names of the various types of machinery there was hardly a single case in which the same name occurred twice. That state of affairs was reflected in the composition of his staff; the engineering staff was practically half the entire executive. The costs Mr. Symons had achieved with that dispersed and old machinery were very creditable.

Mr. N. H. Monro said that he was particularly interested in the phenomenal footages achieved with the tungsten carbide drill-bits described by Mr. Dennison. He would be glad if the author would say what reduction in gauge there was at the end of the runs. It was an interesting point, although not particularly important, as there was no question of one drill having to follow another in the same hole. With ordinary carbon steel loss of gauge was often a serious aspect and for that reason the loss in drilling 4,468 ft. with one bit was an interesting point.

With regard to the interesting fluorescent tube lamp which Mr. Dennison illustrated; in case members had not seen it he would like to mention a lamp which he saw at the British Thomson-Houston works the other day. It was very similar to that described by Mr. Dennison, with two fluorescent bars driven by a compressed-air motor.

Mr. E. J. Pryor said that several of the points he wished to make had been anticipated. One of the first which interested him was, as Dr. Keep had pointed out, the unusual agreement between the assay plan and the mill head assays. It struck him, trying to reason out what had happened, that the samplers had been up against that curse of underground sampling—obdurate material followed by rather friable stuff. He had carried out sampling work on just such ground and had found that unless one was prepared to take an inordinate time and to be extremely gentle in cutting samples, the results were very unreliable, owing to the fact that the friable material began to shake loose before the sampler's groove reached it.

He was interested in the possibility of utilizing the obvious differences in density between the minerals in this ore. From its geology it seemed the kind of deposit one would expect to be heavily diluted in stoping. Possibly there would be enough density difference to be exploited by sink-and-float separation methods. If 23 tons/hr. were going through and 30 tons of waste were removed daily by hand sorting, a small 'heavy media' plant might be justified.

Washing on the belt was an interesting feature. With regard to recovery he would like to know more, if it were possible. Twenty per cent of the gold in the mill feed was being recovered on the blanket strakes, another 18 per cent by jigging, and, later on, gold flotation saved up to 15 per cent; a total of 53 per cent of the gold in the ore. Most of that was concentrate and apparently was going to the calciner and there would presumably be an operating loss from that time on. Further recovery from cyanidation of the flotation tailing was mentioned as an operation which was 'not always profitable' and no figures were given to show what happened. If it was possible to give it, the total recovery based on the dwt./ton in the mill heads would be very helpful for record. Experts in that field would appreciate that high efficiency of extraction was rarely possible with a nasty combination of minerals such as existed there.

Mr. F. T. C. Doughty said it was a nice change to have so much information about milling in a general paper, but thought the value of the paper would be improved if more detailed information about flotation were given. The literature on stibnite flotation was scanty and Taggart did not mention it at all.

He was particularly interested in the flotation of talcy gangue prior to stibnite. His experience with stibnite ores was that a portion of the stibnite, sometimes as much as 40 per cent, floated without any promotor at all. Obviously that did not happen at Murchison, as the overall recovery was 75 per cent in a 60–62 per cent concentrate. As the ore undoubtedly contained oxide antimony minerals which would not float, that was an excellent recovery, but he wished the paper had said how it was done. What was the antimony content of the talcy gangue concentrate? There were various references in the literature to depressants and activators for stibnite. Caustic soda was said to depress that mineral, and lead acetate and lead nitrate said to activate it without affecting arsenopyrite or pyrite. He would like to know whether any tests had been made with such reagents.

One other point that puzzled him was the statement under 'Distribution of Antimony' in the Antimony Recovery section, where the author mentioned various minerals and said that all were heavily penalized but were largely removed with the gold. That seemed odd, as one would not expect minerals such as nickel sulphides to float with the talcy gangue. Perhaps, however, they were coarse and were removed in the gravity plant. He hoped the author would give some information about that.

The gold recovery was not very high and it would be interesting to know where the losses were. No doubt there was some gold in the final antimony concentrate.

Mr. J. B. Richardson said that, in prospecting by diamond drill, the author and his geological staff were to be congratulated on the speed of drilling and the excellent core recovery from the orebodies, especially as loss of water was so frequent. For comparison with similar work elsewhere could the author give the average cost of drilling?

The ore being exploited appeared to occur in a series of deep, steeply-inclined lenses varying from a few feet to over 40 ft. wide and several hundred feet long. They had weak, ill-defined walls and mineralization of an erratic, complicated sulphide nature which could be followed apparently between the lenses. From the paper it was not clear how much of the prospecting programme was along the 2½ miles to the east of the main shaft nor the degree of continuity in the line of lenses discovered, although the work was for the discovery of new lenses rather than for their appraisal.

The system of hoisting the ore through a number of small shafts and then transporting it by 5-ton lorries to the main ore bin seemed an expensive way of collecting ore, although probably unavoidable at present. He wondered, therefore, if the management contemplated eventually linking up the numerous lenses by an underground haulage system. The ore itself appeared to be strong, as it needed no support over 45 ft. width, and if such a tunnel were driven mainly in ore part of the cost would be offset. It was noted also that in the deeper stopes the haulage levels were in the foot-wall, and if the distances between the lenses was not great, linking up those at some convenient depth would appear to provide a tunnel to serve as a main haulage road. The paper would have been improved had the author provided a longitudinal section, even in the form of a sketch, so that some idea of the number, size, and shape of the lenses and the distances between them could have been made clear.

If the sketches given in Figs. 3 and 4 represented an average stope, excluding the roof pillar a 100-ft. lift would appear to represent some 15,000 tons of ore. Thus no great number of stopes in various stages of extraction was required to ensure a fairly steady production. The author would appear to have achieved that because the main ore-bin only contained less than a day's supply for the mill.

He had not the same objection to shrinkage stoping as Dr. Keep, as in comparatively narrow steep orebodies, with miners not highly skilled, it was a suitable method. But the author stated that in some sections the walls were weak, so presumably there were considerable dilution and possibly difficulties in extracting the last of the ore, although they were told that the travelling ways were not affected.

Would the author give details of the method of reclaiming pillars, as if the walls sloughed badly that work would appear to be difficult to complete.

In his introduction Mr. Dennison had told them they had gone over to a cut-and-fill method. How did that affect the efficiencies and cost in mining?

Was it not also possible that dilution of the ore was the answer to the anomalies in gold and antimony recovery remarked upon by several speakers, and if Dr. Keep's suggestion that the high gold assays were severely cut was correct, was not the wide divergence in assay plan factors explained by a larger tonnage being drawn than the assay plan calculation gave, so that the true recovery of both metals was higher than 82·75 per cent? That happened in many copper mines using block caving methods where dilution was unavoidable.

The speaker asked why winzes were continued vertically even if they passed out of the orebody and why was winzing preferred to raising. It was possibly a function of the skill of the miners employed, because with skilled miners raises were usually cheaper to make than winzes, where the broken ore had to be raised to the level above and not just allowed to fall by gravity. Might not a training school in a study stope alter those conditions to advantage?

The author gave a few efficiency figures for breaking ground. Fathoms presumably meant cubic fathoms or, he asked, did it mean square fathoms over a width not given; why were cubic feet not used as units? One had to look from p. 14 to p. 36 to find the usual efficiency of pounds of explosive per unit of advance.

The tonnage broken per underground worker employed was given, but not the ratio of natives employed in breaking ground to the total native underground labour force, so that it was difficult to judge whether the figure given was good, bad or indifferent.

He hoped the author in his reply would round off his paper, which left so much unsaid, by filling in the many gaps and so completing an interesting descriptive account of an important mining project.

Dr. Keep said, regarding Mr. Richardson's plea for the use of generally-understood terms in place of those of purely local usage, that he would like to appeal to all, particularly the younger members present, to get into the habit of always defining what ton they were using. The paper under discussion did not specify whether a long or a short ton was used on the mine. Being in South Africa it was probably a long ton, as in Canada it would be usually a short ton, but in Australia it might be either, and even local knowledge would not help one under those circumstances.

It had struck him, as well as Mr. Richardson, that it was curious that at that mine winzing was preferred to raising. Here again customs varied: in Western Australia the miners detested raising and always wished to sink, whereas in Canada the opposite held true, as it also did on the Witwatersrand.

Professor C. W. Dannatt said that the flow-sheets for the East and West mills were so different that they suggested the treatment of ores of quite different types, yet there was nothing in the paper to indicate that that was the case. He therefore asked if the author would explain the reasons for those variations.

Mr. J. C. Mance said that he could give a little of the early history of the Murchison Range. He was there 40 years ago when the Free State mine had been opened up to 200 ft.; the mine started on a quartz reef and there was rich gold and antimony mixed together. At that level a bore-hole was put in horizontally and a pyritic body was struck carrying good gold values and a little antimony, a drive exposing a length of some 75 ft. of ore. A vertical shaft was put down 600 ft., and the pyritic body was struck at the 3rd, 4th, 5th, and 6th levels. At the 6th, where it came into the shaft, a crosscut exposed a width of 90 ft. of mineralized body carrying gold, antimony, and pyrite. In one sample a 3-ft. section gave 10 dwt. gold to the ton. The length of driving in the mineralized body increased in depth at the 3rd level some 300 to 400 ft. and on the 5th some 700 ft. That work was for prospecting only.

In those days there was no railway within 136 miles of the property, gold was £4 an oz., and there was no work on flotation of antimony or anything like that.

Dr. S. W. Smith said he might perhaps make a comment on the paragraphs headed 'Assaying' where the procedure was given for the assay of gold and the make-up of the stock flux. The author said that 'per assay, 100 g. of the stock flux was used', but he did not give the amount of the sample taken for assay. If 'per assay' meant 1 assay ton then he thought with Dr. Keep that the ton should be specified—whether a long ton, a short ton, or a metric ton. The author went on to say that when over 3 per cent antimony was present nitre and additional litharge were added, the quantity required depending upon the amount of antimony and being regulated by experience. It would seem from the paper that there might be cases when the antimony did exceed 3 per cent, perhaps to some considerable degree. As a matter of personal experience in assaying antimonial gold ore he had found that it was most important that the nitre should not merely be added to the charge of stock flux, but should first be mixed intimately with the sample of the ore itself. The object of adding the nitre was, of course, to oxidize the stibnite as completely as possible during the earlier stages of the fusion, otherwise some metallic antimony would be reduced and would pass into the lead button from which it was difficult to remove it either by nitre or even by a subsequent scorification. The author said nothing about scorification and perhaps he found it was unnecessary, although one would have anticipated trouble in the subsequent cupellation if that were not done. He thought that intimate mixing of sample and nitre was a safeguard. It was dangerous to rely merely upon a 'chance medley'.

Those remarks had some relation also to the subsequent treatment of the gold concentrate by roasting. The author admitted that antimony caused trouble and he said the cause appeared to be the fusion of stibnite particles, which prevented further oxidation and encased the gold. One would never, of course, attempt to roast an antimonial gold ore for purposes of assay, but it seemed necessary to the author to roast the concentrate for the purposes of subsequent treatment. He said, 'helpful factors seemed to be the insulation of stibnite particles by gangue as in low-grade concentrate and the existence of the antimony sulphide in coarse particles, e.g., in jig concentrates'. The speaker wondered if those two factors were really effective in ensuring a smooth and efficient running of the charge and a satisfactory extraction. Would it be out of the question to attempt to oxidize the stibnite during the roasting operation by some oxidizing agent? Stibnite was likely to be an inherent source of trouble and it might possibly account for some of the shortages in the ultimate extraction of the gold. The hearth of the roasting furnace, after a period, might be found to carry values.

Mr. F. D. L. Noakes asked to add a rider to Dr. Smith's earlier remarks about the assaying procedure. He noticed that seven parts of fluorspar were used in the flux and he wondered what its function was and whether it tended to give a slag that was too thin. Was it considerably cheaper than borax or did it perform some special function?

The President said that in the earlier part of the paper the orebodies were described as lenticular in character—those towards the eastern end of the mine generally pitched east, and those at the western end pitched west. The pitch of oreshoots was always an interesting feature, but its explanation seemed in many cases to be rather obscure. In view of the fact that there were two distinct pitches in this case it would be interesting to hear if the author had any views or explanations.

Referring again to the question of the high assay plan factor: one point which he would like to mention which had a possible bearing on the question, was the obvious 'spotty' and irregular occurrence of the gold there. It was said that it was patchy and erratic, and if there were a really appreciable number of places where the gold was occurring in that form he thought it was common experience that higher results would be obtained than was indicated by any form of sampling which one could think of. It depended on the absolute number of the high spots which occurred.

One other part of the paper which interested him was the water supply. It was an interesting little scheme and admirably described, except that there was one vital figure which he could not see mentioned—the 'head' from the river to the tank at the mine. If that figure could be given it would complete an interesting account. The author referred to the quantity of suspended solids in the water in the summer months which made it unsuitable for domestic purposes. A filter had recently been introduced and it would be interesting to know where the filter had been fitted.

In regard to the figures of the tungsten carbide bits and drills he would like to know the actual drilling speed attained with those bits, both in the exceptionally hard quartzite and in the softer places. That led him again to the question of drilling cost, because tungsten carbide bits were expensive and although they gave remarkable results it was the overall costs which mattered; if there was any information available as to the difference in cost between the use of ordinary bits and tungsten carbide bits it would be interesting to have it.

If there were any points to which Mr. Dennison wished to reply the Meeting would be glad to hear them.

Mr. Dennison, in reply, said he thought it would be better to get correct replies from the author, but he would like to tell his friend Dr. Keep that he himself did not say anything about loss of fines in cut-and-fill, and he would not like that idea to go any further. He was a great believer in cut-and-fill.

With regard to electric lamps; he had not meant to imply that he mistrusted the ordinary cap lamp. His doubts had been in reference to the new type he had mentioned.

Dr. Keep had mentioned the large labour force. That was very much bound up with the distance from end to end of the group of mines. The mines were very much separated and it was quite a job getting ore from the mines to the mill. That referred to the point raised by Mr. J. B. Richardson, who asked why they were not joined up by underground haulage. That would be an extremely serious matter; he was horrified when he thought of the amount of capital expenditure which would be necessary.

Mr. Richardson explained that he said that if prospecting had shown that there was a continuous line of lenses there might be a good case for a haulage tunnel.

Mr. Dennison said that Prof. Dannatt raised the point of the different flow-sheets for the East and West mills and it might make it easier to understand if it was realized that one of the mills was looked upon as the gold mill and the other was looked upon as the antimony mill—that is, one had the richer gold and the poor antimony and the other had the poor gold and the richer antimony.

He thought he had better leave the other points to Mr. Symons to answer.

The President thought it would be agreed that the paper had stimulated a useful and interesting discussion and he hoped the author would appreciate that the discussion was a tribute to his paper. He proposed a hearty vote of thanks to the author for his excellent paper and to Mr. Dennison for introducing it.

CONTRIBUTED REMARKS

Mr. G. F. Laycock : I am sorry I was unable to be present at the meeting when this interesting paper was submitted for discussion. There are several points under ' Prospecting ' that are not quite clear to me and if these have not already been raised I would like to ask the following questions :

Is the amount of 8s. 8d. per ft. of diamond drilling an overall cost or does this only refer to the cost of the cast-set bits ? If it is an overall cost it is remarkably cheap and I would like to see this figure broken down into itemized costs. If, on the other hand, it represents only the cost of the bits, what is the total drilling cost ?

A systematic programme of drilling from surface along the entire line of favourable country is much to be commended, provided it does not prove too costly. Have geophysical survey methods been employed or considered or is the mineralization too inert to give the necessary reactions ?

The author states that, to promote speed in drilling, cores are taken only at likely horizons. Does this mean that a change-over to non-coring bits is made between these horizons or are the cores allowed to grind themselves away ? Are any sludge samples taken ?

It is interesting to hear that ' owing to the sporadic distribution of mineralization no great reliance is placed upon assays from individual holes.' Are any of the holes deflected so as to obtain two or more intersections of the orebody for each set-up of the drill ?

Mr. N. W. Wilson : In my opinion broad shallow folds, with subvertical or pitching axes, in the quartzite ' bars ' are ultimately responsible for the association of antimony orebodies and hills. Aerial photographs show the folds clearly. Movements that accompanied and followed folding would open channels for ascending siliceous fluids. Analogically the pegmatization of the chlorite schists near a sharp fold in the Antimony ' Bar ' makes a hillock a few hundred feet from the Banded Ironstone Shaft and illustrates the process of hill formation on a small scale.

Although antimony minerals are very uncommon, or even absent, in the banded ironstones at the Banded Ironstone Shaft they were stoped, I think, on the 1st and 2nd levels of these workings in a thin band of dolomitic sediments immediately south of the main banded ironstone.

Abundant actinolite (?) crystals in the dark green, much sheared, chlorite schist a few hundred feet south of the Weigel antimony orebodies may mean that the sediments include volcanic flows. The actinolitic (?) band is particularly obvious near the pump chamber on, I think, the 6th level after about an inch of whitewash has been chipped off.

Why cinnabar should occur only from Monarch Kop eastwards and why the western orebodies would pitch westwards and the eastern bodies eastwards are questions that remain to be answered.

Mr. Symons does not list Jack East, some old formerly inaccessible workings several hundred yards east of United Jack Kop, as productive. Tests of these workings at depth by the surface drills now at the Company's disposal might disclose some ore. I am glad the Company eventually acquired the Gravelotte mine as, four or five years ago, I suggested to Messrs. L. S. Cooke and J. W. Findlay, then respectively assistant consulting engineer and mine manager, that they should take a development option on the property. Because of the lower price of antimony and the difficulty and cost of lorry transport at that time nothing was done.

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AUTHOR'S REPLY TO DISCUSSION* ON

A Simple Flotation Cell

By E. J. Pryor, *Member*, and K.-B. Liou, *Student*

Mr. E. J. Pryor : As it has not proved possible to make contact with Mr. K.-B. Liou, the writer is alone responsible for the following remarks. Answering Dr. Sutherland, turbulence is assumed by the author to provide only a transporting force. If its action upon particle and air-bell brought them into collision, adherence might or might not follow. The tenacity of any such adherence would be a resultant of the ultimate forces seeking to maintain the particle in contact with air and those seeking to re-wet its surface and to pull it downward by gravity. Theoretically, it should be possible to state the strength of these forces in terms of the surface tensions of particle, water, and compounds in solution. Practically, only an indirect and insufficiently dependable method of assessing the surface tension of solids is at present available. The writer agrees that the induction period is of critical importance. During this period the air inside the bell must displace water from the particle surface. Research now in progress at the Royal School of Mines may in due course throw more light on this obscure process.

Dr. Hallimond's cell is a valuable aid in research and the sintered glass pad added by the Australian workers is much appreciated in the Bessemer Laboratory. The essential features of the cell—cleanliness, observation, control and simplicity—shorten research work considerably. Commenting on Dr. Hallimond's remarks concerning reaction of the frothing agent with the bubble surface, the position is this : At the instant when an air-bell emerges into the ore-pulp in a flotation cell the surface tension of the air/water system is high—say 72·80. This high potential energy, temporarily created in the system by the force used to introduce the air-bell, seeks to reduce itself to a minimum. It can do so in several ways, one obvious method being by replacement of water with something having a lower surface tension. If frothing agent—e.g. pine oil—is dissolved in distilled water in a cylinder and a stream of air-bells is then blown in at the bottom, molecules of pine-oil will be carried to the air/liquid interface and there remain, since the surface tension of pine oil is far below that of water. The air-bells now rise and burst at the surface, stripping the water of its frothing agent and concentrating substantially the whole of the pine oil at the top of the column of water in the cylinder. The implications of this fact are too complicated to be discussed exhaustively here, but the phenomenon plays an important part in froth flotation.

Mr. Yeates's comparison of the Pryor cell with the one developed by those great pioneers of flotation, Sulman and Picard, is timely.

*Bull. 505, Dec. 1948.

The greatest single contribution since that day has been, in the author's opinion, the development of captive-bubble methods of testing ores at various levels of investigation. As Mr. Fleming has reminded us, we owe a great deal to the team of Melbourne workers (of which Dr. Sutherland was a distinguished member) that has done so much to develop its practical use.

Replying to the remarks of Mr. Wood, Mr. Rickard had mentioned the use of emulsified pine oil. The author finds this adequate. A geometrical segment of a cake filtered from the pulp in a pressure or vacuum filter is also employed in the Bessemer Laboratory to ensure accurate subdivision. Due care is sometimes needed to ensure that abnormal oxidizing conditions are not set up. The author has no experience in testing samples which have been ground dry, and considers grinding under water an essential factor in controlling the surface of the ore during the later stages of comminution. He hopes nothing in the paper has suggested to Mr. Wood that the author favours flotation immediately after dry grinding.

Mr. Doughty rightly insists that the place for ore testing is the laboratory, not the field. Research work now proceeding in the Bessemer Laboratory is throwing light on the physics of bubble action and it is hoped that the results may prove of sufficient importance for later publication. In the meantime, the Pryor cell is proving to be one of a number of useful tools in this investigation. The physics of particle/air/water attachment are far more complex than would appear from the published literature, and more knowledge of them is needed if froth flotation is to be applied to ores at present considered unfloatable. The author considers unfloatability of any mineral as a technical challenge, not as an impossibility.

With regard to Mr. Wood's remark concerning the *p*H at which galena could be floated, this raises an issue rather beyond the scope of the paper. Its authors have not suggested that the limiting *p*H for the given concentration of the given collector in distilled water at the temperature shown is applicable to any other conditions. Again, it is not minerals that are floated, but surfaces, and the generic term ' galena ' can cover a variety of contaminations which seriously affect the *p*H limits. In certain cases repeated distillation of the water used, together with scrupulous purification of every surface coming in contact with the water, is essential in fundamental research, and one must be prepared to use electron-diffraction, X-ray powder analysis, or spectrography before one can be sure that an aberration is not due to the sorption of trace elements by apparently pure minerals.

No. 509 APRIL, 1949

BULLETIN OF THE INSTITUTION OF MINING AND METALLURGY

Principal Contents :

OFFICIAL NOTICES

RECOVERY OF SULPHUR FROM SMELTER GASES BY THE ORKLA PROCESS AT RIO TINTO

By H. R. POTTS *and* E. G. LAWFORD, *Members.*

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No. 509 **APRIL, 1949**

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NOTICE OF GENERAL MEETING

The Seventh Ordinary General Meeting of the Fifty-Eighth Session of the Institution of Mining and Metallurgy will be held, by kind permission, in the Apartments of the Geological Society, Burlington House, Piccadilly, London, W. 1, on **Thursday, 21st April, 1949,** at 5 p.m.

The discussion on **Geophysics and economic geology,** by Dr. J. McG. Bruckshaw, which occupied the whole of the last Meeting, will be resumed during the first half-hour.

The paper entitled **Recovery of sulphur from smelter gases by the Orkla process at Rio Tinto,** by Messrs. H. R. Potts and E. G. Lawford, which is published in this *Bulletin*, will then be submitted for discussion, and will be introduced by Mr. Lawford.

Light refreshments will be provided at 4.30 p.m. for members and visitors attending the Meeting.

The Council invite written contributions to the discussion of papers from members who may be unable to be present at the Meetings of the Institution. The Council reserve the right to edit and condense such contributions.

INSTITUTION NOTES

Bye-laws of the Institution

The revised Bye-laws of the Institution were allowed by His Majesty's Privy Council on 20th February, 1949, and are now in force. A copy of the Royal Charter and new Bye-laws is circulated to members of the Institution with this issue of the *Bulletin*.

The attention of members is drawn particularly to the following:

Associate Members.—All persons whose names were on the Register of Associates on 20th February, 1949, are now Associate Members. Members and Associate Members together comprise the Corporate Members of the Institution.

Affiliates.—A new class of Non-Corporate members has been created, that of Affiliates (see Bye-law 6).

Membership abbreviations.—Authorized abbreviations denoting membership of the Institution (see Bye-law 8) are now as follows:

Honorary Member	Hon. M.I.M.M.
Member	M.I.M.M.
Associate Member	A.M.I.M.M.
Affiliate	Affiliate I.M.M.
Student	Student I.M.M.

Annual Dinner, 1949

The Annual Dinner of the Institution, after a lapse of two years, will be held at the Savoy Hotel, Strand, London, W.C. 2, on Thursday, 5th May, 1949, at 7 for 7.30 p.m. Applications for tickets (price £1 5s. each) should be made not later than Monday, 25th April.

Annual General Meeting

The Annual General Meeting of the Institution will be held on Thursday, 19th May, 1949. A notice of the Meeting will appear in the May *Bulletin*, which will also contain the Annual Report of the Council.

March General Meeting

The Sixth Ordinary General Meeting of the Session was held at the Geological Society of London on Thursday, 17th March, 1949, when about 90 members and visitors attended to hear the discussion on 'Geophysics and Economic Geology'. The author, Dr. Bruckshaw, was present and introduced his paper, and a report of the discussion will be published in the *Bulletin* for May. As there was not time for all intending speakers to take part during the evening, the discussion will be resumed at the April General Meeting.

Fifty-Eighth Session, 1948-49: Dates of Subsequent Meetings

The following are the dates fixed for General Meetings of the Institution during the remainder of the Session 1948–49:

19th May, 1949.
16th June, 1949.

(These dates are the third Thursday of the month.)

Transactions: 15-year Index to Volumes 41-55

An index to Volumes 41–55 of the Transactions of the Institution is in preparation and will be published in due course. One copy of this Index will be available free of charge to each member of the Institution who orders it in advance, and those who wish to have the Index are asked to inform the Secretary *by 31st July*, 1949. An order form for this purpose will shortly be sent to all members of the Institution.

Malaria Control Course for Laymen

The 'lay' course in malaria control for planters and miners is to be re-established by the Ross Institute and will be held from 18th to 22nd July, 1949.

The following is an extract from a notice circulated by the Director of the Ross Institute of Tropical Hygiene:

'Before the war this course was very successful and greatly appreciated. It was the main agency which produced a generation of planters and miners who not only understood the gravity of malaria but had a considerable special knowledge of how it could be controlled.

'This generation is now growing older and is being replaced by one without this knowledge, which is as necessary for efficiency and economy in these days of D.D.T. and paludrine as it was before.

'Future programmes of instruction will depend very much on the success of this next course. Your active co-operation in securing attendance at this course will be of great help in ensuring their continuation, and is earnestly requested.

'It would be appreciated if agencies and firms would inform their managers and assistants that the course is being organized and encourage and assist them to attend. There is no fee.'

Notification of intention to attend the course should be sent as soon as possible to the Organizing Secretary, Ross Institute of Tropical Hygiene, Keppel Street, Gower Street, London, W.C. 1.

Candidates for Admission

The Council welcome communications to assist them in deciding whether the qualifications of candidates for admission into the Institution fulfil the requirements of the Bye-Laws. The application forms of candidates (other than those for Studentship) will be open for inspection at the office of the Institution for a period of at least two months from the date of the Bulletin *in which their applications are announced.*

The following have applied for transfer since 10th March, 1949:

To Membership—

John George Berry (*Ghatsila, India*).
Gustav Anthony Schnellmann (*Egremont, Cumberland*).
Patrick Francis Whelan (*Bristol, Gloucestershire*).

To Associate Membership—

Howard Frederick Burton (*Dartford, Kent*).
Cecil Basil Curtis (*Prestea, Gold Coast Colony*).
Denzil Vincent Sydney Dunn (*Filabusi, Southern Rhodesia*).
John Eddy Kernick (*Champion Reefs, India*).
Terence Albert Rodgers (*Vatukoula, Fiji*).

The following have applied for admission since 10th March, 1949:

To Membership—

William Reid (*Crossgates, Fifeshire*).

To Associate Membership—

Harold David Blackburn (*Newcastle-upon-Tyne, Northumberland*).
Donald Raymond Carlton (*Jos, Northern Nigeria*).
Fernleigh Edmondson (*Johannesburg, Transvaal*).
Ames Gresley Hellicar (*Chester, Cheshire*).
Kabool Chand Maithal (*Ruwanwella, Ceylon*).

To Studentship—

Harry Ivor Dunstan (*Camborne, Cornwall*).
Michael Andrew Grigg (*Thames Ditton, Surrey*).
Alexander Richard Hanvey (*Belfast, Northern Ireland*).
Charles Vivian Hickson (*Camborne, Cornwall*).
Colin McKechnie Marshall (*Dunedin, New Zealand*).
John Samuel Slaney (*London*).
Stanislaw Tokarski (*Camborne, Cornwall*).
James Brian Wall (*Camborne, Cornwall*).

News of Members

Members, Associate Members and Students are invited to supply the Secretary with personal news for publication under this heading.

Mr. H. C. T. Brown, *Associate Member*, has left Nigeria on his return to England.

Mr. A. Carstairs, *Associate Member*, has joined the staff of Settlingstones Mines, Ltd., Northumberland.

Mr. J. H. Chalk, *Associate Member*, has joined the staff of the Colonial Development Corporation.

Mr. A. Savile Davis, *Member*, expects to be in England in June.

Mr. R. W. Diamond, *Member*, has been awarded the Julian C. Smith medal of the Engineering Institute of Canada for 'achievement in the development of Canada'.

Mr. A. H. Douw, *Member*, has resigned from the staff of Goldfields Rhodesian Development Co., Ltd., and is now in private practice in Bulawayo.

Mr. D. Dunn, *Student*, has been transferred from the Bushtick mine to Killarney mine, Southern Rhodesia.

Mr. J. H. Gibbons, *Associate Member*, has moved to Bindura, Southern Rhodesia.

Mr. T. C. F. Hall, *Member*, is returning to England.

Mr. S. Haymes, *Associate Member*, has relinquished the position of manager of British Guiana Consolidated Goldfields, Ltd., and has joined Axel Johnson & Co., Inc., as general manager of their mining operations at Baramita in the North-West District of British Guiana.

Mr. JOHN HUNTER, *Associate Member*, has returned to the Gold Coast.

Mr. R. KENNETH MCLEOD, *Associate Member*, is returning to England from Tanganyika on six months' leave.

Mr. D. MCCORMICK, *Associate Member*, has left the Mines Department, New Zealand, and is now engaged as geologist to Australasian Petroleum Co., at Port Moresby, Papua.

Mr. J. M. H. O'REILLY, *Student*, has arrived in Malaya from New Zealand.

Mr. P. H. G. OWEN, *Associate Member*, is on leave in England from Southern Rhodesia.

Mr. G. C. PENGILLY, *Student*, has joined the staff of the Directorate of Opencast Coal Production, Ministry of Fuel and Power, Newcastle.

Mr. R. QUIRK, *Associate Member*, has been appointed assistant superintendent of Cerro de Pasco Copper Corporation, Morococha Department.

Mr. A. F. SKERL, *Associate Member*, has returned to Tanganyika Territory.

Mr. R. SMYTHE, *Associate Member*, has returned to India from England.

Mr. D. STANTON, *Member*, has left England on a visit to Sierra Leone.

Mr. G. M. STOCKLEY, *Member*, is returning to England on leave from Tanganyika Territory.

Mr. J. F. A. TAYLOR, *Student*, has left England to join the Department of Lands, Mines and Surveys, Fiji.

Mr. F. W. A. TIMMS, *Associate Member*, has arrived in England on leave from the Gold Coast.

Mr. E. H. TREGONING, *Associate Member*, has left England on a brief visit to the Rand and to the Northern Rhodesian copper belt, and expects to return early in May.

Mr. W. J. TRYTHALL, *Associate Member*, is returning this month to the United Kingdom on leave from the Gold Coast.

Addresses Wanted

A. Armstrong.
D. S. Broadhurst.
J. B. Cocking.
E. Dickson.
J. F. Durling.
K. A. Knight Hallowes.
R. B. Hicks.
G. H. Pinfield.
A. I. Scott.
A. Sloss.

OBITUARY

Hugh Frederick Marriott, a Past-President of the Institution, died on 2nd March, 1949, at his home in Northamptonshire at the age of 80. He was educated at Bishops Stortford Grammar School and from 1886 to 1891 was a student at the Royal College of Science and the Royal School of Mines, where he obtained the A.R.C.S. in Geology and the A.R.S.M. in Mining. He worked for six months on silver-lead and copper mines in the Spanish Pyrenees as assistant to Mr. P. W. Stuart-Menteath, and in May, 1892, was engaged as assistant to Mr. Hennen Jennings in the engineering department of Messrs. H. Eckstein & Co., Johannesburg, with whom he remained until his transfer in 1905 as consulting engineer to Messrs. Wernher, Beit & Co. and their successors, the Central Mining and Investment Corporation, Ltd. During this period he was responsible for putting down the first bore-hole that proved the extension in depth of the gold-bearing reefs of the Witwatersrand.

In 1923 Mr. Marriott set up in private practice, and acted as British Government representative at the Pan-Pacific Science Congress in Australia, and in the following year represented the Union of South Africa at the Empire Mining and Metallurgical Congress in London. He had held the position of Governor of the Imperial Institute. He was associated with the Panama Corporation Ltd., and was president of its successor, Panama Corporation (Canada) Ltd. Mr. Marriott was also British Government director of Magadi Soda Co., Ltd., and Somaliland Petroleum Co., Ltd.

He was elected to membership of the Institution in 1902 and served on the Council from 1907, being Vice-President for the period 1912–1915, and President of the Institution for the Session 1918–19. He was active in securing the Institution's Royal Charter granted in 1915.

Mr. Marriott contributed three papers to the *Transactions* of the Institution: 'Deep borehole surveying' (vol. 14, 1904–5); 'A record of an investigation of earth temperatures on the Witwatersrand goldfields and their relation to deep level mining in the locality' (vol. 15, 1905–6); and 'A visit to the mineral deposits of Canada', submitted jointly with Mr. William Frechville (vol. 15, 1905–6). He was awarded 'The Consolidated Gold Fields of South Africa, Ltd.' Gold Medal and Premium for 1905 in recognition of his researches on deep bore-holes and deep bore-hole surveying.

The Council regret to report the death of **Cecil Pearse**, *Member*, on 29th June, 1948; and **William George Wagner**, *Member*, on 19th March, 1949. Obituary notices will be published in a later issue of the *Bulletin*.

ADDITIONS TO JOINT LIBRARY OF THE INSTITUTION AND THE INSTITUTION OF MINING ENGINEERS

*Books (excluding works marked *) may be borrowed by members personally or by post from the Librarian, 424, Salisbury House, London, E.C. 2.*

Books and Pamphlets:

Bergbau-Archiv, band 9. Essen: Glückauf, 1948. 134 p., illus.

British Standards Institution. B.S. 1414: 1949, *flanged steel outside-screw-and-yoke wedge gate valves for the petroleum industry.* 29 p. 7s. 6d.; B.S. 1460: 1948, *precipitated calcium carbonate (determination of density after compaction).* 10 p. 2s. B.S. 1470: 1948, *wrought aluminium and aluminium alloys, sheet and strip.* 30 p. 3s. 6d. London: The Institution, 1948.

Canadian Institute of Mining and Metallurgy. *Structural geology of Canadian ore deposits.* A symposium arranged by a Committee of the Geology Division. Montreal: The Institute, 1948. 948 p., illus., maps, biblio. $10.

Carvel, John L. *The Coltness Iron Company: a study in private enterprise.* Privately printed by T. and A. Constable Ltd., Edinburgh. 200 p., illus.

Otago University, School of Mines and Metallurgy. *Annual bulletin and report of the director for the year 1948.* Dunedin, N.Z.: The University, 1948. 20 p.

Rhodesia. Chamber of Mines. *Ninth annual report for the year 1947.* Salisbury: The Chamber, 1948. 44 p.

Upsala University, Geological Institution. *Bulletin, vol. 32.* Upsala: The University, 1948. 483 p., illus.

*World Power Conference. *Statistical yearbook . . . no. 4: data on resources and annual statistics for 1936–1946.* Frederick Brown, ed. London: The Conference, 1948. 212 p. 45s.

Government Publications:

Canada, Bureau of Mines. *The Canadian mineral industry in 1946.* (Pub. no. 824.) Ottawa: Govt. Printer, 1948. 25 cents.

Canada, Bureau of Mines. *Summary of investigations on New Brunswick oil shales.* (Pub. no. 825.) Ottawa: Govt. Printer, 1948. 24 p., illus., maps. 15 cents.

Canada, Royal Commission on Coal. *Report of the . . . 1946.* Ottawa: King's Printer, 1947. 663 p., maps, tabs., diagrs.

Rodenwaldt, Ernst, senior author. *Hygiene: parts 2 and 3, preventive and industrial medicine, and epidemiology.* (*F.I.A.T. review of German science, 1939–1946.*) Germany: Military Govt., 1948. 213, 356 p. (German text).

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Institution of Mining Engineers. *Transactions, vol. 107, 1947–1948.* London: The Institution, 1948. 709 p., illus.

Maps:

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Chalco Lake, Northwest Territories. Geological survey, second preliminary map 48–20. Scale 2 in. = 1 ml. Ottawa: Dep. Mines and Resources, 1948.

Crowduck Bay, Manitoba. Geological survey, preliminary map 48–22. Scale: 2 in. = 1 ml. Ottawa: Dep. of Mines and Resources, 1948.

Taku River, British Columbia. Geological survey, map 931A. Scale: 1 in. = 2 ml. Ottawa: Dep. of Mines and Resources, 1948.

INDEX OF RECENT ARTICLES

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Bulletin de documentation technique no. 8.—*Inst. Hyg. Min.* (*Publ.*), Hasselt, Gen. 98, Feb. 15 1949, 20 p., tabs. diagrs. (Typescript.)

016 : 622.8

Bulletin de documentation technique no. 7.—*Inst. Hyg. Min.* (*Publ.*), Hasselt, Gen. 90, Nov. 30 1948, 23 p., tabs. (Typescript.)

3 ECONOMICS

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Conditions in the gold industry; further evidence to Commission.—*S. Afr. Min. Engng. J.*, J'burg, **59**, Pt. 2, 1949, Jan. 20, 643–6, tabs.; Feb. 5, 675–9, tabs.

338 Production

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Quelques aspects de la situation economique du Congo Belge. (Mineral production statistics from 1938.) G. Borgniez.—*Publ. Ass. Ing. Fac. Polyt. Mons*, **96**, Fasc. 3, 1948, 23–31, tabs.

338.5 Prices

338.5 : 553

Mineral and metal prices; an analysis of the relation of price to supply. R. F. Podmore.—*Min. Mag.*, Lond., **80**, Jan. 1949, 23–5. 1s. 6d.

338.5 : 553

The premium price plan—its cost and its results. (For copper, lead, zinc.) Henning E. Olund and Samuel A. Gustavson.—*Engng. Min. J.*, N.Y., **149**, Dec. 1948, 72–8, tabs. 50 cents.

53 PHYSICS

539.215.4

The classification of powder particle size by sieve. (Including a comparison of the B.S., D.I.N., I.M.M., and Tyler test sieves.) T. Burchell.—*Murex Rev.*, Rainham, Essex, **1**, No. 2, 1948, 33–8, tabs.

549 MINERALOGY

549.355.1

Solid solution of tetrahedrite in chalcopyrite and bornite. A. B. Edwards—*Proc. Aust. Inst. Min. Metall.*, Melbourne, Nos. 143–144, Sept.-Dec. 1946, 141–55, illus., biblio.

549.9(794)

Minerals of California. Joseph Murdoch and Robert W. Webb.—*Bull. Calif. Div. Min.* no. 136, San Francisco, June 1948, 402 p., illus., biblio.

55 GEOLOGY

55 (...) Regional

55(676.2/.9)

Geology of northern Kenya. Pt. 1—Geology and morphology of northern Kenya; pt. 2—Jurassic succession of north-east Kenya and the Juba river. F. Dixey.—*Rep. geol. Surv. Kenya*, no. 15, Nairobi, 1948, 43 p., illus., maps, biblio. 2s.

55(676.5)

A geological reconnaissance of the area west of Kitui township (with coloured geological map). J. J. Schoeman.—*Rep. geol. Surv. Kenya*, no. 14, Nairobi, 1948, 43 p., map, biblio. 2s.

55(711.14)

Taku river map-area, British Columbia. F. A. Kerr.—*Canad. Geol. Surv. Mem.* **248**, Ottawa, 1948, 84 p., illus., maps, diagrs., tabs., biblio. 25 cents.

55(713.31)

Preliminary report on Echo township, district of Kenora. H. S. Armstrong.—*Ont. Dep. Min. Rep.* P.R. 1948–10, Toronto, 1948, 5 p., maps.

55(713.8)

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55(716)

Londonderry and Bass River map-areas, Colchester and Hants counties, Nova Scotia. L. J. Weeks.—*Mem. geol. Surv. Canad.* no. 245, Ottawa, 1948, 86 p., illus., map. 25 cents.

55(718/9)

Bedrock geology of the seaboard region of Newfoundland Labrador. E. H. Kranck.—*Bull. geol. Surv. Newfoundland* no. 19, St. John's, 1939, 44 p., illus., maps.

55(719)

Geology and mineral deposits of the Fleur-de-Lys area. James Osborn Fuller.—*Bull. geol. Surv. Newfoundland.* no. 15, St. John's, 1941, 42 p. illus., maps, biblio.

55(721.2/.3)

Geology of the Sierra de los Muertos area, Mexico (with descriptions of aptian cephalopods from the La Pena formation). William E. Humphrey.—*Bull. Geol. Soc. Amer.*, Baltimore, Md., 60, Jan. 1949, 89–176, illus., maps, tabs., biblio. $1.

55(794)
Geological guidebook along highway 49—Sierran gold belt; the Mother Lode country. Olaf. P. Jenkins.—*Bull. Calif. Div. Min.* 141, San Francisco, Sept. 1948, 164 p., illus., maps, diagrs., biblio.

55(814.1)
A chamada serie Ribeira. (Conglomerates in the Iporanga district, Sao Paulo, Brazil.) Octavio Barbosa.—*Minerac. Metal.*, Rio de Janeiro, 13, Sept.-Oct. 1948, 187–9, diagrs. Cr$15.00.

550.8 Prospecting

550.8
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Radio-active mineral deposits.—*Miner. Res. Aust. Pamp.* no. 3, Canberra, Aug. 9 1948, 25 p., illus., diagr.

550.8–982(422.5)
Surface problems in the search for oil in Sussex. J. W. Reeves.—*Proc. Geol. Ass.*, Lond., 59, Pt. 4, 1948, 234–69, maps, diagrs., tabs., biblio. 5s.

550.8–982(44)
Orientation et formation necessaires aux recherches de pétrole; application à la France. G. Brognon.—*Publ. Ass. Ing. Fac. Polyt. Mons*, 96, 1948, Fasc. 2, 18–32, diagrs., tabs.; Fasc. 3, 17–22, map, diagrs.

550.81
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551.4 Physiography

551.4 : 55(71)
Physiography of the Canadian Cordillera, with special reference to the area north of the fifty-fifth parallel. H. S. Bostock.—*Mem. geol. Surv. Canad.* no. 247, Ottawa, 1948, 106 p., illus., maps. 25 cents.

552 PETROLOGY

552.3 : 413.164
The history of a name; the word porphyry, with some reflections on igneous rock terminology. T. C. F. Hall.—*Min. Mag.*, Lond., 80, Feb. 1949, 73–9, biblio. 1s. 6d.

553 ECONOMIC GEOLOGY

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Mineral position of ECA nations: no. 10—Great Britain. Frederick R. Brewster.—*Engng. Min. J.*, N.Y., 150, Jan. 1949, 61–4, map, tabs.

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Mineral position of ECA nations: No. 11—Greece. O. Perry Riker.—*Engng. Min. J.*, N.Y., 150, Jan. 1949, 65–6, map, tabs.

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Geology and mineral deposits of the Fleur-de-Lys area. James Osborn Fuller.—*Bull. geol. Surv. Newfoundland* no. 15, St. John's, 1941, 42 p., illus., maps, biblio.

553(73)
A mineral policy for the United States. Elmer W. Pehrson.—*Bull. Min. Metall. Soc. Amer.* 285, N.Y., 41, Dec. 1948, 57–76.

553(94)
The mineral resources and industries of the Commonwealth of Australia and the Mandated Territory of New Guinea. H. G. Raggatt, P. B. Nye and N. H. Fisher.—*Proc. Aust. Inst. Min. Metall.*, Melb., Nos. 143–144 Sept.-Dec. 1946, 188–282, tabs.

553(95)
The mineral resources and industries of the Commonwealth of Australia and the Mandated Territory of New Guinea. H. G. Raggatt, P. B. Nye and N. H. Fisher.—*Proc. Aust. Inst. Min. Metall.*, Melb., Nos. 143–144, Sept.-Dec. 1946, 188–282, tabs.

553.1 Determination, properties, of economic minerals

553.1 : 545.82
Method for the spectrochemical determination of beryllium, cadmium, zinc, and indium in ore samples. Graham W. Marks and Betsy M. Jones.—*U.S. Bur. Min. Rep. Invest.* 4363, Wash., D.C., Nov. 1948, 27 p., diagrs., tabs.

553.1 : 549.282
A method for spectrochemical determination of silver in ore samples. Graham W. Marks and E. V. Potter.—*U.S. Bur. Min. Rep. Invest.* 4377, Wash., D.C., Dec. 1948, 14 p., illus., diagrs., tabs.

553.1 : 553.31(44)
Sur le dosage du fer dans les minerais de fer du Bassin Lorrain. J. Baron.—*Chim. Anal.*, Paris, 31, Feb. 1949, 29–30.

553.1 : 553.494.6
Determination of lithium in rocks by distillation. (Quantitative extraction based on a high temperature volatisation procedure.) Mary H. Fletcher.—*Analyt. Chem.*, Easton, Pa., 21, Jan. 1949, 173–5, illus., biblio. 50 cents.

553.2 Ore deposition

553.2 : 553.441(791.14 Seventy Nine)
Structural control and mineralization at the Seventy Nine mine, Gila county, Arizona. (Lead, zinc sulphide ore occurring as replacement veins in North dike.) George A. Kiersch.—*Econ. Geol.*, Lancaster, Pa., 44, Jan.-Feb. 1949, 24–39. $1.

553.26(755/6)
Investigation of the Hamme tungsten district, Vance county, N.C., and Mecklenburg county, Va. Frank K. McIntosh.—*U.S. Bur. Min. Rep. Invest.* 4380, Wash., D.C., Nov. 1948, 6 p., illus., maps, diagrs., tabs.

553.3/.4 Metalliferous deposits

553.31(794)

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553.311.1(798.11 Tolstoi Mtn.)

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553.411(675.55 Senzere)

Congo Belge; la mineralisation aurifère de la mine de Senzere (Kilo-Moto).—*Chron. Min. Colon.*, Paris, **11**, May 1912, 102.

553.43(711.36 Copper Mountain)

Geology of the Copper Mountain mine. Keith C. Fahrni.—*W. Miner*, Vancouver, B.C., **21**, Dec. 1948, 70–1. 50 cents.

553.43(755.42 Sutherland)

Investigation of the Sutherland copper prospect, Floyd county, Va. Wesley A. Grosh.—*U.S. Bur. Min. Rep. Invest.* 4357, Wash., D.C., Oct. 1948, 2 p., map.

553.43(755.42 Toncrae-Howard)

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553.441(791.14 Seventy Nine)

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553.461(798.4 Knik Valley)

Investigation of Knik Valley chromite deposits, Palmer, Alaska. Stuart Bjorklunde and W. S. Wright.—*U.S. Bur. Min. Rep. Invest.* 4356, Wash., D.C., Oct. 1948, 5 p., maps, diagrs., biblio.

553.464(761.2/3)

Flake-graphite and vanadium investigation in Clay, Coosa, and Chilton counties, Ala. Hugh D. Pallister and J. R. Thoenen.—*U.S. Bur. Min. Rep. Invest.* 4366, Wash., D.C., Dec., 1948, 84 p., maps, biblio.

553.481(795.19)

Variations in garnierite from the nickel deposit near Riddle, Oregon. W. T. Pecora, S. W. Hobbs, and K. J. Murata.—*Econ. Geol.*, Lancaster, Pa., **44**, Jan.-Feb. 1949, 13–23, biblio. $1.

553.487

Cobalt and its uses. (Occurrences.)—*S. Afr. Min. Engng. J.*, J'burg., **59**, Pt. 2, Jan. 15 1949, 603–4. 6d.

553.494.3(81)

Zircônio: Historico, aplicações e ocorrencias. (With special reference to Brazil.) Resk Frayha. *Minerac. Metal.*, Rio de Janeiro, **13**, Sept.-Oct. 1948, 182–6, biblio. Cr$15.00.

553.495

Radio-active mineral deposits.—*Miner. Res. Aust. Pamp.* no. 3, Canberra, Aug. 9 1948, 25 p., illus., diagr.

553.495.2 (713.1 Theano Point)

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553.499(798.6 Rainy Creek)

Investigation of Rainy Creek mercury prospect, Bethel district, Kuskokwim region, south-western Alaska. F. A. Rutledge.—*U.S. Bur. Min. Rep. Invest.* 4361, Wash., D.C., Oct. 1948, 5 p., maps, diagrs.

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553.61(781)

Central Kansas clay deposits. W. D. McMillan and Arthur O. Wilson.—*U.S. Bur. Min. Rep. Invest.* 4379, Wash., D.C., Dec. 1948, 38 p., maps, diagrs., tabs.

553.611

Clays and their use in the ceramic industry and other industries.—*S. Aust. Dep. Mines Min. Rev.*, Adelaide, No. 86, 1948, 14–20.

553.611(942 Arckaringa)

Arckaringa (Imbitcha bore) clay deposits. L. W. Parkin.—*S. Aust. Dep. Mines Min. Rev.*, Adelaide, No. 86, 1948, 102–5, illus.

553.611(942 Birdwood)

Clay mine, Birdwood. L. W. Parkin.—*S. Aust. Dep. Mines Min. Rev.*, Adelaide, No. 86, 1948, 106–7.

553.611.2(942 Lobethal)

Clay and sand deposit, Lobethal. L. W. Parkin.—*S. Aust. Dep. Mines Min. Rev.* Adelaide, No. 86, 1948, 107–8.

553.611.6(764)

Texas white firing bentonite. (Geology, properties, identification, uses.) Forrest K. Pence.—*Min. Engng.*, N.Y., **1**, Jan. 1949, Sec. 5, 27–32, illus., map, diagrs., tabs. 75c.

553.621 : 537.228.1(91)

Piezo-electric quartz. F. Canavan.—*Miner. Res. Aust. Bull.* no. 10 Canberra, 1947, 12 p., diagrs.

553.621 : 537.228.1(94)

Notes on the occurrence of piezo-electric quartz in Australia with special reference to the Kingsgate field. M. D. Garretty.—*Miner. Res. Aust. Bull.* no. 13, Canberra, 1947, 21 p., maps, tab., biblio.

553.623 : 621.742 : 519.621.1

Olivine synthetic moulding sand controls silicosis. George Allen.—*Canad. Metals*, Toronto, **12**, Jan. 1949, 19; **36**, biblio. 25 cents.

553.625(94)

A study of Australian diatomites with special reference to their possible value as filter media. Irene Crespin.—*Miner. Res. Aust. Bull.* no. 7, Canberra, 1947, 40 p., illus., map, tabs., biblio.

553.635.1

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622.235.116(73)
Un procédé américain de tir du bouchon. (Description of the 'Burn cut'.) G. Mathon. —*Mines*, Paris, No. 1, 1948, 14–22, diagrs.

622.235.12 : 622.233.626
Blasthole stoping evaluated; diamond drilling for breaking ore is here analyzed on the basis of an extensive survey by questionnaire. Vinton H. Clarke and John V. Beall.—*Min. Engng.*, N.Y., 1, Jan. 1949, Sec. 1, 15–25, diagrs. 75 cents.

622.235.12 : 622.233.626(689.55)
Blasthole drilling at Nkana 1944–1947. (The work of the Roto Drill dept., and the application of diamond drilling to stoping, pillar recovery and hanging-wall caving.) O. B. Bennett.—*J. Chem. Soc. S. Afr.*, J'burg, 49, Oct. 1948, 95–153, illus., diagrs., tabs. 6s.

622.235.24
Shaped charges applied to mining: Pt. 1. Drilling holes for blasting. Hiram C. Draper and others.—*U.S. Bur. Min. Rep. Invest.* 4371, Wash., D.C., Nov. 1948, 12 p., illus., tabs., diagrs.

622.235.432
Higher quarry profits with delayed action blasting. (Using an electric motor powered blasting switch.) L. F. Miller.—*Excavat. Engr.*, S. Milwaukee, 43, Feb. 1949, 14–19; 51, illus. 35 cents.

622.24 Exploration and well drilling

622.243.414
A study of drilling motion and its application in cable tool blast hole drills. N. A. Webster.—*Quarry Mgrs.' J.*, Lond., 32, Feb. 1949, 412–24, diagrs. 1s. 6d.

622.243.6.051.3(881)
Diamond drilling in the Marudi mountain area British Guiana. (Describes means used to penetrate the deep mantle of overburden.) Keith O'Donnell.—*Canad. Min. J.*, Gardenvale, P.Q., 69, Dec. 1948, 70–3, illus., map. $1.

622.243.6.051.3 : 622.343
Diamond drilling at Union Copper mine, Cabarrus and Rowan counties, N.C. T. J. Ballard and Austin B. Clayton.—*U.S. Bur. Min. Rep. Invest.* 4364, Wash., D.C., Oct. 1948, 9 p., map, diagrs., tabs.

622.243.8
Development of the Calyx core drill: Pt. 1.—*Min. J.*, Lond., 232, Feb. 12 1949, 117.

622.25 Shafts, sinking

622.25(796.311)
Shaft-sinking practice in the Coeur d'Alene. (Mechanical methods applied to sinking of vertical and inclined shafts.) R. W. Lottridge and R. W. Neyman.—*Min. Congr. J.*, Wash., D.C., 35, Jan. 1949, 18–23, illus., diagrs. $0.30.

622.252 : 622.233.626
Shaft sinking by diamond drilling, Bellefonte mine, National Gypsum Co., Centre county, Pa. McHenry Mosier.—*U.S. Bur. Min. Inform. Circ.* 7477, Wash., D.C., Oct. 1948, 6 p., illus., diagr.

622.252.6(747.26 Mineville)
Mucking Fisher Hill shaft with a scraper. (Deepening an inclined shaft, 10 × 30 ft. at an angle of minus 33° 30′ in hard granite rock.) R. J. Linney.—*Engng. Min. J.*, N.Y., 149, Dec. 1948, 92–3, illus., diagrs. 50 cents.

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Core-drilled shaft for ventilation and escapement. (48 ft. in diameter, 202 ft. deep.) Paul T. Porter.—*Min. Congr. J.*, Wash., D.C., 35, Jan. 1949, 50–1, illus. $0.30.

622.258.4(682.3 St. Helena)
Sinking No. 3 incline shaft, St. Helena Gold Mines, Ltd. R. H. MacWilliam.—*Ass. Mine Mgrs. Transvaal Circ.* 1/49, J'burg, Jan. 14 1949, 17 p., tabs., diagrs. (Typescript.)

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Mucking Fisher Hill shaft with a scraper. (Deepening an inclined shaft, 10 × 30 ft. at an angle of minus 33° 30′ in hard granite rock.) R. J. Linney.—*Engng. Min. J.*, N.Y., 149, Dec. 1948, 92–3, illus., diagrs. 50 cents.

622.27 Mining methods

622.271.2
Modern quarrying equipment for the production of aggregates, including ballast, gravel and sand. R. Eric Pochin and F. W. Parker.—*Cement Lime Grav.*, Lond., 23, Jan. 1949, 230–5, illus.

622.273.2.047
Sand-slime stope filling proves satisfactory. (Use of *minus* 200-mesh from flotation mill.) Richard Krebs and J. C. O'Donnell.—*Engng. Min. J.*, N.Y., 150, Jan. 1949, 54–60, illus., diagrs., tabs.

622.28 Support, linings

622.28
Ground control—theory and practice. Jack Spalding.—*Bull. Instn. Min. Metall.*, Lond., 507, Feb. 1949, 1–35, diagrs., biblio. (Paper 2s. 6d.)

622.32 Oilfield production

622.323(73)
Report of petroleum and natural-gas division fiscal year 1947. R. A. Cattell and others.—*U.S. Bur. Min. Inform. Circ.* 7484, Wash., D.C., Dec. 19 1948, 65 p., illus.

622.323(764 Lake Creek)
Petroleum-engineering study of the Lake Creek field, Montgomery county, Tex. H. B. Hill and Felix A. Vogel, Jr.—*U.S. Bur. Min. Rep. Invest.* 4319, Wash., D.C., Nov. 1948, 65 p., illus., maps, tabs., diagrs.

622.34 Metalliferous mining

622.341.1(713.51 Steep Rock)
The Steep Rock iron mine. Geo. E. Cole.—*W. Miner*, Vancouver, B.C., 21, Dec. 1948, 98–104. 50 cents.

622.341.1(792.19) : 622.013.36
Investigation of iron-ore reserves of Iron county, Utah. Paul. T. Allsman.—*U.S. Bur. Min. Rep. Invest.* 4388, Wash., D.C. (Suppl. to *Rep. Invest.* 4076), Nov. 1948, 3 p.

622.341.111(749.2 West Portal)

West Portal magnetite mines, Hunterdon county, N.J. G. E. Botsford and McHenry Mosier.—*U.S. Bur. Min. Rep. Invest.* 4352, Wash., D.C., Oct. 1948, 11 p., map, tabs.

622.341.24(711.19 Silver Giant)

The Silver Giant mine. C. M. Campbell.—*W. Miner*, Vancouver, B.C., **21**, Dec. 1948, 30. 50 cents.

622.341.24(711.38 Lucky Jim)

Zincton has consistent record. (Silver, lead zinc, Lucky Jim property—mining and milling.) J. S. McIntosh.—*W. Miner*, Vancouver, B.C., **21**, Dec. 1948, 126–30, flowsheet. 50 cents.

622.341.24(711.38 Van Roi)

High-grade ore at Van Roi.—*W. Miner*, Vancouver, B.C., **21**, Dec. 1948, 92. 50 cents.

622.341.24(712.1 United Keno Hill)

United Keno Hill. (History, geology, mining.)—*W. Miner*, Vancouver, B.C., **22**, Jan. 1949, 37–40, illus. 25 cents.

622.341.24(786.23 Minah)

Investigation of the Minah lead-silver mine, Jefferson county, Mont. S. H. Lorain and R. J. Hundhausen.—*U.S. Bur. Min. Rep. Invest.* 4359, Wash., D.C., Oct. 1948, 9 p., map, tabs., diagrs.

622.342.1(678.2/.9)

Tanganyika's gold industry ; steps towards higher production.—*S. Afr. Min. Engng. J.*, J'burg, **59**, Pt. 2, Dec. 25 1948, 499–501. 6d.

622.342.1(711.391 Vananda)

New copper-gold producer.—*W. Miner*, Vancouver, B.C., **21**, Dec. 1948, 148–9. 50 cents.

622.342.1(713.61 Renabie)

Renabie mines, Missinable's pioneer gold producer. (Geology, mining, milling.) H. R. Rice.—*Canad. Min. J.*, Gardenvale, P.Q., **70**, Jan. 1949, 61–7, diagrs., tabs. $1.

622.342.1(942 Westward Ho)

Westward Ho gold mine. A. T. Armstrong. —*S. Aust. Dep. Mines Min. Rev.*, Adelaide, No. 86, 1948, 109–11.

622.342.2(711.151 Torbrit)

Torbrit starts production. (Geology, mining and milling at Torbrit silver mine.) H. D. Forman and A. G. Roach.—*W. Miner*, Vancouver, B.C., **21**, Dec. 1948, 88–90. 50 cents.

622.342.2(711.201 Silver Standard)

Silver Standard in production.—*W. Miner*, Vancouver, B.C., **21**, Dec. 1948, 146–7. 50 cents.

622.343(711.391 Vananda)

New copper-gold producer.—*W. Miner*, Vancouver, B.C., **21**, Dec. 1948, 148–9. 50 cents.

622.343(711.41 Britannia Beach)

A visit to Britannia.—*W. Miner*, Vancouver, B.C., **21**, Dec. 1948, 76–80, map, flowsheets. 50 cents.

622.343(712.7 Flin Flon)

Largest western copper producer. (Geology and mining at Flin Flon.) Geo. E. Cole.—*W. Miner*, Vancouver, B.C., **21**, Dec. 1948, 62–7. 50 cents.

622.343(712.7 Lynn Lake)

Nickel-copper developments at Lynn Lake. Geo. E. Cole.—*W. Miner*, Vancouver, B.C., **21**, Dec. 1948, 86–7, map. 50 cents.

622.343(755/6)

Investigation of the Virgilina copper district, Virginia and North Carolina. A. W. Newberry and others.—*U.S. Bur. Min. Rep. Invest.* 4384, Wash., D.C., Dec. 1948, 12 p., maps, diagrs.

622.343(756.124 Union)

Diamond drilling at Union Copper mine, Cabarrus and Rowan counties, N.C., T. J. Ballard and Austin B. Clayton.—*U.S. Bur. Min. Rep. Invest.* 4364, Wash., D.C., Oct. 1948, 9 p., map, diagrs., tabs.

622.343(793.25 Majuba Hill)

Investigation of Majuba Hill copper-tin mine, Pershing county, Nev. E. J. Matson.—*U.S. Bur. Min. Rep. Invest.* 4378, Wash., D.C., Dec. 1948, 10 p., maps, tabs.

622.343(798.11 Salt Chuck)

Investigation of the Salt Chuck copper mine, Kasaan peninsula, Prince of Wales Island, south-eastern Alaska. S. P. Holt and others.—*U.S. Bur. Min. Rep. Invest.* 4358, Wash., D.C., Oct. 1948, 16 p., maps, tabs., diagrs.

622.344.1(711)

Greatest western mining enterprise. (Mining, mineral dressing and metallurgy at the Consolidated Mining & Smelting Co. of Canada, Ltd.)—*W. Miner*, Vancouver, B.C., **21**, Dec. 1948, 54–61, illus. 50 cents.

622.344.1(711.25 Reeves MacDonald)

B.C.'s next base-metal producer. (Mining and milling at Reeves MacDonald Mines Ltd.)—*W. Miner*, Vancouver, B.C., **21**, Dec. 1948, 81–3. 50 cents.

622.344.1(773.52 Royal Princess)

Investigation of Royal Princess zinc-lead deposit, Jo Daviess county, Ill. Stephen P. Holt.—*U.S. Bur. Min. Rep. Invest.* 4386, Wash., D.C., Dec. 1948, 13 p., map, tabs., diagrs.

622.344.6(775.36 Vial)

Investigation of Vial zinc mine, Iowa county, Wis. Alvin M. Cummings.—*U.S. Bur. Min. Rep. Invest.* 4385, Wash., D.C., Dec. 1948, 7 p., map, tabs.

622.345(795.25 Majuba Hill)

Investigation of Majuba Hill copper-tin mine, Pershing county, Nev. E. J. Matson.—*U.S. Bur. Min. Rep. Invest.* 4378, Wash., D.C., Dec. 1948, 10 p., maps, tabs.

622.346(496)

Turkish chromite production up. (Mining and production.) Falih Ergunalp.—*Min. World, San Francisco*, **10**, Nov. 1948, 43. 35 cents.

622.346(748.45 Wood)

Investigation of Wood chromite mine area, Lancaster county, Pa. Frank K. McIntosh and McHenry Mosier.—*U.S. Bur. Min. Rep. Invest.* 4383, Wash., D.C., Dec. 1948, 5 p., maps, tabs.

622.346(786.13)

Investigation of the Four Chromes and other chromite deposits, Red Lodge district, Carbon county, Mont. J. A. Herdlick.—*U.S. Bur. Min. Rep. Invest.* 4369, Wash., D.C., Dec. 1948, 13 p., maps.

622.346(786.311)

Investigation of chromite deposits of the Stillwater complex, Stillwater and Sweetgrass counties, Mont. N. L. Wimmler.—*U.S. Bur. Min. Rep. Invest.* 4368, Wash., D.C., Dec. 1948, 41 p., maps.

622.346(794.18 French Hill)

Investigation of French Hill chromite mine, Del Norte county, Calif. W. O. Sanborn and Spangler Ricker.—*U.S. Bur. Min. Rep. Invest.* 4365, Wash., D.C., Oct. 1948, 9 p., map, tabs.

622.346.3(711.25 Emerald)

Canada's leading tungsten producer. (Production 5,000 units of concentrate monthly.)—*W. Miner*, Vancouver, B.C., **21**, Dec. 1948, 106–9. 50 cents.

622.346.3(756.218)

North Carolina, an important tungsten producer. (250 tons per day; average grade to mills assays 0·9% WO_3. Concentration flow sheet.) M. H. Kline and F. K. McIntosh. —*Engng. Min. J.*, N.Y., **149**, Dec. 1948, 79–81, flowsheet. 50 cents.

622.355.11(797.38 Sauk Mt.)

Investigation of the Sauk mountain limestone deposits Skagit county, Wash. Constantine C. Popoff.—*U.S. Bur. Min. Rep. Invest.* 4355, Wash., D.C., Oct. 1948, 14 p., maps, tabs.

622.36 Mining non-metals

622.362.1 : 537.228.1

Importance and application of piezoelectric minerals. Hugh H Waesche.—*Min. Engng.*, N.Y., **1**, Jan. 1949, Sec. 3, 12–16, biblio. 75 cents.

622.362.355.1(943)

A progressive south coast industry; mining the beach sands of Queensland. O. J. Carlson.—*Qd. Govt. Min. J.*, Brisbane, **49**, Dec., 1948, 476–82, illus. 6d.

622.366(52.1 Matsuo)

Matsuo sulphur mine may become an open pit. (Mining at sulphur-marcasite ore deposit, Honshu, Japan.) Pomeroy C. Merrill.—*Engng. Min. J.*, N.Y., **150**, Jan. 1949, 73–5, illus., diagrs. 50 cents.

622.366.121(481)

Pyrite mines of Norway.—*Min. World*, Lond., **156**, Feb. 19 1949, 102. 6d.

622.366.121(481)

Norwegian pyrites. (Prospects and reserves.) —*Min. J.*, Lond., **232**, Feb. 26 1949, 147. 8d.

622.367.6(71)

The Canadian asbestos industry; facts about the world's greatest asbestos mines. (Production, flowsheet.)—*Rhod. Min. J.*, J'burg, **20**, 373–5, flowsheet. 6d.

622.367.6(942 Hundred of Apoinga)

Asbestos mine—Hundred of Apoinga. A. T. Armstrong.—*S. Aust. Dep. Mines Min. Rev.*, Adelaide, No. 86, 1948, 111–12.

622.368.912(756/7)

New barite possibilities revealed in the Carolines. (Geology, mining, beneficiation.) Earl C. Van Horn.—*Engng. Min. J.*, N.Y., **150**, Jan. 1949, 76–8, map.

622.368.912(942 Blinman)

Blinman barytes mine—Oraparinna station (near the Bunker). L. L. Mansfield.—*S. Aust. Dep. Mines Min. Rev.*, Adelaide, No. 86, 1948, 113–4.

622.368.912(942 Hundred of Julia Creek)

Barytes mine—Hundred of Julia Creek. L. L. Mansfield.—*S. Aust. Dep. Mines Min. Rev.*, Adelaide, No. 86, 1948, 115–7, map.

622.41/6 Ventilation

622.413.4(682.3 Crown Mines)

Rand air-cooling plant. (Ammonia refrigeration at Crown mines.)—*Min. Mag.*, Lond., **80**, Jan. 1949, 20–2, illus., diagr. 1s. 6d.

622.413.5 : 545.75

An improved type of konimeter. J. P. Rees and S. R. Rabson.—*Bull. Instn. Min. Metall.*, Lond., No. 508, March 1949, 21–4, illus., diagr. (Paper 1s.)

622.414

Underground anemometry. (The results of a questionnaire on the use of anemometers.) Cloyd M. Smith.—*Min. Engng.*, N.Y., **1**, Jan. 1949, Sec. 3, 1–11, biblio. 75 cents.

622.6 Haulage, hoisting, transport

622.625.28–831

Electric trolley locomotives. Marcel Tatin. —*Iron Coal Tr. Rev.*, Lond., **158**, Feb. 4 1949, 229–33, diagrs., tabs. 9d.

622.625.28–835

Battery locomotive haulage at Tennessee Copper's mines. (Cost comparison.) Lamar Weaver.—*Engng. Min. J.*, N.Y., **149**, Dec. 1948, 88–91, illus. 50 cents.

622.67

Increasing the capacity of mine shafts. (Shuttle car cage winding.) B. W. Hewitt.—*Sheffield Univ. Min. Mag.*, Sheffield, 1948, 57–63, diagrs., biblio. 2s. 6d.

622.67 : 622.016.8

Deep shaft winding problems. (Survey of various designs of winding engines in use and shaft guides, and their effect on winding ropes in relation to depth.) H. Hitchen.—*J. S. Afr. Instn. Engrs.*, J'burg, **47**, Nov. 1948, 59–91, diagrs., tabs. 3s. 6d.

622.67.078.38(943 Golden Plateau)

Testing safety catches on cages at Golden Plateau mine, Cracow. O. Anderson.—*Qd. Govt. Min. J.*, Brisbane, **49**, Dec. 1948, 488–9, diagr. 6d.

622.671–83

Developments in electric winding technique. R. W. Worrall.—*Sheffield Univ. Min. Mag.*, Sheffield, 1948, 22–33, illus. 2s. 6d.

622.673.2

Construction of headframe and surface plant No. 1 shaft, Jeffrey mine, Canadian Johns-Manville Co., Limited. (A reinforced concrete headframe.) K. V. Lindell and Chas. D. Borror.—*Canad. Min. Metall. Bull.* **440** (*Trans.* **51**, 293–300), Montreal, Dec. 1948, 674–81, diagrs., tabs. $1.

622.673.6 : 620.1

Wire rope unit tests. Richard Saxton.—*Min. J.*, Lond., **232**, Feb. 5 1949, 99. 8d.

622.7 MINERAL DRESSING

622.7–... Dressing specific minerals

622.7–337.1

Laboratory study of the hot-water process for separating hydrocarbons from surface deposits of bituminous sandstones near Edna, Calif. G. B. Shea and R. V. Higgins.—*U.S. Bur. Min. Rep. Invest.* **4246**, Wash., D.C., Oct. 1948, 31 p., illus., diagrs.. tabs., flowsheet.

622.7–341.1(425.3)

Modernisation of British iron and steel works ; ore-preparation plant at Stewarts and Lloyds Limited, Corby. Tom Bishop.—*Iron Coal Tr. Rev.*, Lond., **158**, March 4 1949, 457–60, illus., flowsheet. 9d.

622.7–341.1(776.72)

The Mesabi taconite problem. H. C. Ross.—*Canad. Min. J.*, Gardenvale, P.Q., **69**, Dec. 1948, 57–61, map, diagrs., flowsheets, biblio. $1.

622.7–342.1(713.714 Wright-Hargreaves)

Recent improvements in milling practice at Wright-Hargreaves. Malcolm Black.—*Canad. Min. Metall. Bull.* 440 (*Trans.* **51**, 286–92), Montreal, Dec. 1948, 667–73, illus., flowsheet. $1.

622.7–343

Custom milling, American Zinc, Lead and Smelting Company, Ouray, Colorado. D. C. McLean and others.—*Deco Trefoil*, Denver, **12**, Nov.-Dec. 1948, 5–12, illus., flowsheet.

622.7–343(711.36 Copper Mountain)

Increasing the capacity of the Allenby concentrator. (Concentrator for Copper Mountain, B.C.) L. H. McKay.—*W. Miner*, Vancouver, B.C., **21**, Dec. 1948, 72. 50 cents.

622.7–343(712.7 Flin Flon)

Concentrating, smelting and research at the Flin Flon mine. Geo. E. Cole.—*W. Miner*, Vancouver, B.C., **22**, Jan. 1949, 48–52, flowsheet. 25 cents.

622.7–343(791.2 Superior)

A mill designed for easier operation. (1,500-ton concentrator, Magma Copper Co., Superior, Arizona.) Halder J. Rex.—*Engng. Min. J.*, N.Y., **149**, Dec. 1948, 68–71, illus. 50 cents.

622.7–344.1

Custom milling, American Zinc, Lead and Smelting Company, Ouray, Colorado. D. C. McLean and others.—*Deco Trefoil*, Denver, **12**, Nov.-Dec. 1948, 5–12, illus., flowsheet.

622.7–362.3 : 666

Some glass sands from New Jersey. (Beneficiation, analysis of treated sands, statistics.) Albert S. Wilkerson and Jay E. Comeforo.—*Econ. Geol.*, Lancaster, Pa., **44**, Jan.-Feb. 1949, 63–7. $1.

622.7–363.412

Concentration of fluorite from tailings and mine waste rock in Crittenden and Livingston counties, Ky. M. M. Fine.—*U.S. Bur. Min. Rep. Invest.* 4370, Wash., D.C., Dec. 1948, 21 p., tabs.

622.7–364.1

Florida phosphate milling. John E. Kelly.—*Min. World, San Francisco*, **10**, Dec. 1948, 35–7 ; 40, flowsheets. 35 cents.

622.7.006.25

Laboratory ore testing procedure. Clarence Thom and Frank A. Seeton.—*Deco Trefoil*, Denver, **13**, Jan.-Feb. 1949, 5–8, illus., diagrs.

622.7.076

Concentration of fluorite from tailings and mine waste rock in Crittenden and Livingston counties, Ky. M. M. Fine.—*U.S. Bur. Min. Rep. Invest.* 4370, Wash., D.C., Dec. 1948, 21 p., tabs.

622.74 Screening

622.74

Screens and separators ; the rotary screen, mechanically and electrically vibrated screen, air and unit air separator. J. McBride.—*Edgar Allen News*, Sheffield, **27**, 1949, Feb., 221–5, illus., diagrs. ; March, 244–5, diagr.

622.741

Précisions sur les méthodes d'études par dispersion des phénomènes intéressant le classement par densité dans les installations de préparation mécanique. Jacques Turpin and André Pincon.—*Rev. Industr. Min.*, St. Etienne, No. 541 (Section Mine), Feb. 1949, 67–81, diagrs.

622.742 : 621.928.23

Das Resonanzschwingsieb im Bergbau und seine Entwicklung im letzten Jahrzehnt. (The resonance shaker screen in mining and its development in recent years.) Gerhard Linke.—*Glückauf*, Essen, **85**, Jan. 15 1949, 44–51, illus., diagrs. 4 DM.

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Recovery of Sulphur from Smelter Gases by the Orkla Process at Rio Tinto*

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INTRODUCTION

NUMEROUS patents have been granted for processes designed to recover sulphur from industrial gases, but so far as the authors are aware only two or three of these processes have been successfully worked on a large scale ; the object of this paper is to describe one that has given very successful results in at least three different countries—namely, the process which was worked out in Norway by Mr. N. E. Lenander of the Orkla Grube Aktiebolag and which has the distinguishing feature of recovering sulphur as a by-product of copper smelting.

Sulphur was first successfully recovered as a by-product from the blast-furnace smelting of pyritic copper ore at a small plant at Lokken, Norway, about the year 1928. The success of this pilot plant led to the construction of a large modern smelter with four blast-furnaces at Thamshaven, near Lokken, which was completed about 1932.

The first unit (a single blast-furnace) of the Rio Tinto plant went into production in August, 1930, and the plant has since been gradually expanded by the modification of two more furnaces of the original smelter, so that there are now, in all, three furnaces specially equipped for the recovery of sulphur.

Reference will, from time to time, be made to the plants of both the Orkla Company, Norway, and of Mina de S. Domingos, Portugal, operated by Mason & Barry, Ltd., but the main purpose of these notes is to describe the work at Rio Tinto, as the conditions are, in certain respects, markedly different from those prevailing at Orkla.

For convenience of presentation the paper is divided into three sections—namely, Section 1—Principles of the process ; Section 2—The plant ; Section 3—Practice of the process.

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SECTION 1—PRINCIPLES OF THE PROCESS

It will be remembered that in the smelting of pyritic copper ores the loose atom sulphur is distilled off at the top of the charge column and burns in atmospheric air. The iron monosulphide descends and at or near the focus burns in the oxygen of the blast, forming iron oxide (which is at once slagged) and sulphur dioxide gas; unoxidized monosulphide, together with the sulphide of copper, forms the matte, which also contains some magnetite and the sulphides of lead, zinc, etc. The authors have referred to the compound remaining after the distillation of the loose atom sulphur as monosulphide of iron, but actually this compound is generally considered to be Fe_nS_{n+1}, in which n is generally taken to be about 7. The reactions then are:

$$7FeS_2 = Fe_7S_8 + 3S_2 \qquad (1)$$

$$2Fe_7S_8 + 23O_2 = 14FeO + 16SO_2 \qquad (2)$$

It has long been realized that under certain conditions it is possible effectively to reduce SO_2 by solid carbon according to the equation—

$$SO_2 + C = CO_2 + \tfrac{1}{2}S_2 \qquad (3)$$

The inventor of the process therefore conceived the idea that if an ordinary blast-furnace, primarily designed for the smelting of sulphide ores to matte, could be fitted with a closed top very similar to that used on a conventional iron blast-furnace and that if, furthermore, a substantial excess of coke were added to the charge, two results would follow: First, the loose atom sulphur would not burn, as there would be no oxygen present and, secondly, the SO_2 produced at the focus would be reduced in the course of its ascent through the charge column, by the coke present. In this fashion a gas would be produced containing only nitrogen, carbon dioxide, and sulphur vapour, the last-named derived partly from the loose atom sulphur distilled from the pyrites while descending towards the focus, and partly from the reduction of the SO_2 formed near the focus. On cooling the gas the sulphur vapour would condense and the sulphur could thus be recovered.

The Norwegian experimenters found that it was quite a simple matter to close in the top of the furnace and to use for charging a line of bells similar to those used on an iron blast-furnace. They also found that when using 100 kg. coke per ton* of pyrites in the charge the effect was as expected and a substantial recovery of sulphur resulted.

Ideally, then, the aim of the process is to produce as much sulphur dioxide as possible at the focus and to have this travel up the column in such a way as to effect complete reduction by solid carbon before the gases leave the furnace. It is usually assumed that this reaction takes place for the most part in three stages.

*Throughout this paper the metric ton, 1,000 kg. (2,205 lb.), is used.

The first is represented by equation (3). In the second stage the CO_2 formed is reduced by coke a little higher up the furnace :

$$CO_2+C=2CO \qquad (4)$$

The CO at once reacts with SO_2 thus :

$$SO_2+2CO=\tfrac{1}{2}S_2+2CO_2 \qquad (5)$$

This last reaction goes towards completion at temperatures below 600°C. It must be carefully noted that if equations (4) and (5) be added together and like molecules subtracted from each side of the equation the result is equation (3). Therefore, from the point of view of coke consumption per ton of SO_2 reduced, it matters not at all whether reduction proceeds in one step or in three, provided that all the CO involved is derived from CO_2 produced by equation (3) and not from other sources which will now be discussed.

Unfortunately the reduction of SO_2 by solid carbon—equation (3)—will only take place with reasonable rapidity at a temperature of about 1,200°C. and in the copper blast-furnace the high temperature zone does not extend for a sufficient distance above the focus to give efficient reduction. In other words, there is insufficient contact time at the temperatures prevailing for the coke to burn in SO_2 and much of it passes unburnt down to the focus, where it burns in oxygen to CO_2. The greater part of this CO_2 is reduced to CO just above the focus and then reduces SO_2 in accordance with equation (5). This burning of coke in air at the focus has two very ill effects : First, it means that reduction of SO_2 is done by CO, using 750 kg. of carbon per ton of sulphur instead of only 375 kg., the quantity required for reduction by solid carbon whether directly by equation (3) or in stages by equations (3), (4) and (5) ; secondly, the carbon consumes oxygen, which would otherwise be available for oxidizing FeS and producing SO_2. The practical manifestation of this effect is a large matte 'fall' of poor grade.

The fact has to be faced that, in attempting to carry out the reduction of sulphur dioxide by coke in a water-jacketed furnace within a few feet of the focus, the aim is to superimpose a reduction process on one which is essentially intensely oxidizing. The result is necessarily a compromise, in which the pyritic smelting is not very good and the reduction of sulphur dioxide somewhat inefficient.

A complete carbon and sulphur balance is set out in what follows and a quantitative analysis of the theory developed in an endeavour to assess the amount of SO_2 reduced by C and by CO respectively.

The authors must, however, state at the outset that in developing a quantitative analysis of the theory, they do so with great reserve, because of the enormous difficulties involved in obtaining accurate samples and analyses of the furnace gases ; this subject deserves a paper to itself. Suffice it to say that even to-day it is not absolutely certain that the sampling and analysis at Rio Tinto gives the correct distribution of sulphur in the gases as between the various compounds SO_2, COS, CS_2, etc.

Nevertheless, in spite of this uncertainty and in spite of tho

TABLE I

TYPICAL FURNACE FEED AND PRODUCTS
(in metric tons per furnace day)

	Tons	*Cu* *per cent*	*S* *per cent*	*Fe* *per cent*	SiO_2 *per cent*	*As* *per cent*	*Ash* *per cent*
FEED							
Pyrites.........	188·6	1·70	48·12	41·25	2·60	0·71	
Quartz	51·8			5·33	88·30		
Converter slag	23·0	2·75	1·60	55·53	18·00		
Limestone ...	16·6						
Sulphur	5·0		96·91			2·72	
Coke...........	20·9						
Total	305·9						
PRODUCTS							
Matte	49·6	6·23	25·32	51·45			
Slag	161·6	0·36	2·54	40·78	33·30		
Dust............	1·5						
Crude sulphur	58·5		96·91			2·12	0·20

	Per cent by Vol.	*g. per cu. m.*	*g. per cu. m.*	
GASES			*C*	*S*
CO_2	13·8		73·98	
CO	0·9		4·82	
O_2...............	0·5			
SO_2	3·0			42·7
H_2S	0·41			6·1
CS_2		24·34	3·84	20·5
COS		13·7	2·90	7·7
S		3·0		3·0
Total			85·54	80·0

fact that they make certain broad assumptions the validity of which may possibly be questioned, the authors think it is worth while to set out a quantitative analysis of the reduction theory.

The data required are given in Table I.

The first step is the determination of the volume of gas per ton of pyrites smelted by means of the carbon balance.

	kg. of Carbon per 1,000 kg. pyrites
Charged to the furnace as coke	97·61
Carbon charged to furnace in limestone	9·32
Total carbon charged to furnace	106·93

Carbon in exit gases :

	g. Carbon per cu. m.
CO_2 13·8 per cent	73·98
CO 0·9 ,, ,,	4·82
COS	2·90
CS_2	3·84
Total	85·54

Therefore the volume of gas per 1,000 kg. pyrites is—

$$\frac{106{\cdot}93\times1{,}000}{85{\cdot}54} = 1{,}250 \text{ cu. m.}$$

The total sulphur in the exit gases per 1,000 kg. pyrites will therefore be :

In SO_2	42·7	g. S per cu. m.	×	1,250	53·3 kg.
,, H_2S	6·1	,, ,, ,, ,,	×	,,	7·7 ,,
,, CS	20·5	,, ,, ,, ,,	×	,,	25·6 ,,
,, COS	7·7	,, ,, ,, ,,	×	,,	9·6 ,,
,, S	3·0	,, ,, ,, ,,	×	,,	3·8 ,,
Total	80·0	,, ,, ,, ,,	×	,,	100·0 ,,

The sulphur balance can now be constructed as in Table II on page 7.

The first step in the quantitative analysis of the theory is to determine the quantities of sulphur produced respectively by the reduction of SO_2 and by the volatilization of volatile sulphur. The calculation follows :

	kg. S per 1,000 kg. pyrites
1. *Calculation of volatile sulphur*	
Total sulphur in pyrites	482·2
Less Sulphur combined with Cu, Pb, Zn, etc.	19·0
Sulphur in FeS_2	463·2
Therefore, volatile sulphur (42 per cent of sulphur in FeS_2)	194·5
Add Sulphur in residues	25·4
Total volatile sulphur	219·9

		kg. S per 1,000 kg. pyrites
2. *Calculation of sulphur recovered by reduction of SO_2*		
Total sulphur charged to furnace		509·6
Deduct volatile sulphur, volatilized		219·9
Fixed sulphur entering smelting zone as Fe_7S_8 and non-ferrous metal sulphides		289·7
Deduct Sulphur in matte	66·5 kg.	
Slag	21·5 ,,	
Dust	1·6 ,,	
		89·6
Therefore, sulphur oxidized by blast to SO_2		200·1
Sulphur lost in gases and unaccounted		119·5
Balance, sulphur recovered by reduction of SO_2		80·6

The sulphur lost in gases as S, CS_2, H_2S and COS must also at some stage have existed as elemental sulphur. It does not matter whether this sulphur is assumed to come from volatile sulphur or from SO_2 because if its origin is the volatile sulphur then more than 80·6 kg. sulphur must have been reduced from SO_2. Thus we have :

	kg.	kg.
Sulphur produced by reduction of SO_2		80·6
Sulphur produced by reduction of SO_2 but subsequently reacted to form CS_2, COS, H_2S—		
CS_2	25·6	
COS	9·6	
H_2S	7·7	
	42·9	
Elemental S	3·8	
		46·7
TOTAL S formed by reduction of SO_2		127·3

The carbon entering into the reduction reaction is the total carbon charged to the furnace, 97·61 kg., less the carbon combined to form CS_2, COS and CO, 14·5 kg. Thus the carbon entering the reaction is 83·11 kg.

Of the carbon reaching the focus and burning to CO_2 not all will be reduced to CO. It has been assumed that some 10 per cent will not be reduced and will pass into the gases as CO_2.

With the above data and assumptions it is now possible to calculate what proportion of the total sulphur reduced from SO_2 is reduced by CO and by solid C.

Let x = weight of carbon reducing SO_2 by equation (3).
y = weight of carbon burnt to CO_2 at the focus but subsequently reduced to CO and which then reacts with SO_2 by equation (5).

$$x + y = 83{\cdot}1$$

$$\frac{32x}{12} + \frac{32\,(y - 0{\cdot}1y)}{24} = 127{\cdot}3$$

TABLE II

	Tons	*Per cent S*	*Sulphur tons*	*kg. S per 1,000 kg. pyrites*
CHARGED TO FURNACE				
Pyrites	188·6	48·12	90·7	482·2
Residues	5·0	96·91	4·8	25·4
Converter slag	23·0	1·60	0·4	2·0
Quartz	51·8	—	—	—
Limestone	16·6	—	—	—
TOTAL	285·0	—	95·9	509·6
Coke	20·9	—	—	—
PRODUCED				
Crude sulphur	58·5	96·91	56·6	300·5
Matte	49·6	25·32	12·6	66·5
Slag	161·6	2·54	4·1	21·5
Dust	1·5	18·0	0·3	1·6
Gases	—	—	18·6	100·0
Unaccounted loss	—	—	3·7	19·5
TOTAL	—	—	95·9	509·6

Solving this equation gives $x=18{\cdot}8$ and $y=64{\cdot}3$. Thus the sulphur reduced by CO is 77·3 kg. and the sulphur produced by reaction of SO_2 with solid C is only 50 kg. or less than 40 per cent of the total.

In the Appendix the figures for the three years 1936, 1937, and 1938 have been consolidated and the calculation of the amount of sulphur produced by reduction by C and CO respectively has been made. It shows a widely different result and it must be concluded that no precise quantitative evidence can be produced. All that can be said is that a considerable proportion of the total sulphur recovered by the reduction of SO_2 is produced by CO at a carbon consumption of 750 kg. per ton sulphur and that the proportion produced by the efficient reduction by solid C at 325 kg. carbon per ton sulphur is probably always less than 66 per cent.

As the exit gases contain a substantial amount of unreduced SO_2, it might be thought that increasing the carbon/pyrites ratio—

i.e., adding more coke to the charge—would effect a greater degree of reduction and thus increase the recovery of sulphur. In practice, however, although it is possible that the amount of reduction is increased, there is no improvement in recovery for any increase in carbon beyond about 80 kg. per ton. The total sulphur in the gases remains the same, but with a different distribution between the various compounds ; SO_2 is diminished, but CS_2 and COS are increased.

It would seem in fact that the CS_2 and COS in the exit gases from the furnace are dependent on the amount of coke present, the concentration of the sulphur vapour and the quantity of CO in the gases. For example, the COS is formed by the action of CO on sulphur vapour at temperatures below 800°C., whereas CS_2 is formed nearer the focus by reaction between sulphur and hot carbon. The hydrogen sulphide in the gases depends very much upon the amount of sulphur dioxide in the exit gases and the concentration of water vapour present, a small reduction in the amount of sulphur dioxide present increases the H_2S, the increase being roughly in the proportion of $\frac{1}{\sqrt{\text{partial pressure}}}$ of SO_2 for any given concentration of water vapour.

The effects of varying the carbon/pyrites ratio (i.e. the percentage of coke on the burden) is shown in Table III. From these figures it will be seen that with 61 kg. of carbon per ton the process evidently suffers from a deficiency of reducing agent. The SO_2 is very high and the other sulphur compounds are not markedly low. On the other hand, with 104 kg. carbon there is no decrease in the total sulphur lost although there is much less SO_2, as compared with Gas 2 (Table III).

TABLE III

	1	2	3
Carbon charged per ton pyrite, kg.	104·0	77·3	61·0
Sulphur as SO_2 g./cu. m.	39·2	51·6	70·8
,, ,, CS_2 ,, ...	18·6	5·9	5·8
,, ,, COS	7·0	6·2	4·8
,, ,, H_2S	6·9	7·3	4·6
Total sulphur	71·7	71·0	86·0
CO_2 Vol. per cent	14·3	12·6	11·0
CO ,, ,, ,, ...	0·8	0·5	0·6
O_2 ,, ,, ,, ...	0·5	1·0	0·8

The conclusion is, therefore, that, where conditions make impossible the further treatment of the exit gases, the most economical percentage of coke on pyrite will correspond to about 80–90 kg. carbon per ton of ore. Where, however, gases are free of arsenic and can be subjected to catalysis after they leave the furnace, the coke can be profitably increased because the catalysing of Gas 1 (Table III) will yield a total of 34 g. sulphur per cu. m., whereas the catalysis of Gas 2 would only yield 21 g. per cu. m. In other words, with 1,250 cu. m. gas per ton pyrites, the extra 27 kg. of carbon would produce an additional 16 kg. of sulphur.

The reactions upon which catalysis of the exit gases depends are :

$$SO_2+CS_2 = CO_2 + \tfrac{3}{2}S_2 \quad \text{(6)}$$
$$SO_2+2COS = 2CO_2 + \tfrac{3}{2}S_2 \quad \text{(7)}$$
$$SO_2+2H_2S = 2H_2O + \tfrac{3}{2}S_2 \quad \text{(8)}$$

At Orkla catalysis of the exit gases has always been most successfully practised and has resulted in a much higher overall recovery of sulphur than has so far been possible when treating the arsenical Iberian ores. The precise effect of arsenic is discussed later.

The type of smelting done in the Orkla process is pyritic, as it is the oxidation of a sulphide ore, and nearly 70 per cent of the heat generated is derived from the oxidation of iron sulphide. For the process to work well and smoothly it is essential that a high rate of oxidation should be at all times maintained and to ensure this a flux containing a high percentage of free silica is essential.

From what has been written it can be inferred that the conventional type of water-jacketed copper blast-furnace may not be the most suitable apparatus in which to carry out this process, in which a reaction zone has to be maintained at over 1,200°C. if the sulphur dioxide is to be reduced in the contact time available. This unsuitability has been recognized for a long time, but so far the problem of reconciling copper smelting with high sulphur recovery in one apparatus has not, in the opinion of the authors, been fully solved.

Notwithstanding all its admitted shortcomings, the Orkla process uses much less coke per ton of sulphur produced than any of the other reduction processes. The reason, of course, is that about 65 per cent of the sulphur recovered is the loose atom sulphur and only 35 per cent is derived from the reduction of SO_2, whereas all other processes first burn the raw material to SO_2. Thus at Rio Tinto slightly over 3 kg. crude sulphur per kg. of carbon is obtained, whereas the theoretical equivalent when reducing SO_2 by C is 2·66 ; in practice the figure is almost certainly less than 2·00 kg. crude sulphur per kg. of carbon.

The comparative inefficiency of SO_2 reduction in the Orkla furnace is therefore much less important from the economic standpoint than would be the case if all the sulphides in the feed to the furnace had first to be oxidized to SO_2. This point has an important bearing on the treatment of sulphides other than FeS_2.

Sulphur Purification

The crude sulphur coming from the condensers and mist Cottrells contains 1·5–2 per cent of arsenic and 0·2–0·3 per cent of ash.

Arsenic is removed by circulating milk of lime through the molten sulphur in autoclaves. When the original experiments were carried out to determine the most efficient method of refining, various soda compounds were tried, with the idea of forming sodium thio-arsenate. It was found that when caustic soda was used mixed with lime a much quicker removal of arsenic resulted, but NaOH always gave the sulphur a peculiar greasy appearance and a bad colour. Finally, quick lime was decided upon as the best, cheapest, and most readily-available reagent. This removes arsenic very efficiently, provided the lime is of good quality and correctly used. The lime suspension is pumped through the sulphur and re-acts in the course of its passage with the arsenic present to form complex calcium thio-arsenates, the lime containing them being eventually run to waste. The arsenic in the crude sulphur is present as arsenic pentasulphide, As_2S_5, and the reactions which take place when the crude sulphur is scrubbed with a lime suspension are somewhat obscure, but it seems probable that the reaction between the quick lime and the arsenic sulphide in the crude sulphur proceeds by the formation of calcium thio-arsenate. As H_2S is not produced during refining the authors consider the reaction to be—

$$4\,Ca(OH)_2 + As_2S_5 + 2S_2 = Ca_3As_2S_8 + CaSO_4 + 4H_2O \quad \text{.........} \quad (9)$$

The theoretical requirement is thus 1·47 tons CaO per ton arsenic. It will be seen from Table IV that the actual consumption is between 1·8 and 2·2 tons free CaO per ton arsenic. There is always a portion of the CaO which fails to react because it is inert through being overburnt, also calcium polysulphides are formed. The treated sulphur contains some 3 to 10 parts per million of arsenic and is passed through ordinary steam-heated filter presses, these filters removing the ash so that the final product is sulphur of high purity.

SECTION 2—THE PLANT

The process flow-sheet is comparatively simple (Fig. 1), and various sections of the plant will be described in some detail before proceeding to the discussion of metallurgical results.

Before describing the individual units of the plant, a very brief summary of the sequence of operations will be given.

Sulphur-bearing gases from blast-furnaces are cleaned by passing through Cottrell electrostatic precipitator units ; they then pass first through condensers, where the larger part of the sulphur is recovered, and then through a second set of Cottrells, where a further recovery of sulphur is made. The crude sulphur contains arsenic, which is removed by a washing process employing quick lime.

It should be clearly understood that the Rio Tinto plant is very much an improvization and that it certainly cannot be described as an ideal Orkla process layout. In this respect the Norwegian plants and that of Mina de S. Domingos, Portugal, are both much superior and it is hoped that a full description will be given during the discussion of this paper. The operators referred to had the advantage of being unencumbered with an existing copper smelter and so were able to build from the foundations with the Orkla smelting system in view. At Rio Tinto, on the other hand, there was already a smelter many years old and in many respects already antiquated, but from which the flow of copper had to be maintained. It was, therefore, necessary to create an Orkla smelting plant by a step-by-step modification of existing blast-furnaces, together with the addition of dust precipitators, sulphur condensers, etc., not in

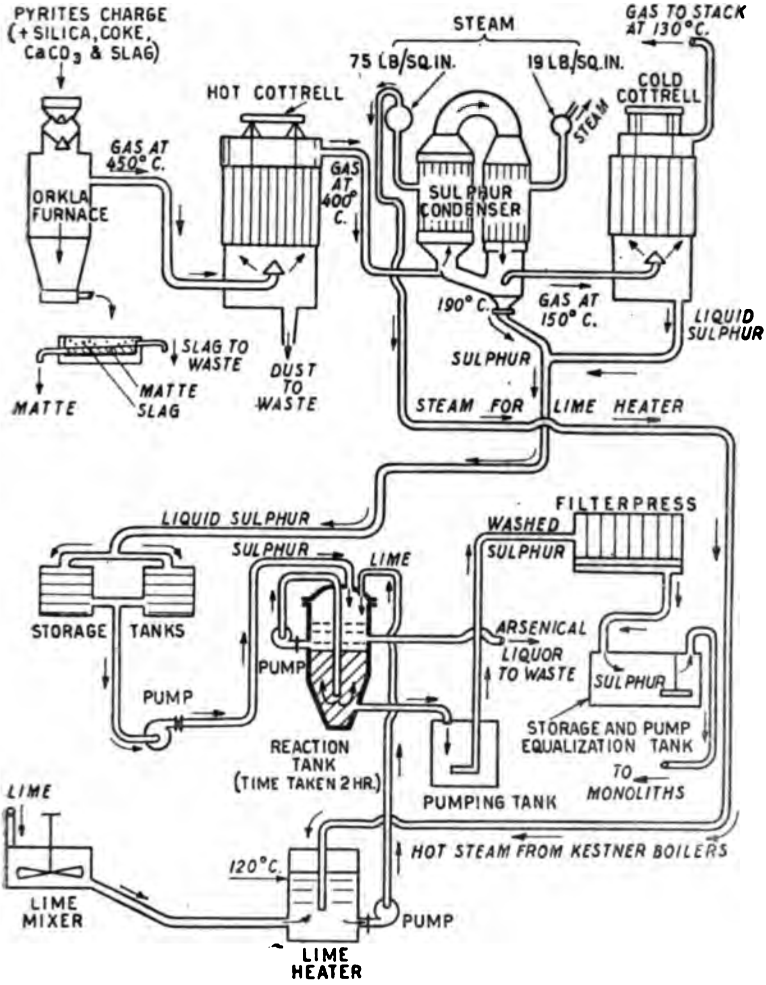

FIG. 1.—Flow-sheet—Orkla furnace and sulphur washing plant.

TABLE IV

ORKLA PROCESS OPERATING RESULTS

SMELTING	Years 1940–1946 *inclusive* 3,721 *furnace days*					Years 1934–1939 *inclusive* 2,928 *furnace days*				
	Metric tons	*Sulphur*		*Copper*		*Metric tons*	*Sulphur*		*Copper*	
		Per cent S	*Tons*	*Per cent Cu*	*Tons*		*Per cent S*	*Tons*	*Per cent Cu*	*Tons*
CHARGE										
Pyrites	634,345	46·80	297,112	1·90	12,050	535,832	46·80	251,255	3·57	19,139
Concentrates	5,285	35·60	1,886	12·80	653	—	—	—	—	—
Precipitate	—	—	—	—	—	69	—	—	69·50	48
TOTAL PRIMARIES	639,630	—	298,998	—	12,703	535,901	—	251,255	—	19,187
Washing plant residues	18,707	85·40	15,927	—	—	12,885	85·90	11,117	—	—
Clean up	2,566	85·00	—	—	—	779	—	—	—	—
Converter slag	58,413	1·51	879	5·21	3,047	59,764	1·75	1,040	4·76	2,848
Matte	9,627	23·6	2,270	9·08	874	7,692	24·60	1,897	16·25	1,246
TOTAL SECONDARIES	89,313	—	19,076	—	3,921	81,120	—	14,054	—	4,094
Silica flux	177,574	—	—	—	—	137,609	—	—	—	—
Limestone	48,276	—	—	—	—	47,332	—	—	—	—
BURDEN	954,793	—	318,074	—	16,624	801,962	—	265,309	—	23,281
Coke	57,638	—	—	—	—	55,816	—	—	—	—
Per cent on ore and concentrates	9·04%	—	—	—	—	10·4%	—	—	—	—
Per cent on burden	6·05%	—	—	—	—	6·9%	—	—	—	—

PRODUCTS										
Crude sulphur	188,433	97·2	183,828	—	—	167,319	97·3	162,740	—	—
Matte	165,248	23·7	39,113	8·55	14,150	135,694	25·1	34,149	14·80	20,178
Slag	535,920	2·92	15,661	0·327	1,755	444,536	2·66	11,823	0·385	1,703
Cottrell dust	5,189	21·21	1,110	1·96	96	3,373	19·40	655	1·84	61
Gases	—	—	69,560	—	—	—	—	50,240	—	—
Unaccounted	—	—	8,802	—	623	—	—	5,702	—	1,339
TOTAL	—	—	318,074	—	16,624	—	—	265,309	—	23,281
PURIFICATION				*Per cent As*	*As tons*				*Per cent As*	*As tons*
CHARGE										
Crude sulphur	188,145	97·2	183,646	2·02	3,696	166,685	97·3	162,299	2·21	3,664
Lime	8,127	—	—	—	—	6,644	—	—	—	—
PRODUCTS										
Sulphur	153,088	99·9	153,087	*Parts/10*^{6} 24	—	133,360	99·9	133,360	*Parts/10*^{6} 6	—
Waste liquor, m^3	80,265	*kg./m*^{3} 119·0	9,517	*kg./m*^{3} 49·5	3,949	90,253	*kg./m*^{3} 102·0	9,184	*kg./m*^{3} 41·5	3,740
Residues	19,150	85·10	16,356	—	—	12,881	87·1	11,222	—	—
Process loss	—	—	4,686	—	—	—	—	8,533	—	—

Of the 318,074 and 265,309 tons of sulphur entering the furnaces in the two periods, 15,927 and 11,117 tons of sulphur are from residues—that is, residues produced when refining sulphur which do not represent fresh sulphur entering the process.

The new sulphur is only obtained from the pyrites and concentrates and perhaps from the converter slag and matte.

If we assume the sulphur is coming from the above-mentioned four materials only, the sulphur recovery is:

1940–1946 $\dfrac{153{,}087}{(318{,}074-15{,}927)} = 50{\cdot}66$ per cent. 1934–1939 $\dfrac{133{,}360}{(265{,}309-11{,}117)} = 52{\cdot}46$ per cent.

If, however, we assume the sulphur recovered only comes from the pyrites and concentrates, then the sulphur recovery is:

1940–1946 $\dfrac{153{,}087}{(318{,}074-19{,}076)} = 51{\cdot}2$ per cent. 1934–1939 $\dfrac{133{,}360}{(265{,}309-14{,}054)} = 53{\cdot}07$ per cent.

ideal sites and properly connected to the furnaces but on sites dictated by the availability of space in the existing plant.

There is no doubt that the ideal layout consists of smelting furnaces each coupled to its own dust precipitator and sulphur condenser, all precipitators and condensers being placed as close as possible to the furnaces. It will be seen from the description which follows how very far it has been necessary in the present case to depart from this ideal.

Blast-Furnaces

Three out of the original six open-topped furnaces have been modified to allow sulphur recovery ; the first being No. 6 furnace, the most easterly one and, therefore, nearest to the hot Cottrells. The conversion of Nos. 5 and 4 furnaces followed later. The dates upon which these three furnaces were first blown in after being modified were : No. 6, 1 Aug. 1930 ; No. 5, 5 Sept. 1932 ; No. 4, 10 Feb. 1942.

The normal copper-smelting furnace is preserved in its usual form up to the level of the charging floor, but instead of the top being left open and charging effected through side doors on the level of the charging floors, the top was heightened, completely closed in, and equipped with four sets of double-valved charging bells, very similar to the gas-tight bells used on the normal iron blast-furnace ; by this means charge can be introduced into the furnace without allowing any access of atmospheric air. Fig. 2 (Plate I) shows the construction.

It will be seen from Table V that the dimensions of the three furnaces are not identical. All of them have the same height and use standard 12-ft. side jackets, but the rake of the jackets is different, being much greater in No. 6 ; this large flare was originally given with the idea of reducing the gas speeds in the furnace and thereby effecting a better reduction of SO_2 ; the authors think that in many ways No. 6 has been consistently the most satisfactory furnace, perhaps because, having the shortest flue connecting it with the hot Cottrells, less back pressure is developed and a faster smelting rate results. However, the wide flare has the disadvantage of needing specially large, non-standard end and breast jackets. The jackets are all made entirely by electric welding, the fire sheets being of a $\frac{3}{8}$-in. mild steel, and the back sheets of $\frac{5}{16}$-in. The 12-ft. jackets are satisfactory on the open-topped furnaces, but they are not sufficiently stiff to resist the heavy thrust imposed by an 18-ft. charge column and they tend to buckle, even though they are reinforced by three heavy buck stays bolted across each side of the furnace. Leading particulars are given in Table V.

The upper part of all the furnaces, above the jackets, is supported on a framework of box girders and consists of a steel box welded at the joints. This is solidly lined with 18 in. of good-quality firebrick ; the roof of the furnace between the charging bells is formed by an arch made of carefully cut and fitted firebricks. The furnace

TABLE V

	No. 6 Furnace	*Nos. 4 and 5 Furnaces*
Length	18 ft.	18 ft.
Number of charging bells	4	4
Height from closed bells to floor	21 ft. 7 in.	22 ft.
Height from top of jackets to roof	10 ft. 2 in.	9 ft. 11 in.
Width at top of jackets	9 ft. 4 in.	5 ft. 5 in.
„ „ tuyères	4 ft. 5 in.	3 ft. 5 in.
Hearth area	82·6 sq. ft.	61 sq. ft.
Number of tuyères	36	36
Diameter of tuyères	4 in.	4 in.
Tuyère area	3·14 sq. ft.	3·14 sq. ft.
Ratio—Hearth to tuyère area	26 to 1	19 to 1
Height c/l of tuyères to floor	2 ft. 6 in.	2 ft. 6 in.
Number of gas ports	5	18
Height of gas ports	1 ft. 6 in.	1 ft. 4 in.
Width of gas ports	3 ft.	7¼ in.
Total area of gas ports	22·5 sq. ft.	14·4 sq. ft.
Number of side jackets	12	12
Dimensions of side jackets	12 ft. 3¾ in. by 3 ft.	12 ft. 3¾ in. by 3 ft.
Flare of jackets from centre line of tuyères	1 ft. in 2 ft.	1 ft. in 4 ft. 9 in.

bottom is put in with one lower row of firebrick and an upper one of magnesite brick; the use of firebrick alone for the furnace bottoms has been found unsatisfactory, because of the scouring effect of the slag.

The practice of lining the inner side of the water jackets completely with good-grade firebricks for the full height of the jacket, has been made standard in the Norwegian plants, upon the supposition that this will maintain a higher temperature next to the jacket and therefore lead to better reduction of SO_2, and also that the hot surface of the brickwork may have a catalysing effect. This lining has been tried once or twice at Rio Tinto, but its use has been discontinued for many years past, as it was found that the lining is soon fluxed away, leaving the jackets bare again.

The gases leave the top of the furnaces through ports and are conducted to the hot Cottrells by rectangular insulated flues, which lead into a junction box, on which the valves for regulating the flow of gas into the various filter chambers are situated, each pair of chambers having its own isolating valve.

The flues connecting Nos. 4 and 5 furnaces have each an internal cross-sectional area of 14·30 sq. ft.; the flues are well insulated with 1¼-in. thick slabs of an asbestos material, backed up by a firebrick lining; the calculated gas speed through these flues at 400°C. is 17·5 ft./sec. These long flues are an undesirable feature of the Rio Tinto plant, but because of the existing layout of the original furnace plant they are unavoidable. They form a great obstacle on the furnace feed floor, hindering ready access to the furnaces; they also slowly fill up with a layered deposit of dust and condensed sulphur, which thus increases the back pressure in the furnace and tends to slow down the rate of smelting.

The valves on the junction box, which are operated by a pinion and rack arrangement, are preferably made of chrome iron, with chrome steel spindles, and when thus equipped they give no trouble.

The furnaces are charged by 5 h.p. Morris pulley block hoists, which travel on an ' I ' section beam running along the centre line of the bells, by means of which small ' buggies ' of ore can be picked up and tipped into the charging bells. These buggies are adapted to sit on the undercarriage of the normal charge wagon and they can, therefore, be run under the bins and filled in exactly the same manner as the side dump cars used with the open-top furnaces. The settlers are the same size as those used with these ordinary furnaces, although the total burden smelted daily in the sulphur furnaces is much smaller—about 280 tons compared with 400–450 tons ; their diameter is 14 ft. unlined and 11 ft. 3 in. lined, and the depth is 4 ft. 8 in. unlined and 3 ft. 10 in. lined. The lining consists entirely of magnesite bricks on the bottom, and two rows of firebricks, backed up by one of magnesite bricks, round the circumference. (The term ' magnesite bricks ' includes chrome-magnesite bricks, which have, on the whole, been found to give better service than straight magnesite ; straight chrome bricks have also been used in the settlers, but were found to be no improvement upon the magnesite bricks.)

The cubic capacity of one of these settlers when newly lined is about 340 cu. ft., which is equivalent to a holding capacity of 40 tons of matte, but, unless very careful furnace control is exercised, the capacity will be drastically reduced by magnetite accretions until it may be no more than a fifth of what it was originally ; this point is dealt with more fully later in the paper. At Orkla and S. Domingos portable settlers of much smaller capacity are used and the substitution of such settlers was at one time under consideration at Rio Tinto, but the idea has been given up now that it has been learnt how to keep the big fixed settlers ' open '.

Hot Cottrells

These are of the vertical chamber type and they originally consisted of two blocks in parallel, each block containing two filter chambers. The material chosen was brick, but new Cottrells have now been built in concrete with a firebrick lining, which should prove a more satisfactory type of construction.

Fig. 3 (Plate II) shows details of one unit of the Cottrell precipitator. When the first furnace, No. 6, was converted for sulphur recovery, four hot Cottrells were provided to clean the gases, and they were guaranteed to give better than 90 per cent clearance on a gas which contains 5–6 g. of dust per cu. m. at 400°C.

The total gas capacity was 9·2 cu. m. (327 cu. ft.) per sec. at 400°C.; as the cross-sectional area of each filter chamber is 6·5 sq. m., the calculated gas velocity is 0·35 m. per sec. (1·15 ft. per sec.).

The brickwork construction of the original Cottrells proved difficult to maintain. Distortion of the structure threw the electrodes out of alignment, causing short circuits and necessitating frequent shut downs. Accordingly in June, 1934, when a second sulphur furnace was blown in, two more chambers were added in order to insure that two chambers per furnace would always be available.

Provided that the electrical installation is functioning properly two chambers will give perfectly satisfactory gas cleaning. The gas produced by each furnace is 400 cu. m. per min. measured at 400°C., and as the cross-sectional area of two chambers is 13·0 sq. m. the gas velocity is 0·5 m. per sec. (say 1·6 ft. per sec.). Table VIII shows that 95 per cent dust clearance is obtained.

The electrode assembly consists essentially of a series of flats, each 70 mm. wide by 1·5 mm. thick and 3·175 m. long, connected by cross members so that in effect they form a plate or ' frame '. There are 11 such plates or ' frames ' to each chamber and they are spaced 251 mm. apart. These plates are earthed. Between the earth plate assemblies are the emission strips which are flats 18 mm. wide by 3 mm. thick and 4·21 m. long. These flats are hung with their 18 mm. dimension at right angles to the earth plate, and the distance from the edge of the emission strip to the earth plate is 116 mm. The emission strips are spaced 132 mm. centre to centre. They are supported by a cross member forming a main suspension and are stiffened by a similar cross member at the bottom of the assembly. Fig. 4 shows in perspective the electrode assembly, and leading particulars of the plant are given in Table VI.

All the electrodes in the Cottrells were originally made of mild steel, the emission electrodes being wires, and the earth plate frames were made up of ordinary corrugated-iron sheets; the results obtained were far from satisfactory, as mild steel is rapidly attacked at 400°C. by sulphur vapour. Chrome steel was accordingly substituted and has given very satisfactory results. A typical analysis of the steel used at Rio Tinto is:

	Per cent
Cr	18·53
S	0·015
C	0·11
P	0·016
Si	0·38
Mn	0·22

It has been found by experience that any steel with a lower chromium content than 18 per cent is quickly attacked by the gases. High chromium steels tend to become brittle after prolonged heating at temperatures in the range of 400–500°C., particularly above 475°C. This tendency is much reduced if the

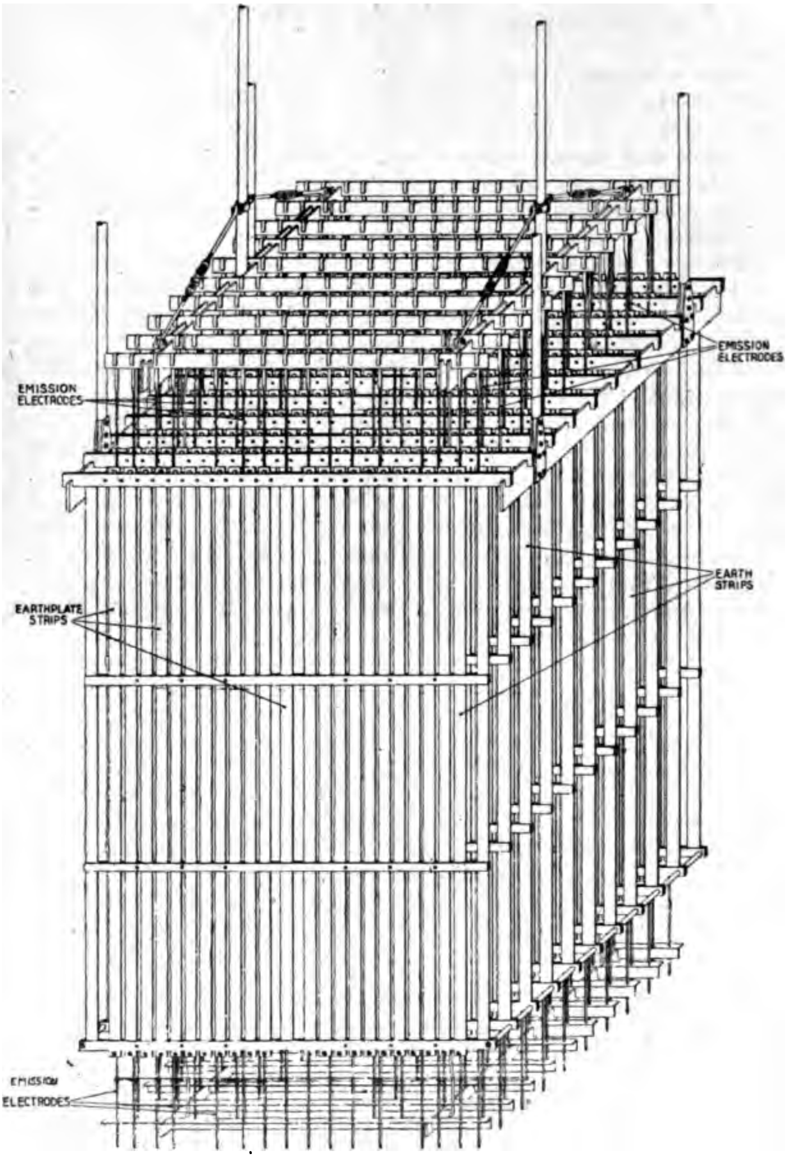

FIG. 4.—' Hot ' Cottrell—electrode assembly (perspective view).

TABLE VI

HOT COTTRELL—LEADING PARTICULARS

Total overall height — cover plates to hoppers	41 ft. 0 in. (12·50 m.)
Height from c/l of gas inlets to cover plates ...	30 ft. 0 in. (9·10 m.)
Distance from c/l of gas inlets to bottom of hoppers....................	9 ft. 6 in. (2·904 m.)
Distance from top of emission electrodes to cover plates..............	7 ft. 0 in. (2·14 m.)
Distance from top of earth frames to cover plates...	8 ft. 6 in. (2·58 m.)
Distance from c/l of gas inlets to bottom of electrodes.................	10 ft. 0 in. (3·05 m.)
Distance from top of gas distributor cone to bottom of electrodes ...	5 ft. 6 in. (1·67 m.)
Total height occupied by electrodes.................	14 ft. 0 in. (4·25 m.)
Internal dimensions of chambers.................	8 ft. 3 in. by 8 ft. 6 in. (2·51 m. by 2·58 m.)
Areas of each filter.........	70 sq. ft. (6·50 sq. m.)
Size of earth plates.........	2·5 m. by 3 m. high
Dimensions of earth plate strips.......................	3175 mm. by 70 mm. by 1·5 mm. thick
Number of negative emission strips per chamber	160
Dimensions of emission strips.......................	4260 mm. by 18 mm. by 3 mm. thick
Spacing between earth plates	251 mm.
Spacing between emission strips—centre to centre	132 mm.
Spacing between earth plate strips—centre to centre	92 mm.
Spacing between edge of emission strips and earth plates...............	116 mm.
Power used per 100,000 cu. ft. of gas cleaned (measured at 400°C.) ...	1·724 kWh
Number of earth frames in each chamber	11

chromium content is below about 15 per cent, but increases rapidly above about 17 per cent Cr. This embrittlement can lead to excessive breakage of costly steel and is avoided to a large extent by using steel of 18 per cent Cr rather than 20–25 per cent Cr.

The rapping gear for dislodging dust is automatic; the frame from which the emission (negative) electrodes are hung (one frame in each chamber or filter) is suspended from two shafts, each of which receives a blow every 50 sec.—i.e., the frame receives a blow every 25 sec. delivered from alternate sides. The hammers, which have wooden shafts shod with a steel ring, are chain-driven through small motors. The earth plates (positive) are rapped continuously in the same manner. There is also a hand rapping apparatus for the bottom of the emission electrodes, which is used as and when required, but naturally the current must be taken off the filters before this can be done, as the hand rapping gear is not insulated.

The present electrical equipment of the Cottrell chambers is :

3 30-kVA transformers, 220/68,000 V, 50 cycles.
3 Rotary disc rectifiers.

Armoured cables are used for the high tension supply to the chambers.

Each rectifier can supply a maximum of 60 mA on the hot Cottrells and 90 mA on the cold. The load on the transformers is between one third and one half of their full rating.

Each pair of chambers is connected in parallel electrically, and normally consumes 35 mA of rectified current on the hot Cottrells and 35–55 mA according to the temperature of the gas on the cold Cottrells.

The total consumption of the whole Cottrell plant is approximately 34 kW per hour. The polarity of the current used, as has already been indicated, is negative on the emission electrodes, or strips, and positive on the precipitator electrodes, or earth plates.

Sulphur Condensers

The three sulphur condensers are each constructed as two fire-tube boilers operating in series, the gases passing through tubes surrounded by water. The first, or high pressure boiler, has a drum with an internal diameter of 6 ft. 1 in. and contains 831 tubes, each of 1½-in. internal diameter and 12 ft. 4 in. long, with a surface of 3,920 sq. ft. The second, or low pressure, contains 250 tubes, each of 2-in. internal diameter, and 6 ft. 4 in. long, with a surface of 782 sq. ft. The total heating surface of one cooler is therefore 4,702 sq. ft., or 437 sq. m. Fig. 5 (Plate III) shows the general arrangement of one unit.

As each furnace needs its own cooler there is, if it is assumed that a furnace smelts 180 tons of mineral per 24 hr., 2·4 sq. m. of cooling surface per ton of mineral smelted. Later experience has indicated that it would be advantageous to have a surface of about 3 sq. m. per ton of mineral smelted.

The boilers were designed for the conditions set out in Table VII.

Table VII

	H.P. Side	*L.P. Side*
Steam pressure	75 lbs. p.s.i.	19 lbs. p.s.i.
Gas volume N.T.P., c.f.m.	7,900	7,900
Spec. heat	0·232	—
Sulphur content	235 g./cu. m.	—
Temperature of gas :		
Entering	380°C.	180°C.
Leaving	180°C.	150°C.
Evaporation : lb./hr.	2,500	537

The Norwegian and Portuguese plants both use coolers which are designed on the opposite principle—that is to say, the water passes through the tubes, which are surrounded by the gases. In general there is little doubt that this water-tube boiler is superior to the Rio Tinto type, and gives better results; a greater proportion of the sulphur is condensed, not because the gases are cooled better than by the Rio Tinto boilers, but because the tubes act as baffles and help to precipitate the sulphur mist; also less frequent cleaning is required.

It should here be remarked that at the Norwegian plant very efficient use is made of the steam raised. The steam produced is superheated in an independent oil-fired superheater and then used in a steam turbine. The power generated not only supplies the blowers, motors, pumps, and all other requirements of the whole smelting plant, but also puts a small amount of surplus power back on to the mine grid.

At Rio Tinto the use of steam for raising power is prevented largely by the fact that it is required in the sulphur-washing plant for heating purposes.

Cold (Mist) Cottrells

After leaving the coolers, the gases pass to the mist Cottrell, which is similar in all respects to the precipitators used for dealing with the hot gases, except for the fact that it is smaller, consisting of four chambers only; the electrode assemblies are made of plain mild steel.

The function of the cold Cottrells is not, of course, to clean the gases of dust, but to precipitate the mist of liquid sulphur which has been carried past the coolers; the sulphur dripping from the electrodes is collected at the bottom of the chamber and flows against the incoming gas stream and joins the sulphur stream from the bottom of the cooler.

Typical operating results for the hot and cold Cottrells are shown in Table VIII on the next page.

Sulphur Purification Plant

The equipment of the purification plant, or washery, is simple, as befits what is essentially a simple process: it consists of large open storage tanks for sulphur, autoclaves, tanks for preheating the lime suspension, agitator tanks in which further arsenic removal can be carried out if necessary, filter presses, together with pumps for transferring sulphur from one tank to another, and a storage tank for filtered sulphur.

All the tanks and autoclaves are fitted with internal steam coils and are well lagged externally; the filter presses, and all pipe lines through which sulphur flows, are steam-jacketed, the steam being supplied by the sulphur coolers.

There are three autoclaves and each has its own tank for heating the lime suspension. There are two filter presses which serve the three units.

Table VIII

Cottrell and Condenser Operating Data

Hot Cottrell

Furnace days	519
Pyrites smelted, *tons*	90,487
Burden smelted, *tons*	134,133
Crude sulphur produced, *tons*	28,023
Ash in crude sulphur, *per cent*	0·24
Equivalent to dust, *tons*	67·0
Dust collected, *tons*	1,414·0
Total dust burden of gas, *tons*	1,481·0
Dust per ton pyrites, *kg.*	16·36
Efficiency of clearance, *per cent*	95·5
Dust burden of gas, *g. per cu. m. at 400°C.*	5·25
,, ,, ,, ,, ,, ,, ,, ,, ,, *N.T.P.*	13·09
Power used, *kWh.*	173,000

Condenser

Gas cooled per min., *cu. ft. N.T.P.*	5,900
Temperatures :	
Entrance to H.P. side, *°C.*	395
Exit of H.P. side and entrance to L.P. side, *°C.*	180
,, ,, L.P. side, *°C.*	149
Steam pressures :	
H.P. side, *p.s.i.*	45
L.P. ,, ,,	12
Evaporation :	
H.P. side, *lb. per hr.*	2,700

Cold Cottrell

Temperature of gas at entrance, *°C.*	155
,, ,, ,, ,, exit, *°C.*	125
Power used, *kWh.* (519 furnace days)	100,100

Storage Monoliths

These are simply very large wooden boxes, each approximately 24 m. by 10 m. by 8 m. ; the ends are permanent, but the sides are raised as required by bolting additional planks upon the uprights which are evenly spaced down the long sides. When full, each monolith contains about 3,100 tons of sulphur, which is dug out and loaded by hand into railway wagons.

SECTION 3—THE PRACTICE OF THE PROCESS

Smelting

The sulphur furnaces will each smelt on the average 180 tons of pyrites per day ; they normally produce some 45 to 50 tons of matte and about 50 tons of crude sulphur per furnace day. In Table I are given details of a typical furnace charge together with the analysis of ores and furnace products ; the ratio of concentration, mineral to matte, during the past five years has been :

Ratio of Concentration	
1944	3·8 to 1
1945	3·2 „ 1
1946	3·5 „ 1
1947	3·55 „ 1
1948	3·5 „ 1

The grade of ore now being smelted at Rio Tinto lies between 1·5 and 2·0 per cent Cu and the matte produced by the furnaces contains about 6–8 per cent Cu. Matte of such low grade is unsuitable for direct converting and up to the present has been concentrated by resmelting in a normal blast-furnace. In the future, however, it is contemplated that this re-smelting may be done in a closed-top furnace and a portion of the sulphur contained in the low-grade matte recovered. This re-smelting of the low-grade matte in a closed-top furnace with the recovery of the sulphur is current practice at Thamshaven.

In regard to the actual smelting practice from the point of view of copper, there is little to say, except that troubles will arise from the freezing-up of the settlers unless the nature of the slag produced is carefully controlled. It is difficult to maintain a satisfactory slag at Rio Tinto, because of the lack of a suitable siliceous flux with a high free silica content, e.g., 95 per cent SiO_2 ; the fact that the siliceous flux contains considerable quantities of combined silicates has a detrimental effect upon the sulphur reduction process, because it lowers the focus temperature. If the silicate degree is allowed to fall too low the slag will contain a large quantity of magnetite, which settles out and builds up on the bottom of the settler : this process will be appreciably hastened whenever the blast is taken off the furnace and the molten contents of the settler are allowed to remain quiescent for any length of time. This fact has led to the use of portable settlers at both Orkla and S. Domingos, but by careful furnace control and the emptying of the settlers, whenever the blast is off the furnace for more than a very short time, settler difficulties at Rio Tinto have been quite overcome, as can be seen from the fact that the average length of a furnace campaign is six to eight months. At Rio Tinto therefore, conventional settlers have been retained.

It may be of interest to note that experience, both at Orkla and Rio Tinto, has shown that tuyère punching is not desirable and tends to upset the steady working of the furnace : as little punching as possible is, therefore, done and the tuyères rarely show any ‘ fire ’.

With regard to smelting practice, from the point of view of sulphur recovery, the sizing and condition of the whole charge, which includes coke, has an important bearing upon the results obtained. It is, of course, obvious that the more evenly the coke is distributed throughout the charge, the better will be the reduction of SO_2, and that the less fines there are in the charge generally the less work will be thrown upon the hot Cottrells in removing dust from the gases. The ore is prepared by screening on 1-in. square mesh and the upper limit is an 8-in. ring. It has not yet been possible to explore the effect of closer sizing and possibly bedding of the furnace charges ; closer sizing may be more important than has hitherto been believed, and it would be interesting to see the effect of smelting a carefully-sized charge in which everything would have passed through a 3-in. ring and be held on a 1-in. ring, coke and fluxes included. A charge of this kind might possibly lead to better reduction of SO_2 by preventing channelling of the gas streams through the furnace column. Such a reduction in the size of the charge would undoubtedly require an increase in the blowing pressure and would call for more powerful blowers than those used at present. Furthermore, the cost of preparing a graded charge would be considerable.

The coke used for the process must on no account contain more than 1½ per cent of volatile matter if bad discolouration of the resulting sulphur is to be avoided. Experience has proved the desirability of having the coke carefully sized, because a large piece of coke of, say, 8 in. presents a much smaller surface for SO_2 reduction (or CO_2 reduction) than would the equivalent weight of coke in pieces of 1½-in. diameter. The specification aimed at for coke for the furnaces is :

Volatile matter	not to exceed 1 per cent
Ash	,, ,, ,, 8 ,, ,,
Moisture	,, ,, ,, 3 ,, ,,
Size	40–60 mm.

Actually great difficulty has been experienced in obtaining Spanish coke which even approaches this ideal ; most of the coke, received from the North of Spain, contains not less than 15 per cent ash, and on occasions 26 per cent, while the moisture is rarely less than 10 per cent and sometimes reaches 15 per cent.

The Furnace Gases

Although a direct analysis of the gases as they issue from the furnace top has never been made, it is known that substantially they consist of nitrogen, sulphur vapour, and carbon dioxide, together with some unreduced SO_2. In addition, the gases contain varying amounts of carbon disulphide, carbon oxy-sulphide, and hydrogen sulphide.

After the gases have been cleaned in the hot Cottrells, and have been passed through the sulphur condensers and mist Cottrells, they are discharged to atmosphere. The average analysis of our gases at present as they leave the cold Cottrells is :

Sulphur as SO_2	58 g. per cu. m.	(4·1 per cent SO_2 by volume)
,, ,, H_2S	6 ,, ,, ,, ,,	(0·4 ,, ,, H_2S ,, ,,)
,, ,, COS	6 ,, ,, ,, ,,	
,, ,, CS_2	6 ,, ,, ,, ,,	
,, ,, S vapour	4 ,, ,, ,, ,,	
Total	80 ,, ,, ,, ,,	
CO_2	10·0 per cent	

It was found that the composition of the end gases fluctuates violently with changing furnace conditions, but the figures given represent the average of several hundred samples and may be taken as typical of the gases to be obtained with 9–10 per cent of coke on the ore.

Catalysis has not so far been used at Rio Tinto, mainly owing to the fact that the Rio Tinto ore contains substantial quantities of arsenic. The presence of arsenic introduces complications throughout the process which are not met with at the Norwegian plant. It is, in fact, the presence of arsenic in the Spanish ores which explains the difference in practice and also the lower recovery of sulphur obtained at Rio Tinto compared with the plant in Norway, where pyrite with a very low arsenic content is smelted. The reason why arsenic has such a marked effect on the whole operation resides in its effects on the viscosity of sulphur and the dew point of the gases. At Rio Tinto, with gases carrying 230–250 g. of sulphur and about 5 g. of arsenic per cu. m., a sulphur containing a high percentage of arsenic begins to condense at temperatures as high as 350°C., whereas with an arsenic-free gas the saturation point would be much below that temperature. Furthermore, the liquid sulphur condensed will contain 14 per cent or more of arsenic, which imparts to it a degree of viscosity which practically inhibits flow. Once a deposit of high-arsenic sulphur has formed a subsequent rise of temperature will not re-evaporate it ; some sulphur is re-evaporated, but the whole of the arsenic is concentrated in the remainder, increasing the viscosity and further raising the boiling point.

Any attempt to pass the gases over a catalyst mass between the hot Cottrells and the coolers is doomed to failure, as it would be quite impossible consistently to maintain the gases at a temperature above 350°C. Variations in conditions inside the furnace frequently cause a lowering of the temperature of the gases at the exit of the hot Cottrells to something considerably below 350°C. The effect of this would be the rapid deposition of arsenical sulphur on the mass, which would thus be completely blanketed in a very short time.

The problem cannot be solved by running the furnaces with a low column and consequently a hot top, as there is a limit to the temperature of the gases entering the Cottrells. This is imposed by the ability of the structure and the steel electrodes to withstand high temperatures, and it is preferred to keep the gases at something less than 430°C.

This inability to catalyse the exit gases of the process is a very severe handicap. The gases contain about 80 g. S per cu. m., of which it is probable that 50 per cent could be recovered by catalysis. This would increase the overall recovery from the present figure of about 55 per cent to 65 per cent or possibly more.

Recovery

The authors have always been keenly aware that the loss of sulphur as SO_2 is most unsatisfactorily high at Rio Tinto and it is believed that the reason for incomplete reduction of SO_2 is lack of temperature and contact time—that is to say, that the 1,100°–1,200°C. zone, in which the reduction of SO_2 proceeds most rapidly, is too short.

In regard to temperature it is believed that it would be possible to intensify the whole process and increase the rate of smelting if the blowers were of larger capacity. At present experiments are being conducted to ascertain the precise effect of harder blowing.

From time to time the effect of a large increase in the coke consumption has been investigated and a test was recently run, using 13 per cent coke on mineral. As has already been explained this does lead to a substantial lowering of the SO_2 content of the gases, but the sulphur present as CS_2, COS, and H_2S increases and the net result is the same total quantity of sulphur lost in exit gas.

Increasing the coke tends to lower the grade of matte by inhibiting the oxidation of FeS and thus increasing the sulphur lost in matte. So far, therefore, the most economical figure seems to be between 10 and 11 per cent (i.e. 85 to 95 kg. C), although local circumstances often have made it necessary to work with as little as 8·5 per cent on pyrites for many months. With 10–11 per cent coke the blast-furnace recovery is just over 60 per cent.

Gas Cleaning

Because the Cottrells had to deal with a much larger volume of gas than that for which they were designed, it was early found to be necessary, in order to reduce pressure in the system, to install a large exhauster fan on the exit side of the cold Cottrells. This fan has a capacity of 32,000 c.f.m.

The dust precipitated is, to some extent, pyrophoric, and occasionally sticky and difficult to handle ; this condition is brought about by the fact that it always contains sulphur, which will catch fire if air is being drawn into the filters. The hoppers in the bottom of the Cottrell chambers, in which the dust is collected and from which it is withdrawn, are fitted with double-bell doors, so that the dust can be withdrawn without admitting air into the Cottrell chamber ; dust extraction by means of screw conveyors has been tried, but has been given up as unsuitable, the dust being too sticky.

The percentage of dust recovered upon the burden smelted is approximately 0·57 per cent.

A typical dust recovered at Rio Tinto had the following chemical composition and screen analysis :

Chemical Analysis

	Per cent
Cu	1·3
S	25·32
Pb	21·20
Zn	4·97
Fe	13·20
As	2·93
SiO_2	6·28
Al_2O_3	1·57
CaO	3·08
MgO	0·70
Sn	0·42
C	5·09
Sb	1·21
Bi	0·38

Dry Screen Analysis—Tyler Mesh

+ 60 = 12·0 per cent
— 60 + 100 = 22·0 „ „
— 100 + 200 = 30·57 „ „
— 200 + 325 = 24·0 „ „
— 325 = 11·43 „ „

The ore smelted contains on an average at least 1 per cent of lead, present as a sulphide. This compound is very volatile and some of it comes over with the gases, being partly reduced to metallic lead in the process ; for instance, in December, 1944, the Cottrell dust contained 7 per cent of free metallic lead and 8·7 per cent of combined lead, a total of 15·7 per cent. Lead fume is a very difficult material to precipitate and, if the Cottrells are not working very efficiently, the metallic lead, *plus* lead sulphide, is carried over with the sulphur vapour and causes a great deal of trouble in the subsequent washing and filtering operations. Fortunately, it is only a minor portion of the total lead charged which is retained in the crude sulphur, as the following lead balance, made for the month of June, 1944, shows :

Total tons of Pb charged to furnaces, 99 tons.

Distribution :

In matte	49·6 per cent of total lead charged
„ furnace slag	21·2 „ „ „ „ „ „
„ Cottrell dust	20·5 „ „ „ „ „ „
„ sulphur	8·7 „ „ „ „ „ „

Lead sulphide is sometimes found consolidated in quite large pieces, which are drawn out with the Cottrell dust.

Sulphur Washing

This is a true refining process, consisting as it does of reducing a very considerable percentage of arsenic, 1·5–2·3 per cent, contained in the sulphur, down to 3–10 parts per million (0·0003–0·001 per cent) ; at the same time the ash is virtually eliminated.

A typical analysis of the crude sulphur to be purified is :

	Per cent
S	97·62
As	2·10
Pb	0·11
Fe	0·017
Zn	0·016

Minor amounts of Cu, SiO_2, Al_2O_3, Bi, Sb and CaO are also present.

Fig. 6 shows details of a unit of the washing plant.

The autoclaves in which refining is done are vertical cylinders 16 ft. long by 6 ft. wide and they can withstand a working pressure of 50 p.s.i. The volume or cubic capacity is 452 cu. ft. or 12·8 cu. m.

The milk of lime is made up in a large open agitator tank ; a sample of the suspensions is drawn every four hours and tested to

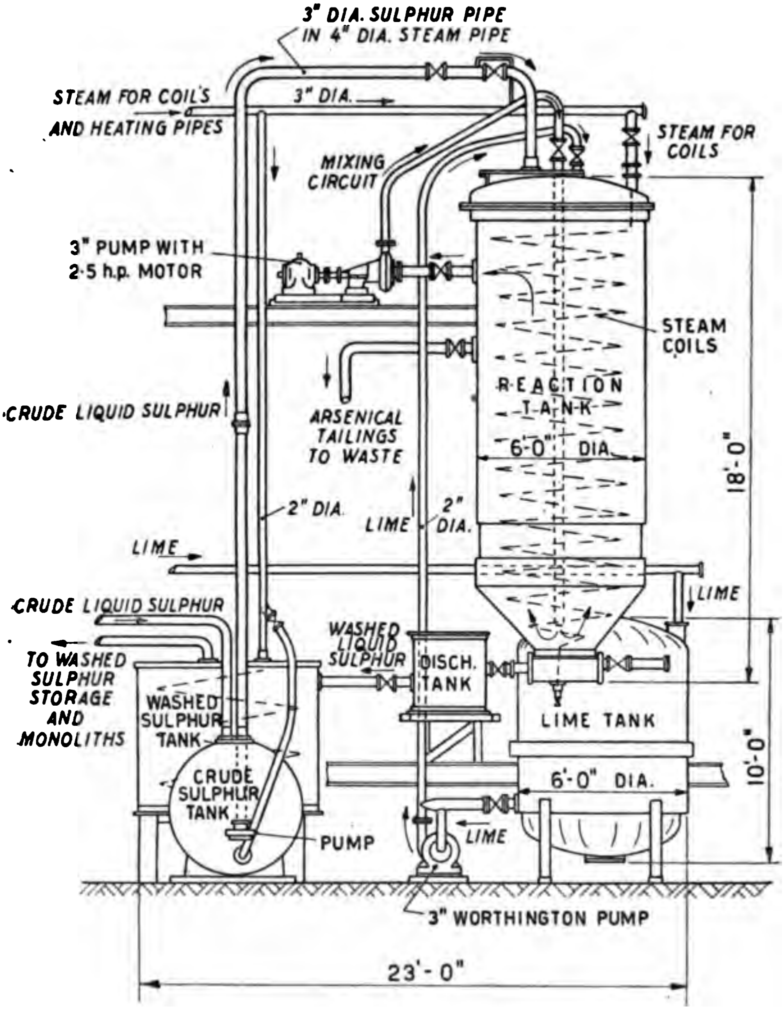

FIG. 6.—Sulphur washing plant—arrangement of single unit.

determine its free CaO content. The amount of arsenic in the crude sulphur and the free CaO content of the suspensions determine the amount of milk of lime required. It is run by gravity to the pre-heating tank, which is then filled up with water to a fixed volume ; the pre-heating tank then contains 4½ cu. m. of diluted milk of lime. Steam at 25–30 lb. pressure is now turned into the tank in which, if necessary, the milk of lime can be circulated by a pump, and is kept on until the temperature of the limewater is about 120°–125°C. This heating takes about half an hour. The milk of lime is then pumped into the autoclave. If the crude sulphur is charged into the pressure vessel before the milk of lime, the sulphur is liable to solidify at the bottom of the vessel and stop up the end of the circulating pipe. The crude sulphur is therefore pumped to the autoclaves after they have received a charge of milk of lime.

All entrance cocks on the autoclaves are then closed, and the circulating pump, capable of handling 90 cu.m./hr., is started. The milk of lime is drawn off near to the top of the vessel, about 2 ft. 6 in. down, and is forced by the pump through a 4-in. pipe, which passes through the cover of the autoclave and down to within about 9 in. of the bottom of the vessel ; in this way a very violent and intimate mixing of the sulphur and milk of lime is obtained, so that the arsenic can be reduced from about 2·0 per cent down to about 5 parts per million in two hours. During the two hours in which the lime suspension is being circulated, the pressure in the autoclave rises to about 45 p.s.i.

When the scrubbing period, during which the suspension is pumped and circulated through the sulphur, is finished the pump is stopped and the contents of the autoclave are allowed to settle for 20 to 30 min. At the end of this period of repose the contents of the autoclave have separated into three distinct layers, because of the difference of their specific gravities. The uppermost consists of the arsenical liquors, the middle of an emulsion of sulphur and water and other insoluble material, and the lower one of normal clean, washed sulphur. Frothy residue is only obtained with high ash content in the crude sulphur ; if ash content is less than 0·1 per cent no residue is formed.

The arsenical liquors are discharged through a long 3-in. pipe in which the hot liquor is mixed with cold water, and then the clean sulphur is run into the refined sulphur tank. When dirty, or frothy sulphur appears the sulphur cock is closed and finally the residues are blown out under pressure through a 4-in. pipe direct into wagons. The sulphur residues amount to about 6·5 per cent, by weight, of the crude sulphur washed ; an analysis of these residues is as follows :

	Per cent
S free	92·26
S combined	1·03
S total (includes Se)	93·29
As	0·60
Pb	1·22
Cu	0·06
SiO_2	0·53
Al_2O_3	0·32
$CaSO_4$	3·52
MgO	0·13

The following is a typical analysis of the ash in the crude sulphur :

	Per cent
Pb	25·5
PbS	29·7
Cu_2S	1·94
Bi_2S_3	0·67
As_2S_5	5·57
Sb_2S_5	3·81
ZnS	12·48
FeS	10·25
Al_2O_3	3·51
SiO_2	3·29
CaO	2·25
MgO	0·72
Mn, Ni, alkalis	traces

After the washed sulphur has been run into the storage tank it is elevated by an immersed pump to one or two open storage tanks, which are, of course, well lagged and fitted with internal steam-heating coils. From these tanks, it is forced by immersed centrifugal pumps, of the Company's own design and make, through one of two steam-heated filter presses ; each press has 30 leaves, which are 28 in. by 31 in. The average capacity per press is 10 tons of filtered sulphur per hour, with a maximum of 16 tons an hour. Ordinary canvas is used as the filtering medium ; wool was tried in the early days of the process, but gave no better results although it was much more expensive. All the frames are steam-heated, so that no sulphur can solidify during filtration. Each pump has a capacity of 25 cu. m. of sulphur per hour against a pressure of 40 p.s.i.

The rapidity with which the filtering operation proceeds, and the amount of sulphur which one set of filter cloths will serve to filter, depends upon how well the hot Cottrells have cleaned the gases and thus lowered the ash content of the crude sulphur. If the ash content is very low, not exceeding 0·10 per cent, a set of filter cloths will filter up to 300 tons of washed sulphur, and the filter press cake will build up to its maximum thickness of just over one inch. In this case the filter press residues will contain little lead, but a higher proportion of insoluble matter derived from the lime. On the other hand, in the event of the ash in the crude sulphur being really high—from 0·5 per cent to 1·0 per cent—a set of filter cloths will not filter more than 25 tons of sulphur, sometimes only 8 tons, because the metallic lead and lead sulphide contained in the ash at once block up the pores of the canvas so efficiently that no sulphur can be forced through by the pump ; it is as though the canvas had been varnished with a metallic varnish and if the pump pressure is increased it only results in the bursting of the canvas. Typical analyses of both kinds of residues are given below.

	High Ash *per cent*	*Low Ash* *per cent*
Free S	57·0	57·27
Combined S	7·9	11·69
As	2·93	1·86
Pb	12·67	2·56
Cu	0·09	0·06
SiO_2	0·36	4·06
Zn	3·63	0·80
Fe_2O_3	1·27	1·84
Al_2O_3	0·38	1·75
CaO	4·43	14·03
MgO	0·14	0·69
O, etc.	9·20	3·39
	100·00	100·00

The physical condition of these filter press residues is quite different from that of autoclave residues ; they are much denser, present a stony, clinkered appearance, and are difficult to break down, whereas the autoclave residues resemble very friable pumice stone because water was present when they solidified. Filter press residues are also much darker in colour, being black, while the others have a yellow-green colour. These filter-press residues are separated as far as possible and thrown away, to avoid a high lead concentration in the furnace charge.

The sulphur balance of the washery operations is, over a long period, approximately as follows :

	Per cent
Filtered sulphur to monoliths	84·0
Sulphur residues and filter cake resmelted	6·0
Lost in arsenical liquors, filters, etc.	10·0
	100·0

The filtered product flows to a 60-ton steam-heated storage tank, from which it is pumped by immersed centrifugal pumps, through steam-heated pipe lines to the storage monoliths.

Recoveries

The actual recovery of sulphur at Rio Tinto rarely exceeds 55 per cent, and in order to achieve this it has been found necessary to use between 9·5 per cent and 11 per cent of coke on the pyrites. The major loss of sulphur undoubtedly occurs as unreduced SO_2 in the furnace gases, and, so far, it has not been possible to reduce their content with any economic consumption of coke, nor are there any indications that improved recovery is likely to result solely from the use of largely increased quantities of a carbonaceous reducing agent, because ' what is gained on the swings is lost on the roundabouts ' ; the sulphur reduced from the SO_2 forms other compounds with carbon and no net increase of sulphur results.

It will no doubt be pointed out that the actual recovery of sulphur which is being made is not very much more in quantity than that due to loose atom sulphur only, and it may be argued that a recovery of sulphur nearly as good as that actually being made could be achieved by reducing the coke to, say, ½ per cent on mineral, and running the furnace as a true pyrite furnace. Experiments along this line have been made at Rio Tinto and at Mina de S. Domingos, but so far no success has been obtained in running closed-top furnaces as true pyritic units ; eventually, the reduction of coke has always led to a falling-off in the production of sulphur. The difficulty lies not so much in the actual operation of the furnaces, although troubles develop if the coke is reduced below 5 per cent on the pyrites, as in the change in the composition of the gases ; these become much poorer in sulphur and dustier, with the result that they are less efficiently cleaned by the Cottrells ; the arsenic-sulphur ratio in the gases is also altered in such a way that arsenic-sulphur compounds condense and are deposited at a higher temperature than normal. Experience has definitely proved that the loss of sulphur more than offsets any saving made in coke. Further, certain experiments conducted in high-temperature furnaces seem to indicate that there is, in fact, no dissociation beyond Fe_nS_{n+1} at any temperature attainable in a copper-smelting furnace.

The authors' conclusion is, therefore, as already shown in Section 1, that a fair amount of SO_2 formed by the combustion of FeS in the focus of the furnace is reduced by coke, and that the amount of volatile sulphur distilled from pyrites by heat alone is probably not in excess of that required to form a compound approximating to Fe_7S_8.

Conclusions

A process which only recovers, at best, some 55 per cent of the sulphur in the pyrites fed to the furnaces can hardly be accepted as efficient. The inherent difficulties involved in attempting the oxidation of iron sulphide with efficient reduction of SO_2 in one and the same vessel and in contiguous zones within that vessel have already been pointed out. Much still remains to be done and eventually, in spite of the inherent difficulties, greatly improved recovery may be attained, even with the difficult Iberian ores. The two principal lines along which improvement is being sought to-day are :

(1) Greater oxidation of FeS and greater reduction of SO_2 by more intensive working of the furnace with resulting higher temperature.

(2) Possible treatment of exit gases by catalysis after removal of the bulk of the arsenic.

It is hoped that within the next three to four years it may be possible to contribute a further paper giving an account of improvements in the process.

ACKNOWLEDGEMENTS.—The authors acknowledge with most grateful thanks the invaluable assistance they have received from Mr. Salkield, of the Metallurgical Staff of the Rio Tinto Company, and they also take this opportunity to pay tribute to the work of a former member of the Company's Metallurgical Staff, the late Mr. L. A. Lawrence, upon whom fell many of the trials and tribulations incidental to starting a new process and bringing it to the stage of successful operation.

Their thanks are due to the Directors of the Rio Tinto Company for permission to publish the information contained in this paper.

APPENDIX—TABLE A

OPERATING RESULTS 1936–1938 INCL. AND CALCULATION OF REDUCTION BY CO AND C

1,378 *Furnace days*	*For three years 1936–38*			*Per furnace day*			*Sulphur per 1,000 kg. Pyrites (kg.)*	*Per cent of Total*
	Tons	*Per cent S*	*Sulphur, tons*	*Tons*	*Per cent S*	*Sulphur, tons*		
Pyrites	233,332	47·50	110,734	169·5	47·50	80·40	—	—
2nd class ore	17,879	38·25	6,809	12·9	33·25	4·94	—	—
Siliceous ore	1,991	24·60	491	1·45	24·50	0·36	—	—
Concentrates	87	41·20	35	—	—	—	—	—
TOTAL	253,289	—	118,069	183·85	—	85·70	—	—
To Furnaces:								
Pyrites equivalent of 2nd class and siliceous ores	15,400	47·5	7,335	—	—	—	—	—
Pyrites	233,332	47·5	110,734	—	—	—	—	—
TOTAL	248,732	47·5	118,069	180·3	47·5	85·70	475	—
Converter slag	30,739	1·4	430	22·2	1·4	0·31	2	—
Matte	4,982	24·45	1,223	3·6	24·45	0·89	5	—
Residues	5,313	85·30	4,530	3·9	85·30	3·23	18	—
Silica flux	62,049	—	—	45·0	—	—	—	—
Limestone	21,474	—	—	15·6	—	—	—	—
TOTAL	—	—	124,252	—	—	90·13	500	100·0
Coke	25,162	—	—	18·25	—	—	101·2	—
From Furnaces:								
Crude sulphur	78,613	97·60	76,587	57·00	97·60	55·60	309	61·8
Matte	64,437	24·50	15,764	46·80	24·50	11·40	63	12·6
Slag	208,805	2·62	5,451	151·00	2·62	3·96	22	4·4
Dust	1,648	18·91	312	1·20	18·91	0·23	1	0·2
Clean up	76	—	1	—	—	—	—	—
Gases	—	—	22,800	—	—	16·50	92	18·5
Unaccounted	—	—	3,337	—	—	2·44	13	2·5

APPENDIX—TABLE B

	Total	g./cu. m.	
		C	*S*
CO_2 per cent by volume	12·9	69·1	—
CO „ „ „ „	0·5	2·7	—
SO_2 „ „ „ „	2·34	—	43·5
H_2S „ „ „ „	0·18	—	2·55
COS g./cu. m. „ „	15·40	3·0	8·20
CS_2 „ „ „	17·30	2·8	14·50
S „ „ „	4·00	—	4·00
TOTAL	—	77·6	72·75

	kg.
Carbon in coke per 1,000 kg. pyrites	88·20
„ „ limestone	8·60
	96·80

Gas Volume :

$$\frac{96\cdot8 \text{ kg. C in charge}}{77\cdot6 \text{ g./cu. m.}} = 1{,}250 \text{ cu. m. per ton pyrites}$$

		kg. S per 1,000 kg. pyrites
1. *Calculation of volatile sulphur*		
Total sulphur in pyrites		475·0
Less sulphur combined with Cu, Pb, Zn, etc.		19·0
Sulphur in FeS_2		456·0
Therefore, volatile sulphur (42 per cent of sulphur in FeS_2)		191·5
Add : Sulphur in residues		18·0
Total volatile sulphur		209·5
2. *Calculation of sulphur recovered by reduction of SO_2*		
Total sulphur charged to furnace		500·0
Deduct volatile sulphur, volatilized		209·5
Fixed sulphur entering smelting zone as Fe_7S_8 and non-ferrous metal sulphides		290·5
Deduct : Sulphur in matte	63·0	
slag	22·0	
dust	1·0	
		86·0
Therefore, sulphur oxidized by blast to SO_2		204·5
Sulphur lost in gases and unaccounted		105·0
Balance, sulphur recovered by reduction of SO_2		99·5

Sulphur produced by reduction of SO_2 and collected....			99·5 kg.
„ „ „ „ „ „ but subsequently reacting with C or lost as S vapour :			
As CS_2	14·50	g./cu.m.	
„ COS	8·20	„	
„ H_2S	2·55	„	
„ S	4·00	„	
Total	29·25	„	
		× 1,250=	36·5 kg.
Total S from reduction of SO_2			136·0
Carbon available			88·2
Less carbon in CS_2, CO and COS			10·5
			77·7

Let x = carbon acting as C

y = „ „ „ CO

$$x + y = 77\cdot7$$

$$\frac{32x}{12} + \frac{32}{24}(y - 0\cdot1y) = 136\cdot0$$

whence $x = 29\cdot2$ kg. C

$y = 48\cdot5$ „ „

Sulphur reduced by C = 78 kg. = 57 per cent

„ „ „ CO = 58 „ = 43 „ „

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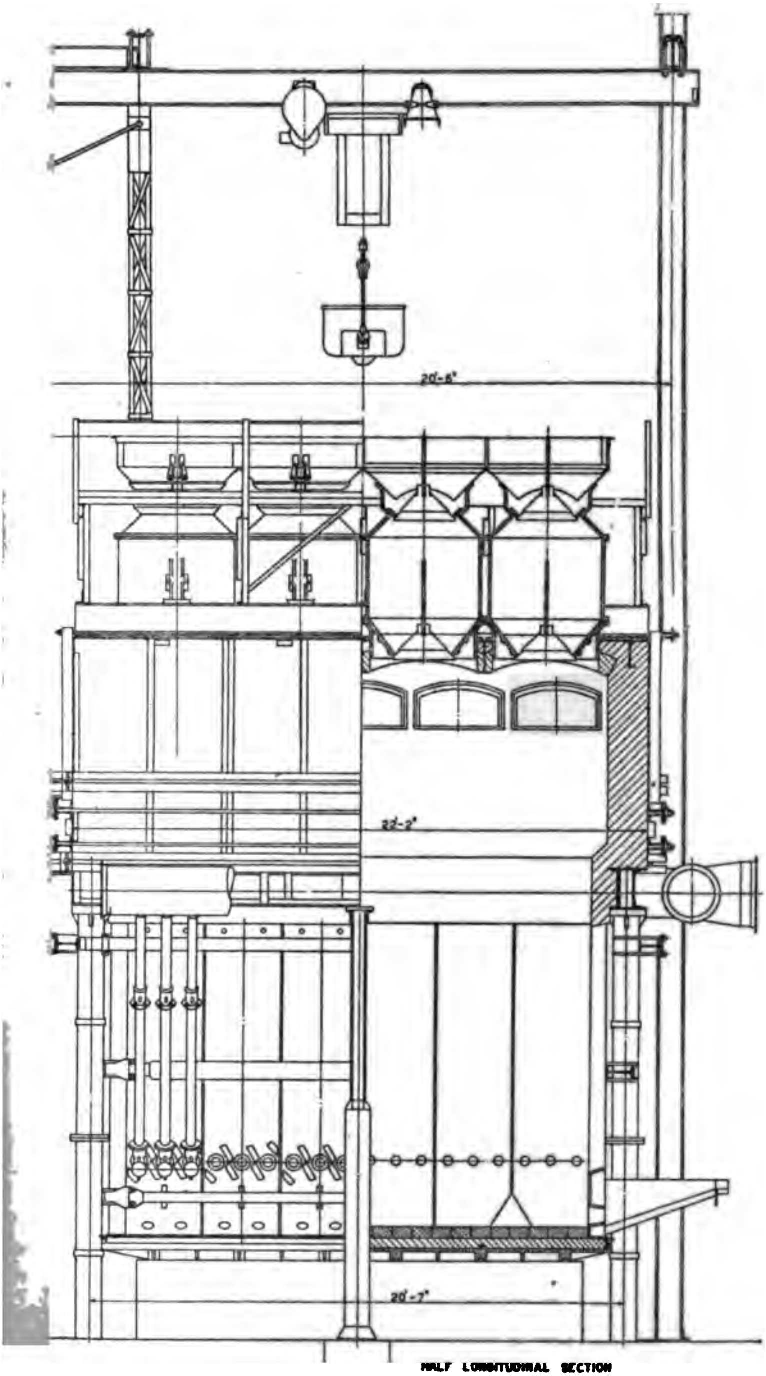
Plate I.



HALF LONGITUDINAL SECTION

THE INSTITUTION OF MINING AND METALLURGY

FIFTH ORDINARY GENERAL MEETING of the 58TH SESSION held in the Rooms of the Geological Society of London, Burlington House, Piccadilly, London, W. 1, on Thursday, 17th February, 1949.

Mr. S. E. TAYLOR, *President*, in the Chair

DISCUSSION ON

Ground Control—Theory and Practice

By JACK SPALDING, *Member*

The President said that the author of the paper was well known and did not need any introduction to members. It was twelve years since he presented a paper on the same subject; further researches had been made and much experience gained during the interval, while the theory of the subject had also been advanced. The author was now giving the Institution the benefit of the progress of the knowledge and thought which he had brought to bear upon that complex problem and it was fortunate that he could be present to introduce his paper and take part in the subsequent discussion.

Mr. Jack Spalding said that the state of a rock under stress was a most complicated but interesting study, of which the knowledge of the mining community was very meagre. He understood that a symposium on the subject was now being prepared at Heerlen in Holland. He would like to stress that the following remarks were his *ideas* on the subject—they must not be taken as statements of fact. They were offered simply to stimulate discussion, in the hope that knowledge might thereby be increased.

In considering the effect of stress on rock it was frequently assumed that the rock in question was a solid homogeneous substance. Tests were made on specimens, and elastic constants and crushing strengths estimated. While the figures obtained might be true for a rapid application of stress—such, for instance, as was imposed in mining when a mass of rock was blasted out—they should not be applied to rocks which had been submitted to stress for geological periods of time. That was largely due to the fact that rocks were generally not solid homogeneous substances. Granite, for example, was a solid compact rock, free from voids, but it was built up of a number of constituents of widely different properties. If a specimen of granite were submitted to a steadily-increasing stress occasional grains of various constituents would be so disposed that their natural cleavages permitted them to avoid the stress by shearing movement. That increased the stress in the neighbouring grains. As the loading increased there came a time when the grains of the weakest constituent yielded and became plastic, and a further rearrangement of stress was caused within the specimen and a further loading on the other constituents. According to the cleavability and orientation of the various crystals and to the proportion and distribution of the constituent that had failed, the next weakest constituent might or might not be brought to the yield point by that failure. Should it yield, the remaining constituents would be still more highly stressed, and would in turn fail one by one in increasingly rapid succession until the specimen as a whole was fractured. In that case then, the failure took place at a little above the yield point of the weakest ingredient.

Thus failure through the chain of successively-stronger constituents might take place so quickly as to be almost instantaneous, or there might be delay between each separate action, as the very slow movement of plastic flow operated. It was merely a question of the relative proportions and strengths of those constituents. On the other hand, should the weakest constituent be so sparsely distributed that its failure did not stress the other constituents to the yield point, then further stress must be applied before the next weakest yielded, and so on. In that case the rock as a whole was stronger than its weakest ingredient.

Some crystals, however, subjected to stress for a very long period of time, although that stress might be below the yield point of the weakest ingredient, would shear along their cleavages, others would become twinned, and some projecting corners of crystals would become plastic, owing to the unequal distribution of the stresses. Chemical changes might also occur, owing to the stress or to other agencies, with the result that some crystals might grow at the expense of others.

The result of all those actions would be a slow and gradual re-crystallization of all or the majority of the constituents. The effect of any such re-crystallization, whether due to stress, temperature, internal chemical change, or chemical change due to external additions or subtractions, would be to relieve any component of stress which was greater than the others.

Tectonic movements of the earth's crust were usually too large in proportion to the time over which they operated to permit a complete equalization of stress in that way, and so the rocks yielded to them by folding, faulting, cleaving, cracking, and all the other effects familiar to the structural geologist. However, in a solid rock mass which had lain undisturbed for a long period of time, and in which one component of stress was originally greater than the other, whether that extra stress was due to the weight of superincumbent rock or to residuals of tectonic stresses, it was suggested that the rock would slowly yield to the greater component of stress until that stress was reduced to near that of the other components. It was further suggested that this took place, always assuming that sufficient time was available, not *only* at the great depths at which it was estimated that the rock *as a whole became* plastic, but at depths comparatively much less.

Assuming the truth of those suppositions, a state of stress of hydrostatic character could be expected in solid compact rocks, of mixed constituents, at depths common in deep mining. Where overlying strata had been removed such a state of stress could be expected right up to surface, but where the upper rocks were of recent origin and of open texture, or where they contained voids, it was to be expected that the vertical component of stress would be greater than the horizontal.

He did not think it was sufficiently emphasized in the paper that the theory of rock pressure there put forward was built on the assumption of a rock mass which was solid and free from voids, either inter-granular, or in the form of a number of vughs in fissures—such a rock mass, in other words, as was commonly encountered in deep metalliferous mines.

In a different type of rock—a mass of solid quartz for example—there was only one constituent, silica, all in grains of the same order of size and lacking prominent cleavage. Under an increasing applied loading a specimen of that rock would be expected to withstand the stress until the yield point of silica was reached, at which point the whole specimen would fail as one. It would seem that such a rock would be more liable to rockburst—that is to fail suddenly and with violence—than one composed of various constituents which would yield progressively.

A third type of rock, a sandstone consisting of quartz grains imperfectly cemented together with silica, also consisted only of silica ; but between the grains there were voids, and under stress the quartz was free to expand into those voids. The behaviour of the quartz grains would therefore be different from that of crystals which were rigidly confined on all sides by other crystals. Under suitable conditions, there also slow re-crystallization of the distorted grains was to be expected, but until the voids had become completely filled up in that action, converting the rock into a quartzite, the build-up of a stress of hydrostatic character could not be expected. The physical properties of silica being what they were, the conditions necessary to start re-crystallization in a quartz rock were likely to be more severe than in rocks containing some weak and cleaved constituents.

He mentioned again that his remarks were merely tentative, but that did not mean that if they were ill-founded the theory in the paper broke down. There were sufficient other arguments brought forward to make the assumption of equal stress a reasonable one in deep mining conditions, and the present argument was introduced merely to ventilate the subject the more completely.

In any mining operation involving the control of ground it was most important that the operator should have a proper conception of the movements which would occur during the closure of the stope walls—how they would close in (by shear or bending), to what extent they would close in (depending of course on the method of control), and, most important of all, the effect of that closure on any *materials, excavations, structures*, etc., the use of which was contemplated. For instance, the lining of drives on lode might be considered. It was wasteful to put in an expensive lining for any drive, if, after the lode was stoped, the closure squeezed the lining to such an extent as to prohibit the passage of cars. The total amount of closure expected must be gauged and the lining made large enough to be still sufficiently big when closure was complete. The extra expense of that larger lining would be less than the cost of complete relining after stoping.

He would give an example of the effectiveness of being prepared for closure. He once took over a stoping section which was served by a small timbered incline shaft, sunk on a lode dipping at 45°. The depth below surface was 3,000 ft. The lode was 10 to 12 ft. wide and exceedingly rich. The ore-shoot had been completely stoped out, except for a chain of pillars measuring 30 to 60 ft. on the strike each side of the shaft. The instructions were to win the pillars.

Mining was started by underhand stoping from the top down. Massive supports of granite masonry were used, so that closure, and therefore damage to the shaft lining, would be reduced to a minimum. As the stopes went down, the shaft timber was removed and replaced by a brick arch with a plentiful cushion of packing round. The main level was at the top of the shaft and the support of the shaft station at that point was the problem. Closure would inevitably occur right across the station, which ruled out the possibility of using a rigid concrete lining: moreover, rockbursts were to be expected, which militated against the use of timber.

It was decided to use a rigid lining of concrete, divided into two separate units, one (B in Fig. 22) founded on the foot-wall and integral with the shaft lining (AC), and the other (D) attached to

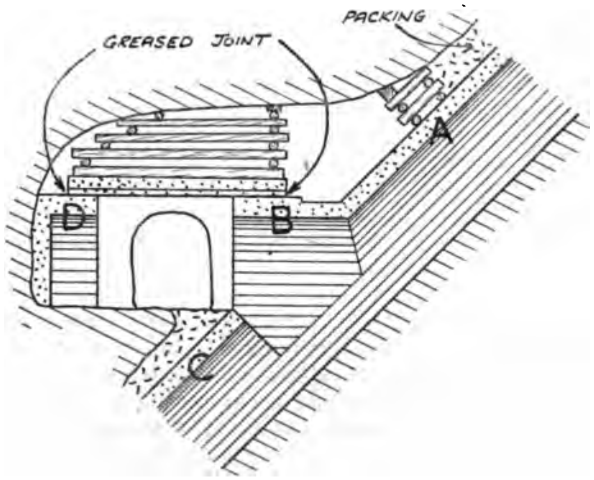


FIG. 22.

the hanging-wall only. The arches B and D were made of concrete and constructed with flat tops, as shown in Fig. 23. The sides of

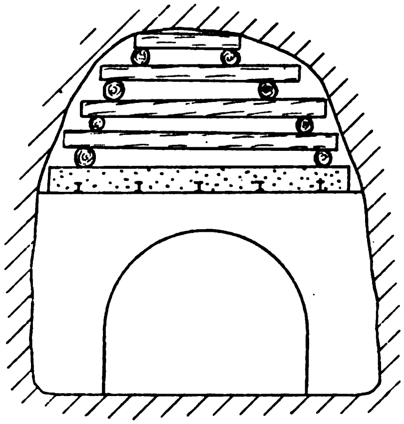


FIG. 23.

the station were left unsupported. The tops of B and D were smoothed off and greased. Steel girders (6-in. by 3-in.) were then thrown across from B to D at 3-ft. intervals, and concreted in as shown. The concrete was made 15 in. thick. The resulting flat roof was fully floating and it was hoped that relative movement of ABC and D could take place without causing damage. When the stope faces had worked down to 150 ft. below the shaft station, there had not been the slightest sign of any movement taking place. However, when the distance between B and D was checked it was found to be 6 in. less than when constructed.

The chain of pillars had originally been hour-glass shaped, and the stopes had now reached the waist, so that ahead of the faces there was a bare 30 ft. of solid each side of the shaft (Fig. 24). At that point a violent rockburst occurred. The shaft was wrecked for 100 ft. below the faces, and choked solid with broken rock, but damage in the stopes was comparatively trivial.

That rockburst was a failure of the rock under the excessive ring stresses caused by the heavy loading on the pillar—it was an extreme example of violent arching. The shaft had burst to an effective diameter of 20 ft., as was clearly to be seen during clearing operations in the back of the next station below (Fig. 25). The burst had no visible effects on the top station, but on measuring up again it was found that B and D had closed in another couple of inches.

Between that station and the stope faces the brick-arch lining of the shaft was damaged by the sudden movement of closure that accompanied the rockburst. The cushion of packing round that arch, already *compacted by the normal* closure that had occurred, was insufficiently resilient to absorb the live load occasioned by the rockburst.

The operation was described in some detail because it furnished an excellent example of what could be done by allowing for the

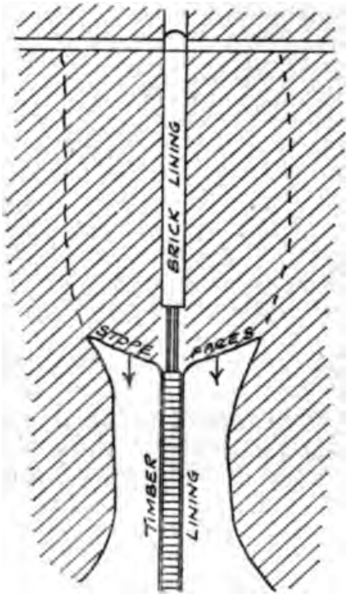


FIG. 24.

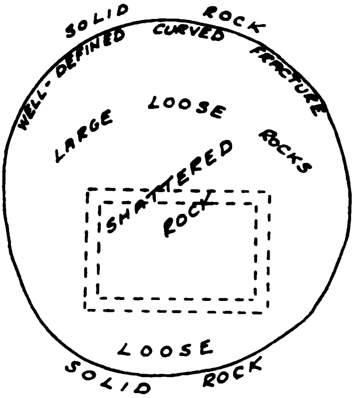


FIG. 25.

effects of closure, as was exemplified in the shaft station, and because it also demonstrated the result of not having a proper conception of those effects, as was shown in the case of the shaft lining.

The President said that he had particularly asked that students should open the discussion, partly because he was very keen that students should take part in the discussions, but also because much of the paper was theory founded upon what they had been studying, so they should be able to contribute something very useful to the problems under review. He would call on two Royal School of Mines students, Mr. Gray and Mr. Sales.

Mr. K. G. Gray, referring to Fig. 13, said he noted that the stress envelope was of a different shape, dependent on the different types of support. He rather gathered from the paper that the stress envelope was formed when excavation was made and a circle of no stress formed along the inner circumference of that circle, so that in either case, before any support was put into the excavation, it seemed to him that the same fracture dome must be *supported.* *In* the diagram it was shown that one fracture dome was very much larger than the ones at the bottom. At first that fracture dome was supported by the adhesive binding force between the rock in the dome and the rock in the stress envelope, and that support would depend on the tensile and shear strength of the rock itself. The tensile strength probably decreased with time owing to the creep of the rock, so that the weight of the rock in the fracture dome might thus become greater than the adhesive strength along the surface and the dome would fail. The rôle of the support was, he suggested, to make up the discrepancy between the initial strength of the rock and the weight of the rock in the dome.

The point he wanted to make was that he did not see why the fracture dome was different in the two cases. As far as he could see it was the same, but, in the lower drawing with the rigid support, there was the adhesive strength at the dome circumference added to the strength of the support. If the rigid support were subject to permanent compression the fracture would tend to form as in the top diagram and it would appear to him that the only way of preventing that fracture forming in time was to have an elastic and rigid support. The only one which would suit those conditions permanently was a steel support, being the only elastic material possible; wood might splinter and collapse and concrete would be permanently deformed in the same way as the rock was deformed when pressure was relieved. With a steel support, on the other hand, as the strain increased owing to creep of the dome, so in proportion would the force, aiding the adhesion, be transmitted to the dome. He would like the author to explain why different stress envelopes were set up, depending on the type of support.

Mr. T. J. R. Sales referred to Fig. 6 relating to effective excavation. The figure gave what the author claimed to be the practical application of the theory using the two mathematical formulae given at the beginning of the paper. The whole application of the mathematical formulae seemed to depend on the 'effective excavation' being known exactly; if it were not known exactly, when '*a*' was squared an even larger error than before would be obtained. He would like to know, therefore, if the radius of the effective excavation could be found theoretically. In the example given that was apparently arbitrarily selected from the curve in Fig. 5 of the paper: indeed, it appeared to be selected from the dotted curve, which was also calculated in the formulae using the effective radius: the $1+1\frac{1}{16}\,Q$ seemed to be arbitrary and had no connection with the theory. If that could be determined correctly it would be of great value in driving drifts, especially in coal mines in Britain. It would help in reducing leakage of ventilation if one could find the exact limit to the area of disturbance which caused it. He would like to see the question cleared up.

Professor W. E. G. Sillick* said that at the commencement of his paper the author had stated that since his paper in 1937 'further knowledge and experience have been gained, in the light

*Department of Mining, Royal School of Mines.

of which the theory there propounded and its applicability to mining practice have been expanded. In the present paper the theory of rock pressure . . . has been brought up to date and from the conclusions reached the theoretical and practical aspects of ground control have been deduced '.

The paper was so long that he had not had time to read much more than the first few pages devoted to theory and perhaps it was at that stage that he could make the best contribution. The material with which they were dealing was of such a curious nature that he looked upon it as something intermediate between a piece of spring steel and a packet of face powder. He preferred to keep down to the face powder end of the scale and treat the material as granular, because he did not think he was quite so interested in it during the period in which it was completely elastic. He was more interested in the applicability of the diagrams to the case when the elasticity region was passed.

In his previous paper Mr. Spalding said that he had opened up places where the stuff came out like dust, which indicated that the material had failed long before the opening was made. The mass of the rock at depth was subject to exceedingly great pressure vertically and also to horizontal pressures of great and unknown amount. It was easy to conceive that the resultant crushing of the material went far beyond the limit of its elasticity, that the

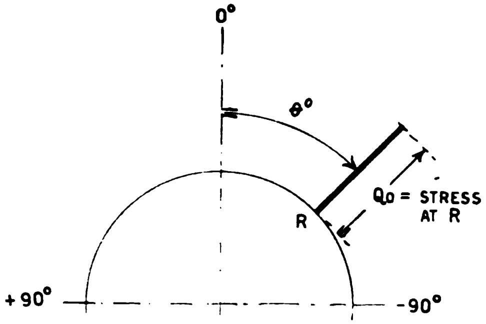


FIG. 26.

material failed completely, and that, having been reduced to a powder, it was able to transmit very great pressure by exerting a compressive resistance—which it was enabled to do by virtue of its being confined. As soon as an opening was made, the equilibrium of the material was disturbed. Some of it immediately escaped, and eventually, due to the flow towards the hole, a new state of equilibrium was established.

The re-distribution of stress around the opening could be calculated if one made certain assumptions, and there were two ways of showing the re-distribution graphically :

(i) that adopted by the author, using rectangular co-ordinates, *the base of the* diagram being distance from the centre of the hole, and the ordinates the stress at that distance in terms of the stress which existed before the hole was made ; and

(ii) that which the speaker would suggest, using polar co-ordinates, in which the axis of co-ordinates was taken at the centre of the hole, the profile of the hole was taken as the base line, and the radius vector was set out, having a magnitude equal to the stress at the edge of the hole at that point (Fig. 26).

The author's method illustrated variation in stress in depth away from the free surface of the hole into the material, and for hydrostatic loading only. The speaker's method illustrated the variation in stress at the free surface all round the opening and was applicable for hydrostatic and all other forms of loading, or ratios of vertical to horizontal loading.

The polar diagrams were quite easy to construct. It could be shown that, for a single applied stress Q, the stress Q_o at any point around the edge in the case of a circular opening was :

$$Q_o = Q\ (1-2\cos 2\theta),$$

where θ was measured as shown in Fig. 26, the direction $\theta=0°$ being chosen parallel to that of the applied stress Q.

If the polar curve were plotted, by giving values to θ, up to $\theta=90°$. the result was as shown in Fig. 27. The other quadrants

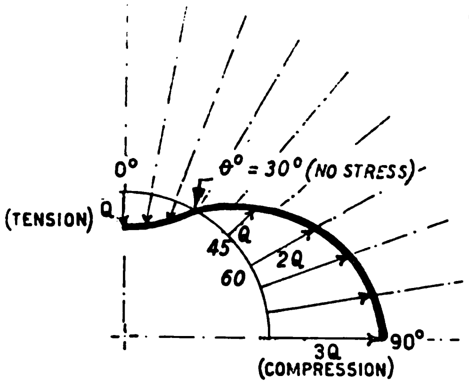


FIG. 27.

were similar and the whole polar diagram looked like Fig. 28.

That was most interesting, especially because it showed :

(i) That the stress at G was compressive and was equal to $3Q$.

(ii) That the stress at T ($\theta=0°$) was equal in magnitude to Q but was of the opposite sign to Q and was therefore tensile.

(iii) That at F ($\theta=30°$) the stress was zero.

(iv) That the tensile region extended 30° on each side of T at the top, and that there was a similar tensile region extending for 30° on each side of B, at the bottom edge of the hole.

Having thus illustrated the nature of the stress variation for a single stress Q applied in one direction, the corresponding diagrams could be obtained for *any multiple* of Q and for any other direction,

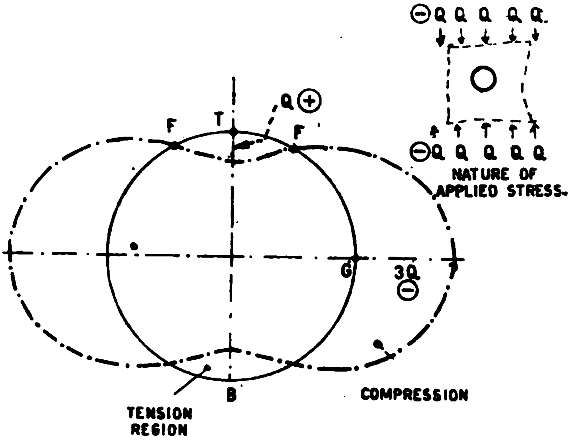


FIG. 28.

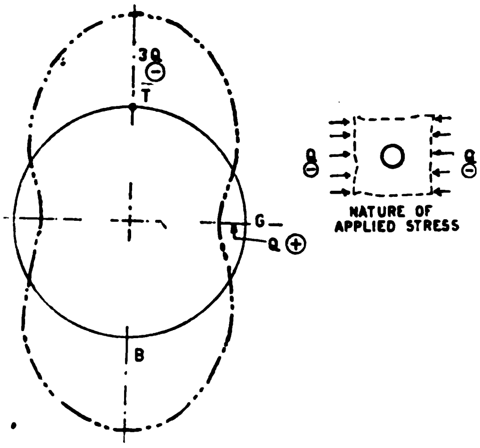


FIG. 29.

and, by combining the diagrams, the result when they acted simultaneously could be examined.

The speaker wished to take just two particular cases only :

(*a*) Mr. Spalding's hydrostatic loading—which amounted to Q vertical plus Q horizontal ; and

(*b*) a case in which the horizontal applied stress was half the vertical stress.

Fig. 29 showed the stress distribution around the hole for a horizontally-applied stress$=Q$.

Fig. 30 showed the result for case (*a*) above, and he was sure the *author would* be happy to note that the addition of Fig. 29 to

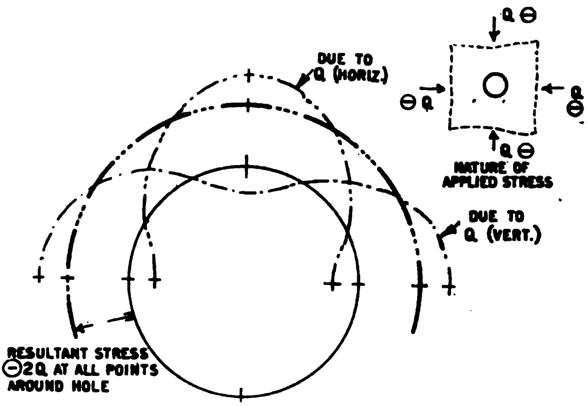


Fig. 30.

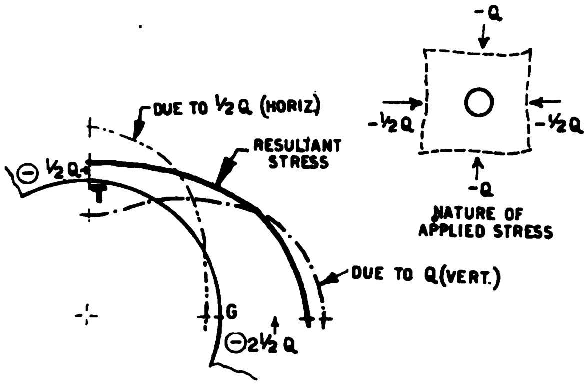


Fig. 31.

Fig. 28 gave $2Q$ as the stress at all places round the edge of the hole.

The speaker's second case (*b*) amounted to halving the radius vectors in Fig. 29 and combining the diagram thus obtained with that shown in Fig. 28. The result was Fig. 31, showing that the tension in the roof and floor was now relieved and that there was compressive stress all the way around the edge of the hole and that its magnitude varied from point to point. It was further apparent from Fig. 31 that when the horizontal applied stress was equal to $\frac{1}{2}Q$ tension ceased at T, and that as the horizontal applied stress was increased beyond Q in intensity the resultant ring stress remained compressive until the horizontal applied stress$=3Q$, when cracks would appear at G.

Returning to the face powder theory—which amounted to assuming granular inelastic material—the speaker said a relationship between the vertical and horizontal pressures in a mass of such material was established by Rankine as follows :

$$P_{H}=P_{V}\ \frac{1-\sin\phi}{1+\sin\phi}$$

where ϕ was the angle of repose for the material, and if ϕ was known in any case the likely stress distribution around the opening could be worked out.

For elliptical openings the polar-co-ordinate method was the only satisfactory way of showing the stress distribution graphically. The diagrams would show a high concentration of stress at the ends of the major axis, and from the diagrams relating to elliptical openings it would be possible to draw certain conclusions relating to rectangular-shaped openings.

There were one or two errors which had crept into the paper. Fig. 3 was plotted all wrong, the horizontal co-ordinates puzzled him. He did not know where the axis of the figure was, and he himself would have put O at the centre of the opening along the axis, but there was something else there, and perhaps the author would explain the real figures which should appear on those graduations. There were one or two other places in which the wording should be corrected : in the first paragraph under ' The Nature of Rock Pressure ', sixth line, he thought ' deduced ' should read ' assumed '—one did not know enough about the subject to deduce anything. Poisson's Ratio was introduced very subtly on the next page, and he asked for a little enlightenment on paragraph (4) because of course the vertical stress would be increased also ; it was just a question again of the ratio of the vertical stress to the horizontal stress.

Mr. D. J. Burdon said that the last speaker had raised the point whether a rock encountered in deep mining deformed as an elastic body or as a granulated mass. It would appear that it behaved partly in one way, partly in the other ; for both Rankine's formula for granular material, and the equations arising from Hooke's Law for elastic bodies seemed to yield solutions to the problems encountered. He had taken those two view-points when elucidating the different states of strain found in quartz from the Kolar Gold Field.

Mining induced stresses which caused the crenelated and interlocking edges of large quartz crystals to shear off, so that large unstrained quartz was found set in a granulated base—a form of ' mortar-structure '. Quartz which had formed part of an old pillar in a stoped-out area very often showed that phenomenon. It would appear probable that a rock composed of such material would deform further after the manner of a granulated mass.

Geological forces comparable to those which produced faulting, etc., stressed the quartz beyond its yield-point, so that a permanent deformation was given to the crystal, which showed some form of 'undulose extinction'. This deformation took place at an early age in the history of these quartz-reefs, and under high confining pressure; at that time it would appear that the quartz was stressed beyond its yield-point, but not beyond its point of rupture. Such deformation should be studied from the view-point of Hooke's Law. It was, however, probable that mining in such geologically-deformed quartz might permit the material to yield now as a granulated mass.

In his introduction, Mr. Spalding spoke of the behaviour of rock at great depth as similar to that of a fluid in the transmission of stresses. While at some period of their history metamorphic and metasomatic rocks might have so behaved, conditions were very different when they were opened up by mining. The ease with which some minerals deform was not always recognized; and it was probable that once a mineral which formed a reasonable percentage of a rock had started to deform plastically, then the whole rock was in a semi-hydrostatic state. The flowing mineral would act as a lubricant to the rest of the rock. Quartz, which appeared to have all the elastic qualities of a very hard mineral, deformed much more readily than felspar, although the crystal lattice of the latter seemed to invite yielding by twinning, etc. Mining did not seem to induce such high stresses as to cause quartz to deform plastically, probably due to the fact that, by its very nature, mining destroyed the high confining pressure which helped a rock to yield by flow and not by rupture.

Mr. J. Norman Wynne said that the subject was of particular interest to him, because he was one of a very small band associated over 30 years ago with P. J. Crowle, whom he regarded as the pioneer in combating rockbursts on the Kolar Gold Field. He vividly recalled those (literally) shattering experiences, which happened at a time when none understood their cause, much less how effectively to combat them. At that time they were called 'air blasts', until that term was officially deprecated and substituted by 'rockbursts'. The violent concussion and tremendous rush of air along the levels following rockbursts extinguished carbide lamps and candles and, perhaps naturally, the occurrences were associated with some sort of explosion. He had noticed a look of incredulity on the faces of some mining engineers when he had recounted the quite common results of rockbursts on the Kolar field. One had to experience rockbursts to appreciate what they could do in the way of damage, and that was something which the average mining man seldom had the opportunity to see. Thus Mr. Spalding referred to 'the release of tangential pressure in a newly-opened development face': simple enough words, but immediately recalling the occasion when he saw a body that had been practically decapitated by a slab of razor-edged quartz projected from the face of a development drift; such was the violence of release of tangential pressure in that particular case.

He had always been particularly interested in the effects of such pressure releases in development ends far removed from stopes. One could understand normal subsidence in stopes, but it was astonishing to observe the effects of pressure release hundreds of feet from any large-scale excavations. In those early days the whole emphasis was on providing compressible supports, which were at the same time sufficiently rigid to support the hanging-wall until finally it reposed on the foot-wall. Timber stulls were systematically withdrawn and pack-walls, incorporating stone-packed ' pigstyes ', were built from bottom to top of stopes to provide staggered semi-rigid support of the hanging-wall.

He remembered that the effect of the closure of the hanging-wall in many cases seemed to throw additional load upon pillars purposely left standing with a view to supporting the roof of contiguous stopes, and as a result those pillars became liable to sudden failure. He would like to ask if there was anything in the nature of a fulcrum effect upon such pillars. One had to visualize the stoping plans of the Kolar Gold Field, over a distance of several miles, showing part of that enormous area stoped out and part left in the form of pillars in stopes, shafts, etc. Mostly the discussion referred to pressures produced by load directly overhead, but he wondered whether a large area of collapsed hanging-wall shifted part of its load on to unstoped areas. If that were so, was it not possible that extensive subsidence, even when systematically controlled, did in fact result in additional stesses in ground yet to be stoped ?

He had not noticed reference in the paper to the rupturing of the *foot*-wall. There again he recalled a development level in the Mysore mine in which the floor was lifted with such force that ore trucks were squashed against the roof of the drift. Why was that pressure release manifested in the *foot*-wall of a drift far distant from the stoping areas ?

It seemed to him that rockbursts on a major scale did not become common in the Mysore mine until roughly the 51st level had been reached, although of course there had been minor bursts before that horizon was reached. It happened that he was in Johannesburg when almost the first major rockburst occurred there. As those workings both in Kolar and the Rand could not have been very much deeper than they were say three or four years earlier, why was it that ground suddenly became subject to rockbursts ? He wondered whether a certain degree of weight and resulting pressure had first to be reached, and that then some ' trigger action ' threw the ground into a succession of movements. It certainly seemed in his day in the Kolar mines that several rockbursts occurred in fairly rapid succession (the native superstition was that there were always three), followed by often long periods of quiescence. He wondered what was the experience in, say, the St. John d'el Rey mine.

Dr. F. E. Keep said that he had not read Dr. Phillips's papers and was accordingly unable to decide whether all the conclusions *Mr. Spalding* had shown on p. 2 were his own or were those of

Dr. Phillips. He understood that those on p. 4 were Mr. Spalding's own conclusions. The idea that the action of hydrothermal solutions or ground waters at depth would result in any appreciable effect in ground pressure thousands or hundred of thousands of years later seemed rather far-fetched. Any increase of ground pressure due to such action at or about the time of the influx of hydrothermal waters would surely by now have been relieved by faulting or ground movement. He believed that Mr. Spalding had worked in an area which must at one time have suffered from very great increase in pressure due to that very cause ; he referred to the Shabani asbestos area. The serpentinization of the dunite at great depth there must have been associated with heavy stresses, but he remembered vividly the occurrence of large dry cavities, quite unrelated to the present ground surface, encountered in the early underground work below the main open-cut. It was difficult to reconcile such cavities with the presence of heavy pressure due to serpentinization and active many geological ages later.

The simple hypothesis that the horizontal stress was equal to the vertical stress had been long accepted by students of rock-bursts in deep hard-rock mines : the almost hydrostatic stress present in such mines was evident on all sides. At the end of his paper the author laid down two necessities for the effective control of ground : they were the accepted rules on the Central Witwatersrand mine on which the speaker worked 15 to 20 years ago, and it was interesting to see that they were still considered to be valid. He did not like the recommended faces shown in Figs. 20 and 21 for deep mines, however suitable they might be for American collieries. Where rocks would bend they might be suitable, but where rocks apparently failed by shearing he would consider that each of the points or promontories, located closely to one another as in the figures, would be a death-trap. The longwall faces now popular on the Witwatersrand were not stepped excepting at level, or half level, intervals and then only for the convenience of the installation of stope boxes to aid tramming.

Another point upon which he could not agree with Mr. Spalding was regarding the statement that waste-rock filling was not a suitable medium for control in a mine subject to rockbursts. On the contrary, properly packed waste-rock filling had been found, on the Central Witwatersrand, a more suitable medium than sand-filling. The percentage compressibilities quoted by the author were liable to such variations, owing to so many differing conditions and factors, that little reliance could be placed upon them.

One point which struck him, especially in that section dealing with the bending and closing together of the walls of a stope without excessive fracturing, was that the paper was primarily intended for hard-rock metal miners, but that it contained a lot of theory and practice applicable more to coal mines than to deep metal mines. Some of the practices which appeared to be sound for comparatively shallow *workings* in less brittle rocks seemed to him to be likely to be dangerous if applied in mines subject to rockbursts.

Mr. Spalding stated that one advantage of steel sets was that they could be re-shaped and re-used. He was asking for information and not being critical when he enquired if re-shaped steel sets lost a considerable amount of their original strength as a result of the heat treatment to which they would be subjected on a mine, where expert knowledge and the necessary equipment to treat the steel correctly, in a metallurgical sense, would probably not be available.

The President said the subject dealt with in the paper might appear to some too theoretical and perhaps not sufficiently practical. It would, however, be fair to say that much of the theory had been built up out of bitter experience. There had, over a long series of years on the Kolar Gold Field, been a constant struggle to combat rockbursts, which, at one time or another, had done extensive damage and caused loss of life and injury to underground workers. The gradual evolution of better methods of support and stoping had taken place at the same time as the search for an adequate explanation of those violent movements. While, therefore, the theory presented in the paper did not claim to be a complete explanation of the complex stresses in the rock surrounding mine workings, it did to a large extent fit the facts. What was more important was that, guided by such theory, it had been possible to reopen large areas of ground which had been virtually lost and to work them profitably. It had also enabled mining to continue at ever increasing depths with a diminution in risk rather than an increase. It might be of interest to mention that the deepest point in the Ooregum mine was now 9,376 ft. vertically below surface.

It might be felt that that was a very specialized problem affecting only the few very deep mines. It might be true in a sense, but there must be many mines working deep-seated deposits, and although they might have only reached moderate depths as yet they might ultimately reach the great depths at which such problems arose. Such mines had the opportunity of profiting by the experience of others and might be able to avoid many of the difficulties and hazards which the pioneers had had to endure. For that reason, the theories which were expounded in the paper were deserving of very close attention and should be examined critically by all who might be concerned with deep mining at some future date, as well as the comparatively few who were now confronted by those problems.

The building-up of a theory to meet such a highly-complex problem, if it was to be successful, had to be based upon sound first principles, and therefore any critical analysis of those theories should start from accepted first principles and endeavour to show where the theory might be weak. On the other side of the picture were the practical effects of ground movement in the many different conditions found in mines all over the world. A valuable contribution to the solution of those problems would therefore be accurate observations and records of ground movement over a period of years in any other mining field where ground movement was or might become a problem.

One sympathized with the author in his endeavour to express his theory in terms which were lucid. His use of the terms ‘ ring stress ’ and ‘ stress envelope ’ did not seem to convey quite what he had in mind. The term ‘ ring stress ’ conveyed stress in only two dimensions whereas in the simplest statement of his theory he spoke of a spherical hole. The term ‘ stress envelope ’ could be used in either two dimensions or three dimensions ; what he thought was meant was what might be termed a ‘ spherical stress envelope ’. That theoretical point became more practical when the author referred to a large stoped area above which there were two parts, a central area and an outer ring. There again ‘ ring ’ indicated only two dimensions, whereas the stresses surrounding a stoped area would be roughly hemispherical. In other words, it was a three-dimensional conception. That became clearer where, in describing Fig. 17, the author spoke of a ‘ stress dome ’ covering the whole area and inside it an ‘ annular dome of stress ’. Those terms were three-dimensional. However, in the ‘ Summary ’ the author cleared up the question of terminology when he said in (3) that the shape of the envelope varied from the spherical, through all variations of the ellipsoid, to the cylindrical, according to the shape of the included excavation. If that terminology could have been introduced earlier into the text it would have clarified the conception of the zone of extra stress which surrounded an excavation.

The practical application of the point was that the problem of a single area unaffected by any other workings seldom occurred. More often in practice there were two or more areas whose stress domes merged into one another involving far more complicated results, and unless the problem were viewed in three dimensions some quite erroneous conclusions might be drawn. If the author could develop the application of his theory to a rather more complicated and at the same time more practical case it would be most instructive. For instance, what happened when two stoping areas approached one another ?

Towards the end of the section dealing with control of stoped walls the author stated ‘ the use of a regular system of supports will lessen this unevenness of stress distribution especially if more rigid supports are used in the wide places than in the narrow ’. In advocating the use of supports of varying rigidity in wide and narrow places it would be well to emphasize that that practice should be confined to such places and that for general application, where the width of the opening did not vary greatly, it was highly important that the supports should not vary in their compressibility ; in fact, a mixture of different types of support of varying rigidity could be disastrous in the proper control of ground movements.

Mr. J. A'C. Bergne said that a great deal of work had been put into the paper and it was of fundamental importance. He had not had much experience of deep mining—the deepest he had been was about 5,000 ft.—and he had not studied the effects of excavations in very deep mines. He thought, however, that there must be some threshold loading of the rocks before failure commenced, in spite of the findings of the research workers on concrete. There were caves in existence which went right back to Miocene times : the evidence was on the walls in the shape of paintings of prehistoric man, but no flaking of any kind had taken place. Also, in his own experience, a mine had been stoped over an area of approximately half a mile square with no failing whatsoever. It was a narrow orebody and only pillars were left. One could go into those old workings and there were no signs of flaking anywhere, just a little dust ; yet, when mining was re-commenced for some ore which had been left behind some equilibrium was upset, weight came on, and all the signs of heavy pressure were observed, the pillars being crushed into the softer foot-wall. Again, in the Morenci mine, in Arizona, they were attempting in 1928 and 1929 to use the caving method and while he worked there an adit, capable of taking 50-ton gondolas on standard gauge track, was being driven with the object of mining the immense Clay orebody by caving. When it came to mining they found they could not make the rock cave. Not only did they undercut, but they ran shrinkage stopes around the perimeter of blocks in the attempt to cut the rock away from the mass beside it, and still they could not obtain any ore easily by that method. He understood that that orebody had since been mined by open-cut methods. There was thus something to be said, he felt, for the view that a threshold loading proportional to some rock characteristic must be reached before which no rockbursts or caving would occur. He would be very glad to hear the author's comment on that point.

Mr. Spalding in reply said he apologized for bringing mathematics into the subject. It was not a suitable subject for mathematics, because nothing was fixed or definite—there was doubt expressed as to whether the rocks could be treated as elastic or whether they should be considered granular. A solid compact rock could not be treated as being entirely inelastic. Dr. Phillips's experiments clearly showed that the specimens of rock he treated behaved elastically up to certain limits for immediate loading or unloading. It was only when the time element was brought in that the elastic properties were masked by creep. The interesting point, and it was one on which further enlightenment was required, was that creep seemed to tend to a limit for any given loading, after which the body again appeared to show elastic properties. It would seem then that the rocks examined possessed the quality of elasticity superimposed on what was another property (creep), of which next to nothing was known. All one could do in such an extraordinary situation was to reason and calculate assuming one condition and then another, comparing the results on the different assumptions. From whatever angle approach was made, he found that the indication was that in a solid homogeneous rock mass there was a state of stress which approached the hydrostatic in character.

He said he was very glad that Professor Sillick produced, by a completely different method, the same basic formula for the stress in the rock surrounding a cylindrical excavation as that used in the paper. The angle-of-repose formula suggested by him, however, was completely inapplicable, because it referred only to unconfined material subjected to a limited weight. Loaded with an increasing pressure, the angle of repose of a granular material was constant, but beyond a certain critical point it changed completely. Experience with dumps of fine-ground tailings indicated that if the slime were piled more than about 400 ft. high, the base of the dump burst out. At a stress equal to the weight of 400 ft. of material, therefore, the angle of repose disappeared.

Replying to Mr. Taylor, he agreed that the term 'ring-stress' was a bad one, but having started using it in the first paper he had carried on with it, although he preferred the term 'stress envelope'. Both those terms referred, of course, to a three-dimensional conception, but if it were attempted to represent a complex three-dimensional stress system by diagram, the result would be extremely confusing—nor was it easy to envisage such a system without the aid of diagrams. Luckily, however, most mining excavations were elongated in at least one direction, so that the stress in that direction was less important than in those at right-angles to it. The latter stresses could be simply represented on cross-sections of the excavation. In the case of a horizontal tunnel, the longitudinal component of stress was little affected by the excavation, and one was less interested in it than in the stresses at right-angles to it. In that case the additional stresses appeared on the cross-section as rings surrounding the excavation, which was the reason for the original terminology. For excavations elongated in two directions, such as stopes, the stresses could be viewed on any plane perpendicular to the lode, because the dip and strike components of stress were less important than the normal component. Equi-dimensional excavations such as chambers could be viewed on any convenient plane through the centre.

Fig. 3 had been, perhaps, poorly described. In order to place the various curves in such a manner that comparisons could be made between them, the side of the excavation had been made the zero point on the distance scale, that being the point from which the stress curves started in all cases. Measurements to the right of that point then represented distances into the solid rock; measurements to the left represented distances from the side to the centre of the *excavation*. In order to measure from the centre to any point in the rock, therefore, the two measurements had to be added.

In the discussion an increase in the vertical stress component had been postulated. Except in the cases of geological or geographical abnormalities, with which general theories should not be expected to deal, an increase in the vertical stress was, however, an impossibility. It depended solely on the weight of superincumbent rock, which, in turn, depended only on the depth. If an extra vertical stress were conceivably produced the surface of the ground was a free face, and that extra stress would immediately be relieved by expansion upwards towards it.

In reply to Dr. Keep he said that the eight conclusions given on p. 2 were among the findings of Dr. Phillips, a result of that gentleman's personal research ; the numbered paragraphs on p. 4 were original. Dr. Keep had mentioned that the horizontal stress would be relieved after a period of time, but he, the author, asked how could the stress disappear except by expansion and if there was no room to expand how could the stress be relieved ?

Dr. Keep interjected, By faulting !

Mr. Spalding agreed that that was possible, but stated that they were concerned not with tectonic movements but with residual stresses. With regard to the Shabani mine, he had worked on it himself—the workings were very near surface and there were many fissures in the rock, and in ground of that nature one could not expect any horizontal stress. Near the surface the rocks were less compact and the stresses in them were comparatively minute ; therefore he did not think that in such ground they could look for any of the effects which they had been discussing. Rocks near surface were weak, but stresses were likewise less severe. Whether yielding or complete failure occurred depended entirely on the ratio of stress to strength.

With regard to caves in which historical relics showed that there had been no flaking of the walls for thousands of years, those caves having existed for a long period of geological time prior to that, it was to be expected that any rock movement due purely to stress would have occurred long before man appeared and that equilibrium would by then have been established. Where recent flaking or faults did appear in such cases they were probably due to the decay of the rock owing to chemical action by air or water.

Professor Sillick had said that in the previous paper the author had written ' the stuff came out like dust, which indicated that the material had failed long before the opening was made '. A material subjected to pressure of hydrostatic character could not fail—rigidly confined on all sides it could not disintegrate ; at the worst it became plastic—it was only when the stresses were unbalanced by mining operations that shear and crushing could take place, and so the dust-like material mentioned must have been crushed in the immediate vicinity of a mine opening, during and as a result of the advance of that opening.

With regard to Mr. Gray's remarks on the use of compressible supports in development excavations, the author explained that when a rectangular excavation was first opened up the back was solid. The stress envelope then ringed the excavation closely. After a while the back began to sag a little, a fracture dome began to form, and the stress envelope was expanded. Seeing the sagging roof, or perhaps merely anticipating it, the miner would decide to put in a support, his object being to stop further sag. If he inserted a rigid support immediately, the slightest further movement of the roof would throw stress on to it. The more rigid was the support, the more rapidly it would take up the stress. Having taken the stress, the rigid support would then divide the stress envelope into two smaller ones, as was indicated in the diagram. The span being thus halved, further arching or fracture doming would be prevented. If, on the other hand, a compressible support such as a pigsty were put in, at first it would not take any stress other than that imparted to it by blocking up, and the back would still be largely free to sag. It might come down an inch or so and even then the pigsty, unlike the timber, would not oppose a great deal of resistance. Three inches or more sag might be required before the pigsty was sufficiently compressed to arrest the movement of the rock, during which action the fracture dome and the stress rings would be all the time extending upwards. The pigsty would certainly hold up the loose, but the timber would largely prevent its formation.

Mr. Gray had objected to timber posts—he said they were not rigid—but the speaker's experience was that timber under end pressure was extremely strong and could certainly be regarded as a rigid support. Concrete posts were also much used in such circumstances, the creep which Mr. Gray mentioned being extremely small in comparison with the roof movements. While it might be enough to allow a certain amount of adjustment of differential stress, in a case like that, where one was dealing with measurable movements, the amount of creep was trivial. Structural engineers had after all used concrete for tens of years before they discovered there was such a thing as creep.

In reply to Mr. Wynne, the author said that closure on the Kolar Gold Field was by shear, and for that reason rigid supporting was necessary. At intermediate depths compressible supports had been extensively used, but rockbursts of great violence occasioned by shear frequently occurred, and it was only after rigid supports were introduced that some measure of control of the wall-rocks was gained.

The barren areas which lay between the ore-shoots completely isolated those ore-shoots—a series of rockbursts involving a general collapse in one did not affect the others.

Where a remnant was being mined there would inevitably come a time when the remnant was stressed to the point of failure. In a rock of a type which yielded suddenly rather than gradually the remnant would be in a critical state just before failure. It might be in that state for some considerable period of time, if the relief by creep happened to be balanced by the rate of increase in stress caused by its reduction in size. When in that critical state any external influence, such as a seismic wave, might occasion failure. In that way a rockburst in one isolated ore-shoot might initiate another rockburst in an adjacent one, either simultaneously or soon after. It could not be said to be the cause of it—it was merely the last straw on the back of an already overloaded camel. Apart from occurrences such as that the barren zones between ore-shoots formed effective barriers.

Mr. Wynne mentioned pillars in connection with early experiments in ground control on the Kolar Gold Field. If pillars were to be used for that purpose they must be sufficiently closely spaced to prevent closure of the walls between them, otherwise shear cracks formed, sometimes with violence, or alternatively the faces of the pillars might be crushed by what Mr. Wynne termed the fulcrum effect.

With regard to Mr. Wynne's expressed surprise at movement of the foot-wall, Mr. Spalding said that the stress distribution could be demonstrated as a ' field ', represented by lines of force similar to those in a magnetic field. In Fig. 32 the portion of the stress

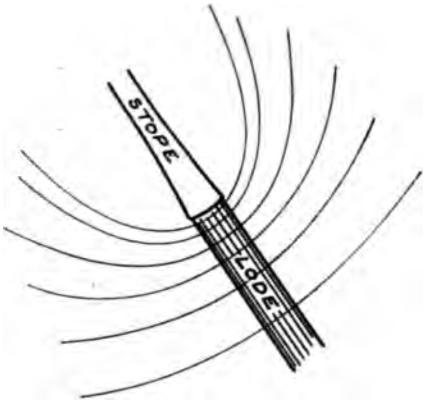


FIG. 32.

envelope near a stope face was shown. Each line represented a definite force, and the closer the lines were together the greater was the stress in the rock. It could be seen from the figure that conditions were substantially the same for both foot- and hanging-walls. If the rock were going to yield or fail completely, it would be as liable to do so in the foot- as in the hanging-wall. In the case of well-developed fracture domes, of course, gravity assisted the movement of semi-loose in the hanging-wall and restrained that in the foot.

He disagreed with Mr. Wynne's remarks with regard to the depth at which rockbursts occurred on the Kolar Gold Field. In opening up some old areas he had found indisputable evidence on the Ooregum mine of a bad rockburst at a depth of about 1,000 ft. The worst rockbursts on the Kolar Gold Field had occurred at between 2,000 and 3,000 ft. in depth—that they had not become still worse at greater depths was due to the fact that at about that time the mine staffs began to understand what was happening and so were able to take corrective measures. As a result from that depth onward rockbursts had not been so widespread nor so devastating as in those particular areas.

Mr. Wynne had mentioned rockbursts occurring in series of several in quick succession. If an ore-shoot were stoped in a haphazard manner, an orderly sequence of stoping being abandoned owing to output demands, a number of pillars and remnants were likely to be formed. As they were gradually reduced in size the loading on the remainder increased until the dangerous critical stage was reached. The failure of any one remnant would then so increase the stress on the remainder as to cause them to fail in succession, until a general collapse of the whole area occurred. Such collapses often took a day or so to become complete, and rock movements frequently continued up to three days after the initial burst. Seismograph records of such a general collapse usually showed first a heavy shock, followed by a prolonged series of minor tremors, interspersed with which would be occasional further heavy shocks, some of which might sometimes be considerably heavier than the original one. It was realization of the importance of a correct sequence of stoping that had led to the great reduction of the incidence of rockbursts in all the fields of deep mining. That factor was even more important than the practice of support.

Replying to Dr. Keep, he said that the method of stepped longwall was regularly carried out on the Rand mines and in other deep metalliferous mines. It was not a practice which was confined to coal mining. He disagreed strongly with Dr. Keep in his plea to separate metalliferous and coal mining practice. There was much that the metal miners could learn from the coal miners, and vice versa; it was, he said, a pity that those two separate professions did not collaborate more.

With regard to the re-shaping of steel sets, he did not know what the effect was on the steel, but the re-shaping of sets was general practice on the Kolar Gold Field and also on the Rand.

Dr. Keep had expressed disagreement with the author when he wrote that waste-rock was not suitable for support—under the heading 'Closure by Shear'. On the Rand, closure, except of the immediate hanging, was by bending, and in such mines waste-rock was a good medium for control. With closure by shear, on the other hand, rigidity of support was necessary, otherwise the shear was liable to get out of hand and might become violent, causing rockbursts. Dr. Keep mentioned that rockbursts were reduced when a certain mine changed over from timber supports to waste-rock filling. He would like to query the implication of that statement—could the connection be proved? About the same time did they not begin to realize the significance of a sequence of stoping, and that the old methods in which every block was reduced to a remnant was the chief cause of rockbursts on the Rand?

Nowadays all deep mines on the Rand were using the longwall or the stepped longwall system of stoping, and by that practice alone they had reduced their rockbursts to a very great degree. On visiting one Rand mine recently he had been informed that by no other method than that of going over to stepped longwall stoping, rockbursts had been reduced from 160 to 40 a month.

With regard to varying the rigidity of supports the suggestion was rather an impractical one, except in certain cases. There were, however, special circumstances in which it was most important—for example, the stoping of a fairly wide lode intersected by a dyke. In a case where, in order to prevent rockbursts, stoping was carried on through the dyke, if the lode were 10 ft. wide, when going through the dyke the stope width would be reduced to the minimum (say 3 ft.), for the sake of economy. If mat packs were used as supports and the total closure were of the order of 2 ft.—that was, from 10 ft. width to 8 ft.—those supports would then be squeezed by 20 per cent of their original width. Through the dyke, however, the supports would be crushed by the 2 ft. of closure from 3 ft. to 1 ft.—that was, by 67 per cent of their original size. Therefore, if those supports were of the same quality as those normally used, they would be squeezed over three times as much and their resistance to the closure, provided they did not disintegrate, would be of the same order. That would stress the dyke rock unduly, and might itself precipitate a burst. Therefore through the dyke a support which was much more compressible than the normal support should be used—such as an open pigsty.

The point which Mr. Taylor raised about mines working deep-seated deposits at depths which were as yet moderate was a most important one. Provided that the ore continued in depth, mines were gradually extended downwards. Frequently, however, their staffs did not realize that they were approaching the state when they could be considered 'deep'. The first evidence they might receive of that might easily be a violent rockburst. There were seldom any warnings of such things—there were usually no indications that one was approaching the critical point at which a sudden and violent rock failure might occur—the first rockburst in a mine came unheralded and unexpected and for that reason was likely to be all the more disastrous.

Rockbursts, then, were not, as Mr. Taylor had pointed out, 'a very specialized problem affecting only the few very deep mines'. They were a potential problem for every mine which was extending in depth. The staffs of all mines which were following down on deposits which continued to depth should therefore keep a critical eye on their sequences of stoping and methods of support, and also on the nature of the rock and its mode of failure. Rockbursts were less likely to occur where stresses could be relieved by rocks which swelled or crumbled under stress, than where no such relief was possible and where the rocks spalled or spat out with violence. For instance, although the reef and the hanging-wall were of quartzite, rockbursts were unknown on the East Rand, where there was a foot-wall of shale which appeared to yield almost plastically. On the other hand, at similar depths bad rockbursts had occurred on the Central Rand where the foot, hanging, and reef were all quartzite.

Dr. Keep said that his previous remarks had referred to 20 years ago, before longwall stoping as such had been introduced on the Central Witwatersrand. On the mine upon which he then worked the incidence of rockbursts had been very high, but the change over from pigsty and timber support to waste-filling had completely changed the picture and rockbursts became fewer and fewer as the waste-supported area increased. It was even then well known that the fewer remnants that were formed the safer would be the mine (one of the chief reasons for the introduction of longwall stoping later), but at that time the mine was not developed sufficiently far ahead to permit the best programme of stoping known to be carried out.

The President expressed thanks to the author for his interesting paper which had produced a stimulating discussion and would, he hoped, produce more in written contributions.

CONTRIBUTED REMARKS

Mr. J. Norman Wynne: Insufficient time was available at the meeting to discuss the manifold considerations raised by this instructive paper and I should like to add a few points to those already contributed.

It seems evident that molecular stresses inherent in rock masses, however produced, rest in natural equilibrium until, in the case of mining, sufficiently extensive excavations upset this natural balance and also provide opportunity for pressure-relief. In the case of earthquakes, stresses which have reached a critical state of disequilibrium in the rock masses forming the earth's crust are able to adjust themselves by relief along the well-defined tectonic lines of pre-existing weakness, and in these zones enormous rock masses are almost continuously and in varying degree 'on the move'.

The author and Professor Sillick illustrated mathematically the theoretical degree of pressure normally to be expected at given points in certain types of localized excavation. The Kolar or Rand miner is, however, concerned with stoping and development areas aggregating a considerable *acreage*, of which shafts, stations, pump chambers and similar openings constitute but a very small proportion. It seems to me that these great areas must be regarded as a whole, and I confess difficulty in imagining formulae practicably applicable, even to *the hanging-wall* alone, in progressively stoped areas of such magnitude. As previously pointed out, the Kolar Gold Field extends over a length of several miles ; and it would be interesting to know what to-day is the ratio of excavated in relation to solid ground along such a length and to a depth of say 6,000 ft. on the Kolar or Witwatersrand field.

During my nine years' underground service in the Mysore mine I gained the impression that certain areas in the Kolar mines were markedly more subject to rockbursts than others, even in the same property—perhaps because I had the doubtful privilege of being in charge of a section at that time particularly susceptible. I would like to ask if this factually is so, and if so whether any explanation is available.

Rightly or wrongly, I also became convinced that some *external* influence 'touched off' the final release of cumulative pressure manifested in the instantaneous and violent rupturing of hanging- or foot-wall that previously had shown no visible evidence of having reached the 'danger point'. Conversely, it was remarkable how not far distant areas stoped many years earlier—unfilled or otherwise unsupported—were entirely unaffected by repeated extremely heavy rockbursts. May I ask if any connection between lunar phases and rockbursts has been observed? It seems at least possible that if ocean masses are so powerfully affected similar 'tidal' influence may be exerted upon rock masses also. And has any seasonal periodicity been noted? In Japan it is commonly believed that earthquakes are more likely to occur in the autumn and at certain phases of the moon. My five years' experience in that country inclined me to think that this is not entirely superstition.

It may be interesting to note that seismic waves themselves do not appear to have any appreciable effect upon mine-workings. I had only just taken over the management of a group of mines in the Idzu Peninsula when the Great Earthquake of 1st September, 1923, occurred, the mines being located less than 80 miles from the epicentre. As I watched the 2,000-ft. high mountain range quivering like stiff blancmange, I appreciated in some degree and for the first time what was meant by 'fluidity' in rock masses. With Kolar experiences in mind, I fully expected the extensive and very ancient stoped areas (entirely unsupported) to be seriously affected. No damage of any kind resulted, while the few men concreting a pump-chamber at a depth of 600 ft. below sea-level on this mine-holiday were blissfully unaware of the convulsion which had laid the whole of Yokohama and large areas of Tokyo in ruins and had caused tidal waves which had swept miles inland. Inquiries showed that none of the other Japanese mines was in any way affected.

Mr. J. A'C. Bergne : In answer to my request for an explanation of some outstanding examples of large voids, all, incidentally, in crystalline rocks, indicating the failure of the formation to cave in, Mr. Spalding revealed the fact that he considered that in these cases an equilibrium had been reached and no more strain was to be expected from the stresses set up by the particular load applied. This statement is in direct contradiction of Dr. D. W. Phillips's conclusion (1) set out on p. 2 of the paper. It would seem, therefore, that there is something to say in support of the thought present in the minds both of Mr. Wynne and myself that there may be some threshold loading connected with rock characteristics which must be applied before creep will occur. Further, Professor Sillick's remarks lead one to suppose that the particular rock characteristics involved may be connected not only with the physical properties of the constituent minerals but also with the shape and orientation of the grains of the rock.

If this were found to be so, it might lead to an explanation of the anomalous fact that the brittle, crystalline, hornblende-schist country-rock of the Kolar Gold Field commenced bursting at the shallow depth of 1,000 ft., while shales and similar rocks have settled quietly at much greater depths, without invoking the pressures to be found in ultra-deep mines.

Mr. H. R. Kerr: Having worked at the Champion Reef mine from 1912–1914 and experienced rockbursts, I think the following remarks may be of interest.

To begin with it must be emphasized that the management took every possible precaution and no expense was spared in attempting to make the mine safe. Despite this I can assure Mr. Norman Wynne that in those days rockbursts, or 'air blasts' as they were then called, were not at all uncommon. The rockbursts occurred without any warning as sudden explosions in hard ground and shattered the heaviest timbers. In some cases large masses of rock were broken down although the bursts were not by any means confined to the proximity of the lode. More or less serious bursts were liable to occur in any part of the mine. I understood Mr. Spalding to say that pillars often proved to be a veritable death-trap: this is absolutely correct. The Mysore mine suffered the most, with Champion Reef a good second.

In those days little was known as to the cause or origin of rockbursts. It was, however, accepted that the quartz veins followed the foliation of the schist beds and that folding occurred as the result of enormous lateral pressure acting on the schist beds subsequent to the formation of the veins. The worst zone for rockbursts was undoubtedly at a depth of around 2,500 ft. In those days the bottom of Champion Reef mine was the 46th station at Carmichael's incline shaft, at a vertical depth of 4,100 ft. or 4,600 on the incline. The portion of Tennant's vertical shaft between the 17th and 20th levels, where it cut the reef, required constant attention and heavy expenditure in replacing and realigning the timbers and guide rails. The travelling roads were examined throughout daily. Below the 17th level the ground was taken out all round the shaft, and timbers 2-3 ft. in diameter were put in so as to form sets (24 ft. by 18 ft. inside measurement) 6 ft. apart outside the shaft. These sets were continued down to the 20th level. An enormous amount of timber was used in the mine and the cost was very heavy, but with the constant rockbursts that occurred no economy was tolerated.

Timbering of levels was by heavy single stulls 7 ft. 6 in. high in the 'head'. Headboards 6 in. thick were used and, in cases where the ground was very heavy or the foot crushed, 6-in. footboards were used as well. As the pressure came on, the stulls were frequently pushed clean through the headboards—like a pencil into butter. This often happened in a remarkably short space of time. The ground round the headboard was sometimes taken out, the stull slung up, headboard removed and a new one put in. This saved the timber—almost doubling its life—and was an economy with large and expensive timbers.

Development was kept well ahead of stoping operations. The roof of a level (or cross-cut) driven in virgin ground—say 500 to 800 ft. below the nearest stope—usually arched itself within a few days or weeks. During the driving the reef was constantly crackling and the quartz would splinter and fly. Pieces of quartz up to the size of an oyster shell would 'fly' with sufficient force to cut a man's arm, clothing, etc.

On one occasion a rockburst occurred in the sink of Carmichael's inclined shaft about 50 ft. below the 46th station, which resulted in serious loss of life. Writing from memory, this was nearly 1,000 ft. below the nearest stope and 200 ft. from the reef in solid country rock. Although the shaft sets were close down to the sink and the hanging-wall carefully lagged, the foot-wall blew out. Personally, I could never really understand this and no satisfactory explanation was forthcoming. One would naturally appreciate a 'fall' or 'burst' from the hanging. However, Mr. Spalding, in replying to this point which was raised in the discussion, stated that he would expect the ring stress to be the same on both foot- and hanging-walls. In my opinion this shows the great value of the paper and proves the author's contention that the more this complex subject of 'ground control' is discussed and ventilated the more chance there is of learning something about it.

Mr. C. A. U. Craven: Mr. David T. Griggs* has for some years carried out controlled experiments on the physical behaviour of rocks in relation to the following characteristics: (1) Confining pressure; (2) shear stress; (3) temperature; (4) time; and (5) the presence of solutions. His work is in the forefront of research in this field, which yields the fundamental empirical data forming the background for considering such practical problems as rockbursts. He sheds considerable light on whether rocks under stress behave as elastic or plastic materials.

In his experiments he puts a cylinder of the rock to be tested under directed stress between the two flat surfaces and a hydro-

*Griggs, David T. (1) Deformation of rocks under high confining pressures. *J. Geol.*, Vol. 44, No. 5, July–August, 1936, pp. 541–77.

(2) Experimental flow of rocks under conditions favoring recrystallization. *Bull. Geol. Soc. Amer.*, Vol. 51, 1940, pp. *1001–1022.*

static or confining stress on the curved surface of the cylinder. In (1) he writes: 'The curves at low confining pressure (1, 2,000 and 4,000 atmospheres) are characteristic of brittle materials which break before entering the region of plastic deformation; in other words, without deviation from pure elasticity. However, the curves at high pressures (8,000 and 10,000 atmospheres) are characteristic of ductile materials which have a "yield point" on entering the region of plastic deformation, and show a "work hardening" or increase of strength before rupture takes place'.

In the same paper he writes: 'If plastic deformation occurs continuously and at a rate which does not approach zero at infinite time, sooner or later the specimen will fail, no matter how slow that plastic flow is. Hence, at the value of strength which holds for infinite time, there must be no finite continuous plastic deformation. Also, no sudden application of high stress is necessary to produce fracture'.

These quotations are given here to show the application of Griggs's work to the problem under consideration, and the reader is referred to the original papers and to another*, which contain much other information which is also of value.

*GORANSON, ROY W. 'Flow' in stressed solids: an interpretation. *Bull. Geol. Soc. Amer.*, Vol. 51, pp. 1023–1034, 1940.

No. 510 MAY, 1949

BULLETIN OF THE INSTITUTION OF MINING AND METALLURGY

Principal Contents :

OFFICIAL NOTICES

ANNUAL REPORT OF THE COUNCIL

REPORT OF DISCUSSION AT MARCH GENERAL MEETING AND AUTHORS' REPLIES TO DISCUSSION ON PAPERS PREVIOUSLY CONTRIBUTED

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No. 510 **MAY, 1949**

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NOTICE OF ANNUAL GENERAL MEETING

The Annual General Meeting of the Institution of Mining and Metallurgy will be held, by kind permission, in the Apartments of the Geological Society, Burlington House, Piccadilly, London, W. 1, on **Thursday, 19th May, 1949,** at 4 o'clock p.m.

AGENDA

Part I at 4 p.m. (*open to Members, Associate Members, and Students only*) :

(1) Minutes of the previous Annual General Meeting.

(2) Appointment of Scrutineers to examine Balloting Lists.

(3) Announcement regarding Benevolent Fund.

(4) Report of Council and Accounts for 1948 (see pp. 1–13 of this *Bulletin*).

(5) Appointment of Auditors.

(6) Votes of thanks.

(Tea and light refreshments will be provided at 4.30 p.m. for members and visitors.)

Part II at 5.15 p.m. (*open to members and visitors*) :

(7) Presentation of Awards :

(*a*) Certificates of Honorary Membership.

(*b*) 'The Consolidated Gold Fields of South Africa, Limited' Premium of Forty Guineas.

(8) Induction of Mr. W. A. C. NEWMAN, O.B.E., A.R.C.S., A.R.S.M., D.I.C., B.SC., as President.

(9) PRESIDENTIAL ADDRESS.

(10) Report of Scrutineers on election of Members of Council.

INSTITUTION NOTES

The Sir Julius Wernher Memorial Lecture

The Council have pleasure in announcing that Dr. C. H. Desch, F.R.S., has accepted an invitation to deliver the second Sir Julius Wernher Memorial Lecture of the Institution. Dr. Desch has taken as his subject 'The effect of impurities on the properties of metals', and the lecture will be delivered at the Royal Institution, 21, Albemarle Street, London, W.1, on Wednesday, 6th July, 1949, at 5 p.m. Admission will be free, without ticket, and visitors will be welcomed.

Additional General Meeting

The Eighth Ordinary General Meeting of the current Session will be held at the Geological Society, Burlington House, W. 1, on Thursday, 16th June, 1949, at 5 p.m. The subject for discussion will be the practical post-graduate training of mining engineers, on which a short paper will be submitted by Professor J. A. S. Ritson, *Member*.

The Refining of Non-ferrous Metals

The Council extend an invitation to members and visitors to attend the Symposium on the Refining of Non-ferrous Metals, to be held on Thursday and Friday, 7th and 8th July, 1949, from 10 a.m. to 5 p.m. on each day, at the Royal Institution of Chartered Surveyors, 14, Great George Street, Westminster, S.W. 1.

A limited number of preprints of the 19 papers prepared for the Symposium will be available for those who wish to attend the meetings, and a volume of Proceedings containing the papers and a report of the discussion will be published after the Symposium has taken place. One set of preprints and one copy of the Proceedings will be available free of charge to members of the Institution who ask for them in advance. The price of preprints to non-members, and to members who require additional copies, will be £1 per set or 1s. 6d. per separate paper.

There will be a corresponding reduction of £1 in the price of the volume of Proceedings (for which the charge will not exceed £2 per copy) in the case of those who have purchased a set of preprints.

The Symposium will most probably be divided into eight sessions, two in the morning and two in the afternoon of each day. A programme will be published as soon as possible.

An order form for tickets of admission, for which there is no charge, and for preprints of the papers has been circulated to members, and further copies are obtainable from the Secretary.

Election of Members of Council for the Session 1950-51

As announced in a notice sent to all members of the Institution last month, the new Bye-laws governing the constitution of the Council and its mode of election will affect nominations for the election of Council for the session 1950–51. Nominations for the election of Ordinary Members of Council [*see* Bye-law 27 (iv)] and Overseas Members of Council [*see* Bye-law 28 (iii)] should be sent to the Secretary of the Institution to reach him not later than 1st November, 1949.

Capper Pass Awards

The generous offer made by Messrs. Capper Pass and Son, Ltd., of £200 per annum for seven years, available for awards to authors of papers on non-ferrous extraction metallurgy published in the *Bulletin* of this Institution and the *Journal* of the Institute of Metals, has been made known to members through these columns. The awards for papers published during 1948 have now been announced by the Adjudicating Committee set up by the two institutions. They are as follows:

For papers on some aspect of Non-ferrous Extraction Metallurgy—

An Award of Fifty Pounds to H. R. Potts for his paper 'Further notes on converter practice at Rio Tinto' (published in the *Bulletin* for March, 1948); and

An Award of Fifty Pounds jointly to R. C. Trumbull, W. Hardiek, and E. G. Lawford for their paper 'Notes on the treatment of pyrites cinders at the plant of the Pyrites Co., Inc., Wilmington, Delaware' (published in the *Bulletin* for December, 1948).

For papers relating to some process or plant used in the Extraction or Fabrication of Non-ferrous Metals—

An Award of Fifty Pounds jointly to C. Blazey, L. Broad, W. S. Gummer, and D. P. Thompson for their paper 'The flow of metal in the tube extrusion' (published in the *Journal* of the Institute of Metals for December, 1948).

Exhibition of Underground Machinery

The Council of Underground Machinery Manufacturers is holding an exhibition at Earls Court, London, from 7th to 16th July from 10 a.m. to 6 p.m. (Thursday, 8 p.m.). The comprehensive collection of underground equipment to be on view includes coal cutters, conveyors, power loaders, electric and compressed-air drilling machines, pneumatic picks, electric equipment and a variety of accessories, and a programme of films to be shown during the exhibition has been arranged for the benefit of visitors. Complimentary tickets of admission may be obtained from the organizers of the exhibition, The Engineering Centre, Ltd., 351, Sauchiehall Street, Glasgow, C. 3, or from the Council of Underground Machinery Manufacturers, 301, Glossop Road, Sheffield, 10.

The afternoon of Monday, 11th July, and the two Saturdays, 9th and 16th July, have been set aside for visits of delegates of the Fourth Empire Mining and Metallurgical Congress and personnel of the National Coal Board, but other visitors are welcome at any other time.

April General Meeting

Some 55 members and visitors attended the Seventh Ordinary General Meeting of the Institution held at Burlington House on 21st April. The first part of the Meeting was devoted to a resumed discussion on Dr. Bruckshaw's paper on 'Geophysics and economic geology'.

Mr. E. G. Lawford then introduced the paper by Mr. H. R. Potts and himself entitled 'Recovery of sulphur from smelter gases by the Orkla process at Rio Tinto', and a full discussion followed.

A report of the Meeting will be published in the June issue of the *Bulletin*.

Candidates for Admission

The Council welcome communications to assist them in deciding whether the qualifications of candidates for admission into the Institution fulfil the requirements of the Bye-laws. The application forms of candidates (other than those for Studentship) will be open for inspection at the office of the Institution for a period of at least two months from the date of the Bulletin *in which their applications are announced.*

The following have applied for transfer since 14th April, 1949 :

To Membership—

William James Bichan (*Regina, Saskatchewan*).
William David Evans (*Nottingham*).
Henry Nisbet Lightbody (*QueQue, Southern Rhodesia*).
Charles Harold William Martyn (*London*).

The following have applied for admission since 14th April, 1949 :

To Membership—

Donald McDonald (*Beckenham, Kent*).

To Associate Membership—

John William Melville Bellasis (*Bulawayo, Southern Rhodesia*).
Lance Gordon Earle (*Bristol, Gloucestershire*).
John Raymond Fletcher (*Selangor, Malaya*).
Daniel Aloysius Harkin (*Dodoma, Tanganyika*).
Kenneth Charles Bramah Morrison (*Tarkwa, Gold Coast Colony*).
George Edward Cunynghame Robertson (*Kisumu, Kenya*).
Struan James Cunynghame Robertson (*Kisumu, Kenya*).
Desborough William Saunders (*Wadebridge, Cornwall*).

To Studentship—

Robert Rennie Bell (*Johannesburg, Transvaal*).
Kenneth Wright Jackson (*Gateshead, Co. Durham*).
Alan Smee (*Barrowford, Lancashire*).
Charles Campbell White (*Dunedin, New Zealand*).

Transfers and Elections

The following were transferred (subject to confirmation in accordance with the conditions of the Bye-laws) on 13th April, 1949 :

To Membership—

Johnstone, Peter Macpherson (*Taiping, Malaya*).

To Associate Membership—

Black, Robert Alastair Lucien (*Johannesburg, Transvaal*).
Bowley, Malcolm Anthony (*Bukuru, Northern Nigeria*).
Collins, Kenneth Michell (*Dunnottar, Transvaal*).
Dell, Christopher Cambridge (*Mufulira, Northern Rhodesia*).
Kelly, Alan (*Mufulira, Northern Rhodesia*).
Parker, Henry Campbell (*Singapore, Malaya*).
Patsalos, Themistoklis Michael Waldemar (*Nicosia, Cyprus*).
Rees, John David (*Holywell, Flintshire*).

The following were elected (subject to confirmation in accordance with the conditions of the Bye-laws) on 13th April, 1949 :

To Membership—

Drouhin, Georges (*Algiers, Algeria*).

To Associate Membership—

Allen, Anthony William (*Pittington, Co. Durham*).
Butcher, Rodney Wesley (*Gwanda, Southern Rhodesia*).
Caddy, John Broominge (*Klian Intan, Malaya*).
Hooper, James Benjamin (*Jos, Northern Nigeria*).
Hooper, William Tregoning Blake (*St. Agnes, Cornwall*).
Howard, Thomas George (*Bromley, Kent*).
White, Donald Campbell (*Champion Reefs, South India*).

To Studentship—

Allen, Kenneth Laidlaw (*Camborne, Cornwall*).
Dunstan, Harry Ivor (*Camborne, Cornwall*).
Hanvey, Alexander Richard (*Belfast, Northern Ireland*).
McAdam, Reginald Culverwell (*Camborne, Cornwall*).
McKinlay, Alexander Cameron MacLean (*Dodoma, Tanganyika Territory*).

Marshall, Colin McKechnie (*Dunedin, New Zealand*).

Oliver, Vincent Hugh Robert (*Mufulira, Northern Rhodesia*).

Slaney, John Samuel (*London*).

Wall, James Brian (*Camborne, Cornwall*).

News of Members

Members, Associate Members and Students are invited to supply the Secretary with personal news for publication under this heading.

Mr. C. ASHBURNER, *Associate Member*, has been appointed mill and cyanide works superintendent at South and Central African Gold Mines, Ltd., Tanganyika Territory.

Professor LEIGH W. BLADON, *Associate Member*, is resigning his Chair at McGill University and has started private practice as a consulting engineer in Montreal.

Mr. C. M. G. BOLTON, *Associate Member*, has taken up the appointment in Gold Coast Colony of geologist to the West African Gold Corporation, Ltd.

Mr. C. B. CURTIS, *Student*, has returned to England from Ariston Gold Mines (1929), Ltd.

Mr. H. C. CURWEN, *Associate Member*, is returning to England from West Africa.

Mr. GORDON S. DUNCAN, *Member*, is at present on a professional visit to the U.S.A.

Mr. R. L. HARVEY, *Associate Member*, expects to leave Australia on a visit to England this year.

Mr. E. H. JAQUES, *Associate Member*, is leaving Nigeria early this month for furlough in England.

Mr. L. M. MCKEE, *Student*, has taken up an appointment in Kampar, Malaya, with Southern Kinta Consolidated, Ltd.

Mr. G. J. MORTIMER, *Student*, was transferred in February from Vogelstruisbult Gold Mining Areas, Ltd., to Simmer and Jack Mines, Ltd., as acting mine captain.

Mr. J. H. POLGLASE, *Associate Member*, is returning on leave to the United Kingdom from Malaya.

Mr. R. C. PULLINGER, *Student*, has been appointed chief surveyor to Frontino Gold Mines, Ltd., Colombia.

Mr. T. A. A. QUARM, *Student*, has left England on his appointment to Cerro de Pasco Copper Corporation, Lima, Peru.

Mr. D. RENOUF, *Member*, is now in Jos, Northern Nigeria.

Mr. W. E. SINCLAIR, *Associate Member*, continues to hold the position of general manager of Cape Blue Mines (Pty.), Ltd., the new name for the Cape Asbestos Co., Ltd.

Mr. JAMES RUSSELL, *Associate Member*, has returned to Sierra Leone from England.

Mr. S. D. SKELCHY, *Student*, is now in British Guiana, having taken up an appointment with Tikwah Gold Developments, Ltd.

Mr. JACK SPALDING, *Member*, is leaving for Dar-es-Salaam to take up the post of mining consultant to the Tanganyika Government.

Mr. T. B. STEVENS, *Member*, has retired from the position of consulting metallurgist to New Consolidated Gold Fields, Ltd.

Mr. C. W. WALKER, *Associate Member*, has left Sierra Leone for leave in England.

Mr. W. H. C. WRIGHT, *Associate Member*, has returned to England from the Gold Coast.

Mr. K. P. WRIGHT, *Student*, took up an appointment in February with New Consolidated Gold Fields, Ltd., at Robinson Deep, Ltd., Johannesburg.

Addresses Wanted

A. Armstrong.
D. S. Broadhurst.
J. A. Cocking.
E. Dickson.
K. A. Knight Hallowes.
R. B. Hicks.
G. H. Pinfield.
A. I. Scott.
A. Sloss.

OBITUARY

Frank Arthur Blakeslee died on 25th December, 1948, in Du Quoin, Illinois, U.S.A., at the age of 84. He was born in Du Quoin, and from 1881 to 1885 learned the machinist's trade in his father's shop in that town.

Mr. Blakeslee was subsequently connected with construction work at a number of metal mines in Nevada, Colorado, Arizona and Utah, and in 1898 he became mill superintendent at the Great Boulder Perseverance gold mine at Kalgoorlie, Western Australia, where he served for six years. He was later connected with the American Smelting and Refining Company, as master mechanic, both in the U.S.A. and Mexico.

After working for a number of years as chief engineer to Ashanti Goldfields in Gold Coast Colony, Mr. Blakeslee came to England on his appointment as superintendent of Fraser and Chalmers Engineering Co., Ltd., Erith, returning to the U.S.A. in 1917. He lived in Vallejo, California, and for some years was employed by the U.S. Navy at the Mare Island Yard on the construction of submarines. He retired in 1934, living at his California home until 1947 when he removed to Marion, Illinois.

Mr. Blakeslee was elected to Membership of the Institution in 1904.

David Cinnamon died in Southern Rhodesia on 12th January, 1949, at the age of 60. Practically the whole of his career was spent in that country. He began as a learner at Penhalonga and Rezende mines, rising during the ten years 1910–1920 to overseer and contractor. In 1922 he was engaged on contracting work on the Trans-Zambesi railway construction. He began his long association with London and Rhodesian Mining and Land Co., Ltd., in 1922, first as shift boss to the Sabiwa mine, and after 18 months took full charge of the Sultan mine. He transferred in 1925 to the position of shift boss of the Cam and Motor mine, and two years later worked on the Fred mine. In November, 1927, he was made mine captain of Rezende mine, and held this position for ten years. He then took up the appointment of manager of the Beatrice Gold Mining Co., Ltd., but owing to ill health left in 1939 to take over King's Daughter claims at Penhalonga, tributed from Rezende Mines, Ltd., on which he worked until his death. From 1939 to 1940 he was consulting engineer for Day Dawn mine.

Mr. Cinnamon was elected to Associateship of the Institution in 1940.

Robert Henry Jeffrey died in Mexico on 31st January, 1949, at the age of 76. In 1895 he took up the appointment of assistant general manager of the Arminius pyrites mine in Louisa Co., Virginia, and also held the position until 1897 of general manager of the Blue Ridge Gold Mines, Ltd. For the next two years he was assistant to the general manager of Pinos Altos mines at Chihuahua, Mexico, and in 1900 was general manager of Milan copper mines in New Hampshire, U.S.A. He was employed by the Globe Minerals Exploration Co. from 1901 to 1902 as general manager of copper mines in Arizona, and was general manager from 1902 to 1904 of Santa Gertrudis Mines, Ltd., at Oaxaca, Mexico.

Mr. Jeffrey spent the rest of his life in Mexico : from 1909 he was general manager of Avino Mines of Mexico, Ltd., and during the period 1914–1916 did consulting work. He joined Mazapil Copper Co., Ltd., in 1916 and was connected with that company until 1939 as manager and managing director. From 1917 to 1930 he held the position of British Vice-Consul at Saltillo, and in 1918 was acting American Consul. For two years from 1925 to 1927 he held a directorship of the Bank of Mexico.

Mr. Jeffrey was elected to Membership of the Institution in 1904.

Cecil Pearse died at the age of 74 on 29th June, 1948, after suffering for over a year with a strained heart. He was surveying in Devonshire and Cornwall from 1893 to 1895, in which year he entered the Camborne School of Mines. In 1897 he went to Malaya as Inspector of Mines, Perak, and in 1901 was appointed Warden of Mines. He resigned from Government service in November, 1902, and went into partnership with Mr. H. F. Rutter under the style of Rutter and Pearse, mining engineers, at Ipoh, Perak. In 1903 the firm was appointed to manage the Tambrin tin mine and later Rahman Hydraulic Tin, Ltd. Owing to amalgamations the title of the firm was changed in 1917 to Aylesbury and Rutter. Mr. Pearse was chairman of the firm from 1920 to 1928, and during the last four years of this period he took over on his own account all the mining activities of the firm. He left Malaya in 1928 and during the next two years was prospecting and reporting in Cornwall for the Siamese Tin

Syndicate, Ltd. He went to Spain in 1934 for Lumbrales Mining and Power Co., Ltd., and in 1936 visited Malaya as consultant to Slim Concessions, Ltd., and Malayan Tin Dredging, Ltd., making another visit in 1938 for the same companies and for Kramat Pulai, Ltd.

Mr. Pearse retired from professional work in 1938. He was elected a Member of the Institution in 1912.

William George Wagner died in London, on 19th March, 1949, at the age of 75. He was educated privately in London and in Germany, and in 1891 was articled for three years to Mr. H. W. Wallis, consulting chemist and assayer, and attended courses of lectures at University College, London. In 1894 he was appointed assistant to Mr. Alfred H. Allen, and two years later to Mr. R. Waterhouse, as chemist and assayer. From 1897 to 1898 he held the appointment of metallurgical chemist and assayer to Sulphides Reduction Syndicate, Ltd., Llanelly, and in 1899 and 1900 was chief metallurgical chemist to British Sulphides Reduction Co., Ltd., at Angoulême, France. For eight years, from 1901 to 1909, Mr. Wagner was connected in technical and managerial capacities with two chemical manufacturing companies, and from 1901 until his death worked privately and with Mr. G. T. Holloway (later Messrs. G. T. Holloway & Co., Ltd.) in London. He first did research work and reporting on metallurgical processes, and in 1910 was made a member of the firm.

Mr. Wagner was elected to Membership of the Institution in 1914 and served for 18 years on the Council, from 1930 to 1948, holding the office of Vice-President for the three Sessions 1933–36.

Dr. S. W. Smith writes: Many of our members and others intimately associated with the metallurgical profession, both here in London and in many parts of the world, will learn with deep regret of the death of Mr. Wagner. Those of us who have served with him on the Council for many years have lost a valued friend whose ever cheerful presence, in spite of a physical handicap, has been an inspiring example to us all.

His wide knowledge and experience, both of men and of affairs, in relation to those matters which are the chief concern of the Institution have always been readily available to those with whom he has shared in the guidance of the Institution's various activities. As Chairman of the Publications Committee during the difficult period from 1942 until 1947 this knowledge and experience has been invaluable in the adjudication of contributions submitted for publication. His acquaintance with collateral matters has often revealed unexpected depths.

Mr. Wagner's professional work as a consultant, extending over 56 years, covered a period when advances in the metallurgical treatment of mined products were undergoing rapid and fundamental changes and expansions in all directions, particularly, perhaps, in regard to the adaptation of flotation methods to processes of concentration. With these developments he had been in close and constant touch throughout, both before and after he became associated with another esteemed member of our Institution, the late Mr. G. T. Holloway, in 1910.

In directing the activities of the firm bearing that honoured name, he may be said to have represented one of the few remaining of those consultants on matters of metallurgical treatment and valuation here in London, from whom guidance and assistance was sought in a particularly intimate and personal capacity.

Former and present Members of Council will remember with gratitude his unceasing interest and help in organizing those social gatherings, under the auspices of the Council Club, which have done so much to afford opportunities of meeting and welcoming members and visitors from overseas.

Walter George Woolston died on 12th February, 1949, at Bognor Regis, Sussex, at the age of 72. He received his technical training at the Penzance School of Mines, Cornwall, and the first 25 years of his career were practically all spent in India. In 1902 he was appointed reduction

officer to the Richmond Gold Mining Syndicate, Ltd., at Pandalur, South India, and in 1904 joined Mysore West and Mysore Wynaad Gold Mining Companies, Ltd., at Oorgaum, where he held the positions of assistant reduction officer, surveyor, and finally chief of the prospecting department. He was in charge of prospecting and mining operations from 1908 to 1911 in the Shimoga and Kadur Districts, and in 1912 was made superintendent of the Kadur Shimoga Gold Fields Syndicate, Ltd., at Tarikere, South India. From 1913 to 1916 he was personal assistant to the general manager of Eastern Development Corporation, Ltd. After six months spent in testing ancient copper workings for the Ooregum Gold Mining Co. of India, Ltd., his services were lent by that company to Messrs. Burn & Co. and during the period 1916–1919 he was engaged as superintendent of copper mining in Sikkim, North India. He remained in North India for the next five years as superintendent of the Sideshur and Kharsawan mines.

After leave in England, Mr. Woolston went to West Africa in 1926, and reported on properties in the Akim District for Effuenta Mines, Ltd., and Fanti Mines, Ltd., and was subsequently assistant to the general manager of Abbontiakoon Mines, Ltd., for a few months. From 1928 to 1929 he acquired bauxite concessions and did prospecting work on the Gold Coast for Fanti Consolidated Investment Co., Ltd., and then reported on properties in Tanganyika for that company, for whom he also took charge of a concessions survey on the Gold Coast from 1930 to 1932 and later did reporting work in 1933 and 1934. He was employed by Anglo-Continental Mines, Ltd., and East Africa Mining Areas, Ltd., as mine manager in Kenya, from 1934 to 1935, and then became resident manager in Kenya for National Mining Corporation, Ltd. For some three and a half years Mr. Woolston was not engaged in the mining profession, but in 1940 he took up the appointment of representative of the General Sandur Mining Co., Ltd., at Bellary, South India, which he held until 1946.

He was elected to Associateship of the Institution in 1912.

The Council regret to announce the death of **Louis Longwood Fewster,** *Associate*, on 4th June, 1947.

BOOK REVIEWS

The Presentation of Technical Information. By Reginald O. Kapp. London: Constable & Co., Ltd., 1948. 147 p. 6s.

Technical Literature. Its Preparation and Presentation. By G. E. Williams. London: George Allen & Unwin, Ltd., 1948. 117 p. 7s. 6d.

The Scientific Paper. How to Prepare it. How to Write it. By Sam F. Trelease. Baltimore: The Williams & Wilkins Co., 1947. 152 p. $2.00.

It is not without significance that these three useful little handbooks should have appeared in quick succession. The need for some guidance to those who aspire to have the results of their work recorded in the Proceedings and Transactions of Scientific and Technical Bodies has long been apparent to those to whom are delegated the tasks of adjudicating on the merits of papers submitted by authors for publication.

In their own particular fields these three writers have already given the benefits of their wide experience in these matters to students and research workers, either by courses of lectures or by occasional addresses. These are now made available to a wider public. Almost every aspect of the preparation and final production of concise and readable records of scientific and technical work, and accompanying illustrations, is covered by one or other of these works. There is, inevitably, much overlapping, but each approaches the problems which confront the inexperienced author, with his own individual outlook. This emphasizes a truth which is made apparent by each—that the communication of knowledge, whether by the written or by the spoken word, is an 'art' acquired only by experience, although it may be given expression in many different ways.

Quite apart from the specific purpose of guiding prospective authors in preparing technical papers for publication, a perusal of these works is likely to be of help to professional men in framing reports on technical matters in a form which may be more readily grasped by those concerned rather with administration than with precise technical matters.

The only danger one foresees in too close a study of these works is lest those with useful knowledge to impart should hesitate or even refrain from doing so, having regard to the 'counsels of perfection' they contain. One has a feeling that even in attempting this short notice of three admirable handbooks one may be open to criticism.

S. W. SMITH.

Introduction to Historical Geology. By RAYMOND C. MOORE. London: McGraw-Hill Publishing Co., Ltd., 1949. ix and 582 p., 6 × 9, illus. 30s. ($5.00).

This lavishly illustrated book, written chiefly for the American reader, is a clear and simply-worded account of the salient features of earth history. After outlining the methods of historical geology, the trends of organic evolution, and the origin of the earth, the main part of the book portrays the changing face of the earth and the fascinating record of life throughout the geological history of North America. The book, which can be recommended to all embarking on the study of geology, avoids excessive technical jargon and explains the cardinal principles of stratigraphy with commendable clarity.

DAVID WILLIAMS.

Reply from Messrs. K. C. LI and C. Y. WANG (authors) to Mr. T. F. Smeaton's Review on **Tungsten**, published in *Bulletin* 506, January, 1949.

The reviewer takes exception to the two following sentences on page 210 and calls them 'bewildering ambiguities':

(*a*) 'Offsetting is due to the growth of large grains in the glowing filament during use, resulting in the development of grain boundaries extending across the full diameter in a plane at right angles to the long axis of the wire'. Dr. Smithells, in his book *Tungsten* (p. 81), wrote about the cause of offsetting thus: 'These crystals are usually somewhat longer than the wire diameter, and are separated by crystal boundaries lying in planes often perpendicular to the wire axis. Such filaments, particularly when heated by alternating current, are liable to offsetting due to flow on the crystal boundaries'.

(*b*) 'To prevent sagging, the structure in the filament should be one of extremely large and long grains'. By 'long grains' is meant of course the longitudinal direction of the grains in respect to the axis of the wire. This sentence is merely a condensation of a statement in *Powder Metallurgy* by John Wulff, pp. 432–3.

The authors cannot see why the reviewer considers that 'these two statements are in conflict'.

The authors purposely tried to treat the metallurgy of tungsten first historically and then critically, hence 'details of patents (many of doubtful value) and illustrations of rather obsolescent reduction apparatus', as the reviewer puts it, were incorporated. The authors concede in part the reviewer's stricture that their knowledge of the manufacture and fabrication of tungsten has been derived 'secondhand', if by secondhand material is meant material based upon the work of other authorities. But what author of a modern technical book could claim that all the material in it was original?

The reviewer assumes that the authors advise the application of a coating of graphite during swaging at 1,500–1,600°C. Of course, at this temperature there is the danger of carbide formation. What the authors did write was (p. 207): 'As tungsten is readily oxidized at these temperatures it is generally protected by a coating of graphite, which is applied in the early stages . . .'. Just before this statement swaging temperatures of 1,350°C., 1,250°C., 1,175°C. and

800–550°C. were listed. There might be a possibility of carbide formation at 1,350°C., but not at the subsequent temperatures.

The reviewer criticizes the authors for having included silica with alumina and thoria as non-volatile additive substances (p. 210—not p. 207 as quoted by the reviewer). In *Powder Metallurgy*, by John Wulff, P. E. Wretblad of Massachusetts Institute of Technology writes (p. 431): 'These [additives] are usually composed of a compound of sodium or potassium mixed with a non-volatile substance such as SiO_2, Al_2O_3 or ThO_2'.

The reviewer attributes to the authors the statement that tungsten rod has never been satisfactorily made from carbon-reduced metal. What the authors did state in the first sentence of the second paragraph on p. 203 was: 'While tungsten powder other than that prepared by hydrogen reduction has never been satisfactorily used in the production of wire, rod, and sheet, many uses for tungsten powder of slightly lower purity and somewhat different physical properties have opened quite a field for carbon or gas reduced powder', and further, 'Consumption of this grade of powder, principally for hard facing metals, welding rod, etc., has, during the past few years, amounted to several hundred tons annually'. The authors cannot see why the reviewer considers these two sentences 'mutually contradictory'.

The reviewer says that 'neither of these [the following] statements is wholly true'. On p. 203, bottom, appeared these statements: 'Any impurities which once get into the oxide will eventually contaminate the resulting powder. Hence only the purest kind of tungsten ore is used for the manufacture of tungsten rods and wire'. These statements are absolutely true. The authors, having been in intimate connection with the Wah Chang tungsten ore-dressing plant at Glen Cove, N.Y., which regularly supplies the dressed purified ore to some of the largest and best-known tungsten manufacturers in America, know what *they* are talking about with regard to the question of objectionable impurities. To illustrate, the following specification was demanded by one of the largest tungsten manufacturers in the United States: WO_3, 68–70 per cent; Sn, 0·50 per cent max.; As, 0·10 per cent max.; P, 0·05 per cent max.; S, 0·10 per cent max.; Cu, 0·05 per cent max.; Sb, 0·10 per cent max.; Mo, 0·002 per cent max.; Bi, 0·10 per cent max.; SiO_2, 0·50 per cent max.; Ca, 0·10 per cent max.

With regard to the methods of analysis of tungsten, each plant of course has its own favourite methods. In fact, the greater part of the chapter on analysis is based on the methods adopted in the Wah Chang ore-dressing plant, and, as such, might or might not be better than some other multifarious methods known.

While the authors appreciate the remark of the reviewer that 'the book contains much that is good' they are rather amazed at the apparent unfairness of his criticism when he makes the flimsily supported blank assertion that the book contains 'too much that is poor'. The authors wish that he had been more circumspect in his criticism of the book, seeing that it treats such diversified subjects as history, geology, ore-dressing, metallurgy, chemistry, analysis, application, substitution and economics.

Mr. T. F. Smeaton replies as follows:

On the effect of large crystals on (1) offsetting and (2) sagging, the authors quote Smithells and Wulff; it would have been advisable to indicate that tungsten may be induced to grow very long crystals, thus combating sag, without simultaneous danger of offsetting, by employing methods which preclude the formation of transverse grain boundaries.

The authors quote from Wulff: 'To prevent sagging the structure in the filament should be one of extremely large and long grains'. This statement is just vaguely true. Equally true is the seemingly contradictory one: To prevent sagging the structure in the filament should be one of extremely small equiaxed grains.

In point of fact, the determining factor governing the highest degree of non-sag quality in a filament is not the grain size at all but the rates of recrystallization and grain growth; if these can be completed in a few seconds, the filament will be non-sag. Further, if recrystallization takes place instantly and subsequent grain growth is prevented altogether by the presence of an effective intergranular barrier material, these filaments will also be non-sag.

The fact that Wulff classes silica as an involatile additive to tungsten does not alter the fact that this substance completely volatilizes during the early stages of the sintering process.

Manufacturers quite naturally prefer to purchase and work ores which contain a high percentage of tungsten oxide and which have low impurity content, but mainly for economic reasons. It is not only possible but has been necessary, during the writer's 29 years' experience, to manufacture tungsten in all forms and in the highest state of purity from ores of very low grade, especially during the last war.

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Wirtschaftlichkeitsfragen beim Hartmetall-Gesteinbohren. (Economic aspects of hard-metal rock drilling.) Herbert Jeschke.—*Z. Erzberg. Metallh.*, Stuttgart, **1**, Sept. 1948, 168–76, tabs. 4 DM.

622.233.3.053 : 669.018.25
Les taillants à carbure dans la foration rotative au charbon: Pt. 2 (*concluded*). Berthon.—*Rev. Industr. Min.*, St. Etienne, No. 541 (Section Mine), Feb. 1949, 51–66, diagrs., tabs.

622.233.3.053 : 669.018.25
Percussion rock drilling with tungsten carbide tipped bits. (Penetration and resistance of rock, change of pressure, tests.) E. J. Wells.—*Chem. Engng. Min. Rev.*, Melb., **41**, Jan. 1949, 135–41, tabs. 1s.

622.233.62
Les taillants à carbure dans la foration rotative au charbon: Pt. 2 (*concluded*). Berthon.—*Rev. Industr. Min.*, St. Etienne, No. 541 (Section Mine), Feb. 1949, 51–66, diagrs., tabs.

622.234 Methods of working

622.234.8 : 622.342.13
Gold dredging in New Zealand.—*Chem. Engng. Min. Rev.*, Melb., **41**, Jan. 1949, 125–8, illus., tabs. 1s.

622.235 Blasting, blasthole drilling

622.235.16 : 622.233.5
Drilling blastholes at the Holden mine with percussion drills and tungsten carbide bits. Elton A. Youngberg.—*Min. Engng.*, N.Y., **1**, March 1949, Sect. Trans., (T.P. 2558 A.) 75–8, diagrs., tabs., biblio. $1.50.

622.25 Shafts, shaft sinking

622.25(682.3)
Shaft design and sinking practice of the Anglo-American Corporation. II. MacConachie and J. B. Mudd.—*J. Chem. Soc. S. Afr.*, J'burg, **49**, Nov. 1948, 165–78, diagrs. 6s.

622.25(682.3–West Rand Cons.)
Brief description of certain features of shafts and shaft lay-out—West Rand Consolidated Mines Limited. W. Scott.—*J. Chem. Soc. S. Afr.*, J'burg, **49**, Nov. 1948, 159–64, diagr. 6s.

622.252/3(682.3-Van Dyk Cons.)
The sinking of No. 5 shaft, Van Dyk Consolidated Mines, Ltd. T. L. Blunt.—*J. Chem. Soc. S. Afr.*, J'burg, **49**, Nov. 1948, 179–92, diagrs., tabs. 6s.

622.271 Mining methods (surface)

622.271.3 : 621.879
Operational statistics of a Marion 5560 power shovel. George B. Clark and George L. Reiter.—*Min. Engng.*, N.Y., **1**, Feb. 1949, Sect. Trans., (T.P. 2531 A.) 44–8, diagrs., tabs., biblio. 75c.

622.272/.275 Mining methods (stoping)

622.273.1
Essai d'un nouveau schéme de foudroyage en grande couche. Charles Huber.—*Rev. Industr. Min.*, St. Etienne, No. 541 (Section Mine), Feb. 1949, 82–5, diagrs.

622.273.2.048
Improvements in roof control by the introduction of pneumatic stowing and power-loading. W. A. Terrell. *Trans. Instn. Min. Engrs.*, Lond., **108**, Pt. 6, March 1949, 213–52, illus., diagrs., tabs. 10s.

622.276 Methods of extraction of petroleum

622.276
Ein neuer Rotary-Spülkopf. (A new swivel joint.) Herwig Rietsch.—*Berg.-Bohr.-u-Erd.*, Vienna, **65**, Pt. 2, 1949, 3–5, illus., diagr.

622.457.3
Etude des canalisatons d'aerage secondaire. (Studies of auxiliary ventilation systems.) Loison.—*Centre Etudes Recher, Charbon. France Note tech.* 49/1, Paris, Jan. 1949, 13 p., diagrs., tabs.

622.6 Haulage, hoisting, transport

622.625.24
Ein neuer Förderwagen. (A new mine car.) Kurt Schnorr.—*Gluckauf*, Essen, **85**, Feb. 12 1949, 122–3, diagrs. 4 DM.

622.625.28
Some problems in the application of locomotive haulage; area reorganisation based on a standard mine car: Pt. 2 (*concluded*), S. J. Temperley.—(*Min. Proc. nat. Ass. Coll. Mgrs.*) *Iron Coal Tr. Rev.*, Lond., **158**, March 11 1949, 513–19, illus., diagrs. 9d.

622.67(435.6)
Schachtfördereinrichtungen und Förderseile im Obergamtsbezirk Dortmund in den Jahren 1936–1947. (Shaft hoisting equipment and winding rope in the Dortmund chief mining district during 1936–1947.) Hermann Herbst. —*Gluckauf*, Essen, **85**, March 26 1949, 215–26, tabs. 4 DM.

622.673.6
Mining ropes. (Brief notes on winding, haulage and guide ropes.)—*Min. J.*, Lond., **232**, April 9 1949, 262, diagr. 8d.

622.69 : 625.92
Aerial ropeways; advantages of ropeway transportation, gravity shoot, single and double jigback, monocable fixed clip, monocable boxhead ropeway.—*Min. World*, Lond., **156**, 1949; March 5, 129; April 2, 181; April 9, 195. 6d. each.

622.7 MINERAL DRESSING

622.7
A lecture on some developments in heavy media, gravity and cyanidation processes. F. T. C. Doughty.—*Trans. Cornish Inst. Min. Engrs.*, Camborne, **3** (N.S.), Sept. 1947–April 1948, 17–21. 10s.

622.7-... Dressing specific minerals

622.7-331.5
Ein Beitrag zur Frage der Erfolgsermittlung bei der Aufbereitung des Siegerländer Spateisensteins. (Note on the problem of determining results in the concentration of Siegerland spathic iron ore.) Hubert Gleichmann and Karl Heinz Zepter.—*Z. Erzberg. Metallh.*, Stuttgart, **1**, Sept. 1948, 178–82, tabs. 4 DM.

622.7-336(437.1)
Über Bergbau und Aufbereitung von Graphit in Südböhmen. (On the mining and treatment of graphite in southern Bohemia.) Werner Gründer.—*Z. Erzberg. Metallh.*, Stuttgart, **1**, July 1948, 109–12, diagrs. 4 DM.

622.7-342.1(682.3 Libanon)
The Libanon reduction plant.—*S. Afr. Min. Engng. J.*, J'burg, **60**, Pt. 1, March 5 1949, 7–13, diagrs. 6d.

622.7-342.1(941 Kalgoorlie)
Milling practice on the Golden Mile. S. G. Salamy.—*Mine & Quarry Engng.*, Lond., **15**, April 1949, 101–6, map, flowsheets. 1s. 6d.

622.7-344.5.084
Méthode d'enrichissement du minerai de plomb applicable notamment aux boues, aux schlamms et aux résidus de lavage. (Procédé dit du Djebel-Hallouf.) Willy Gittens and René Hetzel.—*Ann. Min.*, Paris, **137**, 3, 1948, 29–34.

622.7-367.6
Recent trends in asbestos mining and milling practice. Michael J. Messel.—*Min. Engng.*, N.Y., **1**, Feb. 1949, Sect. Trans. (T.P. 2536 H.), 52–5, biblio. 75c.

622.7.084
Design of slime thickeners.—*Chem. Engng. Min. Rev.*, Melb., **41**, Dec. 1948, 91–5, tabs., biblio. 1s.

622.7.084
Untersuchungen über die günstigste Wichte von Mahltrüben. (Investigations on the most favourable density of mill slimes.) Günther Flügge.—*Z. Erzberg. Metallh.*, Stuttgart, **2**, Feb. 1949, 43–7. 3 DM.

622.73 Comminution

622.73.001.11 : 539.211
Bestämning av specifika ytan pa kross- och malgods enligt gaspermeabilitetsmetoden. (Determination of the specific surface of finely divided materials according to the gas permeability method.) Jonas Svensson.—*Jernkontor. Ann.*, Stockholm, **133**, No. 2, 1949, 33–86, illus., diagrs., tabs., biblio. kr. 2 : 50. (English summary.)

622.736 : 621.926.52
Grundlegende und neuere Erkenntnisse bei der Feinmahlung von mineralischen Rohstoffen in Nasstrommelmühlen und ihre Anwendung in der Erzaufbereitung. (Fundamentals and developments of the fine grinding of crude ore by wet ball milling.) Gotthold Quittkat.—*Z. Erzberg. Metallh.*, Stuttgart, **2**, Jan. 1949, 6–14, illus., diagrs. 3 DM.

622.74 Screening

622.741 : 518.4
Graphische Darstellung und Auswertung von Siebanalysen auf Grund der Rosin-Rammler-Gleichung. (Graphical representation and valuation by screen analysis on the basis of the Rosin-Rammler equation.) Edgar Puffe.—*Z. Erzberg. Metallh.*, Stuttgart, **1**, July 1948, 97–103, diagrs. 4 DM.

622.75/.78 Concentration

622.754.08
Thickening—art or science? E. J. Roberts.—*Min. Engng.*, N.Y., **1**, March 1949, Sect. Trans. (T.P. 2541 B.), 61–4, biblio. $1.50.

622.765 Flotation

622.765 : 549.691.3
The flotation of copper silicate from silica. R. W. Ludt and C. C. De Witt.—*Min. Engng.*, N.Y., **1**, Feb. 1949, Sect. Trans. (T.P. 2535 B.), 49–51, tabs., biblio. 75c.

622.8 HAZARDS, ACCIDENTS

622.81 Gas, dust, explosions

622.811 : 534.647
Seismische Untersuchungen unter Tage. (Seismic investigations underground; in relation to explosions.) Heinrich Baule.—*Gluckauf*, Essen, **85**, Feb. 26 1949, 161–6, diagrs. 4 DM.

622.817
The practical aspects of mine ventilation: Pt. 8, The dust hazard. (Production, determination, suppression.) F. Lebeter.—*Mine & Quarry Engng.*, Lond., **15**, April 1949, 117–22, diagrs., tabs. 1s. 6d.

622.817.9
Etude de produits mouillants. (Study of wetting agents.) Charbonnier.—*Centre Etudes Recher. Charbon. France Note tech.* 49/2, Paris, Jan. 1949, 10 p., diagrs., tabs.

622.817.941 : 622.035.1

Kohlenstaubbekämpfung mit dem Stosstränkverfahren in halbsteiler und steiler Lagerung auf dem Steinkohlenbergwerk Consolidation. (Dust suppression by water infusion in semi-steep and steep seams at the Consolidation colliery.) Hubertus Rolshoven and Alexander Heitmann.—*Gluckauf*, Essen, 85, March 12 1949, 179–86, diagrs., tabs. 4 DM.

622.83 Rock pressure, subsidence

622.831.2 : 622.333

Surveillance des bancs du toit dans l'abatage mécanisé des couches de charbon. A. Winstanley.—*Geol. Mijnbouw*, The Hague, 11, Feb. 1949, 50–76, illus., diagrs., tabs.

622.831.2 : 622.333

Rock pressure in coalmines. H. Labasse.—*Geol. Mijnbouw*, The Hague, 11, Feb. 1949, 31–49, diagrs., biblio.

622.86 Accidents, rescue work

622.861 : 622.662

Safety with shuttle cars. K. K. Kincell.—*Coal Age*, Albany, N.Y., 54, Feb. 1949, 110–12, illus.

66 CHEMICAL TECHNOLOGY

661.31(941–Lake Campion)

Potash from Western Australia; illustrations of the operations at Lake Campion.—*Min. Mag.*, Lond., 80, March 1949, 145–7, illus. 1s. 6d.

662.753(71)

Gasoline survey for summer, 1948. P. B. Seely and F. E. Goodspeed.—*Canad. Bur. Min. Memor. Ser.* 102, Ottawa, Dec. 1948, 8 p., tabs., biblio.

665.54

Integration of operations in a large refinery. K. B. Ross and D. N. McKinlay.—*J. Inst. Petrol.*, Lond., 35, Jan. 1949, 3–27. 7s. 6d.

669 METALLURGY

669.02/.09 Processes and equipment

669.054.8 : 669.25.22

Über die Verhüttung von Neusilberschrott, -abfällen und -spänen. (On the smelting of nickel-silver scrap, waste and turnings. Development of a volatilization process.) Ernst Justus Kohlmeyer.—*Z. Erzberg. Metallh.*, Stuttgart, 2, Jan. 1949, 1–6, tabs. 3 DM.

669.054.8 : 669.7

Technischer Stand und Entwicklungsrichtung der Leichtmetall-Schrottverarbeitung: 1. Leicht-metall-Umschmelzverfahren. (Technical status and trend of development of the consumption of light-metal scrap: Pt. 1. Light-metal remelting process.) Fritz Währer.—*Z. Erzberg. Metallh.*, Stuttgart, 1, August 1948, 129–37, diagrs., tabs., flowsheet. 4 D.M.

669.2/.7 Non-ferrous metallurgy

669.223

Silver extraction. (Lixiviation, cyanidation, base metal ore treatment.) W. H. Dennis.—*Min. J.*, Lond., 232, April 9 1949, 258–61, flowsheets. 8d.

669.537

Arbeiten der Duisburger Kupferhütte über die technische Verwendung von Metallamalgamen. (Operations at the Duisburg copper smelter with reference to the application of metal amalgams; development of the electrolytic zinc process.) Oskar Emert.—*Z. Erzberg. Metallh.*, Stuttgart, 2, Feb. 1949, 47–55, diagrs. 3 DM.

669.539

Penn mine slag dump and mine water, Calaveras county, California. (Experiments to recover zinc from dump and water.) Frank J. Wiebelt and Spangler Ricker.—*Calif. J. Min.*, San Francisco, 45, Jan. 1949, 99–105, map, tabs.

669.7 : 669.054.8

Technischer Stand und Entwicklungsrichtung der Leichtmetall-Schrottverarbeitung: 1. Leicht-metall-Umschmelzverfahren. (Technical status and trend of development of the consumption of light-metal scrap: Pt. 1. Light-metal remelting process.) Fritz Währer.—*Z. Erzberg. Metallh.*, Stuttgart, 1, August 1948, 129–37, diagrs., tabs., flowsheet. 4 DM.

669.717 : 622.002.5

Aluminium alloys in mining equipment. J. C. Bailey.—*Trans. Instn. Min. Engrs.*, Lond., 108, Pt. 7, April 1949, 256–79, illus., tabs., biblio. 10s.

669.725.3

Mineral chlorination studies: 3. The chlorination of Australian beryl. F. K. McTaggart.—*J. Sci. Industr. Res.*, Melb., 20, Nov. 1947, 564–84, diagrs., tabs., biblio.

669.753

Die Antimonanlage der Herzog Juliushütte. (The antimony plant of the Herzog Julius metallurgical works.) Otto Hermann Schütze.—*Z. Erzberg. Metallh.*, Stuttgart, 1, July 1948, 103–9, illus., tab., flowsheet. 4 DM.

669.785

Electrodeposition and electrowinning of germanium. Colin G. Fink and Vasant M. Dokras.—*J. Electrochem. Soc.*, Baltimore, Md., 95, Feb. 1949, 80–97, tabs., biblio. $1.

669.791.5

Arbeiten der Duisburger Kupferhütte über die technische Verwendung von Metallamalgamen. (Operations at the Duisburg copper smelter with reference to the application of metal amalgams; development of the electrolytic zinc process.) Oskar Emert.—*Z. Erzberg. Metallh.*, Stuttgart, 2, Feb. 1949, 47–55, diagrs. 3 DM.

669.872

Gegenwärtiger Stand der Gewinnung von Indium aus Rammelsberger Erz. (Current practice in the extraction of indium from the Rammelsberg ore.) Reinhard Kleinert.—*Z. Erzberg. Metallh.*, Stuttgart, 2, Jan. 1949, 14–18, tabs., flowsheets. 3 DM.

711 TOWN AND COUNTRY PLANNING

711 : 622

The Town and Country Planning Act, 1947: Development charges. (Section 60 deals with minerals.) J. D. Trustram Eve.—*J. Instn. Chart. Surv.*, Lond., (*Trans.*, 81, 1948–49, 103–22.), 28, April 1949.

THE INSTITUTION OF MINING AND METALLURGY

Founded 1892. *Incorporated by Royal Charter* 1915.

ANNUAL REPORT OF THE COUNCIL

The Council of the Institution of Mining and Metallurgy have pleasure in submitting their Report on the affairs of the Institution for the Session 1948–49, and the Statement of Accounts for the year ended 31st December, 1948. As in previous years, the statistics given in the Report refer to the calendar year 1948.

Revision of the Bye-laws

The revised Bye-laws, to which reference was made in the previous Report, were adopted at a Special General Meeting of Members and Associates held on 16th December, 1948, and were allowed by the Lords of His Majesty's Privy Council on 20th February, 1949, on which day they came into force.

The revision marks an important stage in the history of the Institution. The Council believe that the foundation has been laid for the future growth of the Institution in numbers and prestige. The rate of that growth will depend upon the efforts of all members, and the Council urge members to study the new Bye-laws and to do all they can to encourage qualified candidates to apply for admission.

Roll of the Institution

During 1948 there were 8 admissions or readmissions to Membership, 35 to Associateship and 85 to Studentship. In addition, three Members were transferred to Honorary Membership, 23 Associates were transferred to Membership and 29 Students to Associateship. The numbers of members in the various classes were as follows:

	At 31st December 1948	At 31st December 1947
Honorary Members	12	10
Members	622	630
Associates	1,152	1,149
Students	406	370
	2,192	2,159

The names of 5 Members, 15 Associates and 12 Students have been removed from the Register owing to non-payment of subscription, and the resignations of 11 Members, 13 Associates and 7 Students were received and accepted. The names of 12 Students were removed from the Register owing to their failure to obtain transfer to Associateship within the period of eight years laid down in the Bye-laws. The name of one Associate was removed from the Register under Section II, clause 8, of the Bye-laws.

The Council deeply regret to record the death of the following members, and wish to express sympathy with their relatives:

Honorary Member (1): William Cullen.

Members (20): William Thomas Anderson, H. Foster Bain, James Chapman Brown, Henry James Daggar, Bernard John Hastings, Horace Cecil Benjamin Hickling, Arthur Hibbert, Ernest Hibbert, Guy Carleton Jones, Raymond Molloy Kateley, Sibley Byron McCluskey, Eric Ogilvy Macpherson, Godfrey Ewart Morgans, Philip Francis Paterson, Arthur Michael Robinson, Arthur William Ross, Gerhard August Stockfeld, George Arthur Stonier, Gilbert Andrew Syme, John Charles Tonkin.

Associates (9): Thomas Andrew Clarke, Ernest Vaughan Dabb, Nathaniel Gordon Farquhar, William Clarke Grummitt, Oscar Sydney Marks, John Edwin Ogilvie, Hugh McIlfroy Paterson, James Francis Smith, Leonard Stephen Wilson.

Student (1): R. Krishnaswamy.

This list represents a further heavy loss to the Institution and the mining and metallurgical professions. As will be seen the list includes Dr. William Cullen, *Past-President*, Mr. Carleton Jones, a Gold Medallist of the Institution, and Mr. Ernest Hibbert and his brother, Mr. Arthur Hibbert, both of whom served for several years as Members of Council.

Honours and Distinctions

Reports of the award of honours to the following Members have been received during the year, and the Council offer their congratulations to the recipients:

Order of the Bath—

K.C.B.—

Colonel Paul J. G. Gueterbock, C.B., D.S.O., M.C., T.D., D.L., J.P. (*Member of Council*).

Order of the British Empire—

C.B.E. (*Civil Division*)—

B. E. Frayling (*Member*).

B. L. Gardiner (*Member*).

Prof. W. R. Jones (*Past-President*).

Institution Awards

The Right Hon. Viscount Nuffield, G.B.E., D.C.L., F.R.S., has been elected an Honorary Member of the Institution, in recognition of his interest in and benefactions to metallurgical education in the British Empire.

Dr. Sydney William Smith, C.B.E., A.R.S.M., D.Sc. (*London*), Hon. D.Sc. (*Witwatersrand*), M.I.M.M., *Past-President*, and Mr. John Allen Howe, O.B.E., B.Sc., M.I.M.M., *Past-President*, have been elected Honorary Members in recognition of their services to the Institution.

'The Consolidated Gold Fields of South Africa, Limited' Premium of Forty Guineas has been awarded to Mr. Frederick

Harry Fitch, A.R.C.S., B.Sc., A.M.I.M.M., for his paper on 'The Tin Mines of Pahang Consolidated Co., Ltd.' (*Transactions*, Vol. lxvii).

No award of the Gold Medal of the Institution or of 'The Consolidated Gold Fields of South Africa, Limited' Gold Medal has been made.

Sir Julius Wernher Memorial Lecture

The Council have pleasure in announcing that Dr. C. H. Desch, F.R.S., has accepted an invitation to deliver the second Sir Julius Wernher Memorial Lecture. Dr. Desch will take as his subject 'The effect of impurities on the properties of metals', and his Lecture will be delivered at the Royal Institution, London, on 6th July, 1949, the day preceding the opening of the Symposium on the Refining of Non-ferrous Metals to which reference is made later in this report.

Officers and Council for Session 1949–50

As announced in the *Bulletins* for November and December, 1948, Mr. W. A. C. Newman has been elected President of the Institution in succession to Mr. S. E. Taylor, and Mr. Robert Annan has been re-elected Honorary Treasurer. Messrs. A. L. Butler, T. Eastwood, Donald Gill, Vernon Harbord, L. C. Hill and Sir Arthur Smout have been elected or re-elected Vice-Presidents. These elections were made under the provisions of the Bye-laws then in force; the revised Bye-laws since allowed by the Privy Council will govern the election of Officers and Council for the 1950–51 and subsequent Sessions.

The Council have regretfully accepted the resignation of Brigadier R. S. G. Stokes from the office of Vice-President, owing to his having taken up residence in South Africa, and they wish to record their sincere thanks for his valuable services to the Institution as Member of Council and Vice-President.

During 1948, 53 Council and Committee Meetings were held, and Members of Council attended many meetings of other bodies as representatives of the Institution.

Fourth Empire Mining and Metallurgical Congress

The Institution has co-operated with the Organizing Committee of the Fourth Empire Mining and Metallurgical Congress, to be held in London and Oxford from 9th to 17th July, 1949. At the request of that Committee, the Council undertook the responsibility of obtaining papers for three of the technical sessions.

The President-Elect, Mr. Newman, has been appointed a Vice-President of the Congress, and the Institution will be officially represented by three delegates, Mr. G. Keith Allen, Mr. Vernon Harbord, and Sir Edmund Teale.

Symposium on Metal Refining

As members have already been informed, arrangements have been made for a two-day Symposium on the Refining of Non-ferrous Metals to be held in London on 7th and 8th July, 1949. The Council wish to record their thanks to the Committee under the Chairmanship of Sir Arthur Smout, which was responsible for all the arrangements, to the authors of the 19 papers which will form the Symposium, and to the Royal Institution of Chartered Surveyors for the use of their Lecture Hall. The Council trust that members interested in this subject will make a special effort to attend the Symposium and to contribute to the discussion on the excellent papers that will be submitted.

Meetings

Since the last Annual General Meeting, eight Ordinary General Meetings and one Special General Meeting have been held in the Apartments of the Geological Society at Burlington House, Piccadilly, and the Council are grateful to the Society for their continued hospitality.

Visit to Corby

By courtesy of Messrs. Stewarts and Lloyds, Ltd., a party of members visited that Company's opencast iron-ore mines and steelworks at Corby, Northants., in September, 1948. Lunch and tea were generously provided by the Company, and the party was greatly indebted to Mr. J. R. Menzies-Wilson and Mr. J. Mitchell. Managing Directors, Mr. R. B. Beilby, Consulting Engineer, and Mr. F. G. Weller, Mines Manager, for the admirable arrangements for the visit.

Capper Pass Awards

Reference was made in the last Report to the generous action of Messrs. Capper Pass and Son, Ltd., in presenting £200 per annum for seven years, to be available for awards to the authors of papers on non-ferrous extraction metallurgy contributed to the *Transactions* of the Institution or to the *Journal* of the Institute of Metals. The Adjudicating Committee set up by the two institutions have announced the following awards for papers published in 1948 :

For papers on some aspect of Non-Ferrous Extraction Metallurgy :

An Award of Fifty Pounds to H. R. Potts for his paper 'Further Notes on Converter Practice at Rio Tinto' (*Bulletin* I.M.M., March, 1948).

An Award of Fifty Pounds jointly to R. C. Trumbull, W. Hardiek and E. G. Lawford for their paper 'Notes on the Treatment of Pyrites Cinders at the Plant of the Pyrites Co., Inc., Wilmington, Delaware' (*Bulletin* I.M.M., December, 1948).

For papers relating to some process or plant used in the Extraction or Fabrication of Non-Ferrous Metals:

An Award of Fifty Pounds jointly to C. Blazey, L. Broad, W. S. Gummer and D. P. Thompson for their paper 'The Flow of Metal in Tube Extrusion' (*Journal* Inst. Metals, December, 1948).

Joint Library of the Institution and the Institution of Mining Engineers

Members' appreciation of the Joint Library is shown by a further large increase in the use made of it in 1948. Compared with the previous year, the number of books borrowed rose from 1,002 to 1,461, and the number of enquiries dealt with from 1,260 to 1,820. The number of books borrowed by post was 957, and of the enquiries 595 were by telephone and 460 by post. The total number of books, pamphlets and maps added to the Library in 1948 was 1,252.

The Library subject index is now classified under the Universal Decimal Classification, and the List of Recent Articles appearing in the *Bulletin* has been classified in the same way since March. 1949. In the same month, a pamphlet giving details of Library services, and an outline of the newly adopted U.D.C., was circulated to members.

Publications

The monthly *Bulletin* has been published regularly throughout the Session, and Volume lv of the *Transactions* was issued in 1949. The subsequent volume of *Transactions* has been sent to the printers. and a combined index to *Transactions* Volumes xli to lv, inclusive, will be issued later in 1949.

The Proceedings of the Joint Conference on Silicosis, Pneumokoniosis and Dust Suppression in Mines were published in 1948, and copies are still available to members of the Institution at the reduced price of 10s. per copy.

New editions of the List of Members, and of the Royal Charter and Bye-laws of the Institution, were issued to all members with *Bulletins* Nos. 508 and 509 respectively.

The list of papers published since the last Report was issued is as follows:

'Impressions of Mining Practice in North America', by Jack Spalding, *Member*. (*Bulletin* No. 499).

'The Directorate of Colonial Geological Surveys'. (An Address by Dr. F. Dixey, *Member*). (*Bulletin* No. 499).

'Notes on the Mount Kasi Mine, Fiji', by P. M. Johnstone, *Associate*. (*Bulletin* No. 501).

'Notes on a Small Working in Southern Rhodesia', by A. M. Bensusan, *Associate*. (*Bulletin* No. 502).

'A Simple Flotation Cell', by E. J. Pryor, *Member*, and K.-B. Liou, *Student*. (*Bulletin* No. 502).

'The Sinking of No. 5 Shaft, Van Dyk Consolidated Mines, Ltd.', by T. L. Blunt. (*Bulletin* No. 503).

'Diamond-drill Blast-hole Practice at the Roan Antelope Copper Mine', by H. F.-C. Nevill and W. K. Burgess, *Associates*. (*Bulletin* No. 504).

'Notes on the Treatment of Pyrites Cinders at the Plant of the Pyrites Co., Inc., Wilmington, Delaware', by R. C. Trumbull, W. Hardiek, and E. G. Lawford, *Member*. (*Bulletin* No. 505).

'A Note on "Steel" Galena', by G. A. Schnellmann, *Associate*. (*Bulletin* No. 505).

'Mining and Milling Antimony Ore at Consolidated Murchison Goldfields, Transvaal', by Ralph Symons, *Member*. (*Bulletin* No. 506).

'Ground Control—Theory and Practice', by Jack Spalding, *Member*. (*Bulletin* No. 507).

'Geophysics and Economic Geology', by J. M. Bruckshaw. (*Bulletin* No. 508).

'An Improved Form of Konimeter', by J. P. Rees, *Associate*, and S. R. Rabson. (*Bulletin* No. 508).

'Recovery of Sulphur from Smelter Gases by the Orkla Process at Rio Tinto', by H. R. Potts and E. G. Lawford, *Members*. (*Bulletin* No. 509).

Education

During 1948 the Council were asked by Government for their views on training in mineral dressing in Great Britain. A memorandum was submitted, and as a result of subsequent discussions it is hoped that provision will shortly be made for the granting of a degree in mineral dressing.

Three Mond Nickel Fellowships were awarded in 1948 by the Joint Committee of the five metallurgical institutions. The Nuffield Foundation awarded four Travelling Fellowships, five Post-Graduate Scholarships and nine Vacation Scholarships. The Council wish to record their appreciation of the great encouragement afforded to metallurgical education and practice by these Fellowships and Scholarships.

The scheme for awarding National Certificates in Metallurgy has given further encouraging results. Final examinations for Ordinary and Higher Certificates were held at 18 Technical Colleges in 1948, 74 candidates gaining the Ordinary Certificate and 22 the Higher Certificate.

The Report of the Joint Committee on Metallurgical Education for 1947–48 was published in the *Bulletin* of the Institution for January, 1949. An important part of the Committee's work was the preparation of a series of recommendations on the qualifications which should be required of entrants to University courses in metallurgy. The recommendations, which were reprinted in the *Bulletin* for July, 1948, merit the serious consideration of all concerned with technical education of University status. Many of these recommendations are applicable to education in subjects other than metallurgy.

Activities of the Institution

The Council are pleased to report that in response to the appeal made by the President in his Address of May, 1948, several members have put forward suggestions for improving and extending the activities of the Institution. These suggestions have all received serious consideration, and some have already been put into effect. The Council wish to endorse the President's appeal; at all times they welcome any suggestions members can make for increasing the usefulness of the Institution.

Accounts

The Statement of Receipts and Expenditure and Balance Sheet for the year ended 31st December, 1948, are submitted herewith. It will be seen that there was a deficit for the year of £930 12s. 5d.

The cost of maintaining the ordinary services of the Institution still substantially exceeds the normal revenue. The income from subscriptions is approximately the same as it was ten years ago, and the income from investments some £250 per annum less in spite of an increase in the total amount invested.

It is well known that the cost of goods and services has greatly increased in the same period, and there are no signs of any improvement in this state of affairs. Unless the income of the Institution is augmented by an increase in membership it will face a serious financial problem.

The Reserve for Contingencies and Post-War Expenditure built up during the war years will continue to be used for extending the special activities of the Institution, and the Council are most anxious not to be forced to curtail this work.

General

Representation on Other Bodies and at Official Functions.—Shortly before his death, Dr. W. Cullen resigned from the Metallurgical Advisory Committee of King's College, Newcastle-upon-Tyne, and Mr. Stanley Robson was appointed to succeed him. Mr. W. R. Degenhardt has succeeded Mr. H. M. Morgans on Technical Committee CRE/3 (Colliery Wire Ropes) of the British Standards Institution. Mr. Donald Gill, *Vice-President*, and the Secretary represented the Institution at the Centenary Celebrations of the Société des Ingénieurs Civils de France in 1948, and Mr. R. M. Harland attended the Installation of the Chancellor of Sheffield University in a similar capacity. Members of Council have continued to represent the Institution on other bodies.

Taxation.—In 1948 a Departmental Committee on Taxation and Overseas Minerals was appointed by the Government to consider whether any handicap was placed on British mining concerns by the lack of provision in the taxation laws of allowances for capital expenditure in acquiring overseas mineral deposits. A memorandum to the Committee was drawn up by the British Overseas Mining Association, and after careful consideration the Council endorsed this memorandum.

Town and Country Planning Act, 1947.—A memorandum on the effect of this Act upon the British mining industry (excluding coal-mining) was submitted by the Council to the Minister of Town and Country Planning on 14th October, 1948. The memorandum was published in the *Bulletin* for November, 1948. The Institution has subsequently been consulted by the Central Land Board, set up by the Act, on matters concerning the valuation of mineral lands.

Appointments (Information) Register.—The Appointments (Information) Committee of the Council maintains a register of disengaged members. The Committee do not act in any sense as an ' employment agency ' and all that they can do, within the limits of their constitution, is to put members who are registered as disengaged into communication with employers who have posts to offer and have communicated their requirements to the Institution. No charge is made in connection with the register. During 1948 a number of vacancies was brought to the notice of the Institution, and 51 members whose names were on the register obtained appointments.

Annual Dinner.—It was unfortunately not possible to arrange for an Annual Dinner in 1948. Arrangements have been made for the next Annual Dinner to take place at the Savoy Hotel on 5th May, 1949.

Staff.—Mr. Henry Rose, Chief Clerk, retired on 3rd July, 1948, after a period of service with the Institution of over 51 years.

In conclusion, the Council record their appreciation of the loyal services of the Secretary and Staff of the Institution during the past year.

By order of the Council,

S. E. TAYLOR, *President.*

W. J. FELTON, *Secretary.*

13th April, 1949.

[redacted]

Dr. **Balance Sheet,**

		£	s.	d.	£	s.	d.
To Sir Julius Wernher Memorial Fund					10,147	0	0
,, Endowment Fund					47,654	17	4
,, Post Graduate Grants Fund					609	10	2
,, Life Compositions Account					3,979	10	0
,, Sundry Creditors					999	18	4
,, Reserve for Contingencies and Post-War Expenditure as at 31st December, 1947 ...					6,700	0	0
,, Staff Superannuation Fund as at 31st December, 1947		10,609	13	9			
Add Transfer from Receipts and Expenditure Account		495	3	5			
Income from Investments and Bank Interest		196	8	7			
Profit on repayment of Investment and matured Policies		144	3	7			
		11,445	9	4			
Less Matured Policies paid over	£647 9 2						
Gratuity	250 0 0						
		897	9	2			
		10,548	0	2			
Add Investment amortization reserve ...		18	11	6			
					10,566	11	8
,, 'A. G. Charleton Prize' Trust Fund, as at 31st December, 1947		723	19	9			
Add Income for 1948		24	4	0			
		748	3	9			
Less Prize awarded for 1947		24	4	0			
					723	19	9
,, Subscriptions, etc., in Suspense					104	15	5
,, Reserve for amortization of Investments ...					398	5	6
,, Accumulated Fund: Balance as at 31st December, 1947		39,756	5	1			
Add Profit on sale of 2½% National War Bonds, 1951/53		21	11	6			
		39,777	16	7			
Less Balance of Receipts and Expenditure Account for the year ended 31st December, 1948	£930 12 5						
Transfer to Life Compositions Account...	252 0 0						
		1,182	12	5			
					38,595	4	2
					£120,479	12	4

R. ANNAN, *Hon. Treasurer.*

W. J. FELTON, *Secretary.*

We have examined the foregoing Balance Sheet with the Books, Vouchers and of the *Transactions* in hand, and subscriptions in arrear, are not included

London, 27th April, 1949.

	£	s.	d.	£	s.	d.
y Sir Julius Wernher Memorial Fund :						
Investments, at cost :						
£481 19s. 8d. British Transport 3%, 1978/88	388	0	3			
£2,250 London County 3% Consolidated Stock	1,715	12	6			
£5,253 2½% Consolidated Stock	3,943	7	3			
£4,100 3% Savings Bonds, 1955/65 ...	4,100	0	0			
				10,147	0	0
Endowment Fund :						
Investments, at cost :						
£8,461 2½% Consolidated Stock	7,636	6	10			
£20,036 2½% National War Bonds, 1949/51	20,675	7	3			
£1,500 3½% War Stock	1,608	19	0			
£16,700 4% Consolidated Stock	17,734	4	3			
				47,654	17	4
Investments, at cost :						
£8,532 9s. 2½% Consolidated Stock ...	7,497	2	7			
£759 2½% National War Bonds, 1949/51 ...	783	8	5			
£3,000 3% Savings Bonds, 1960/70 ...	3,000	0	0			
£23,980 3½% War Stock	23,642	8	8			
£15,985 3½% Conversion Stock	13,038	10	5			
£800 4% Consolidated Stock	826	2	0			
				48,787	12	1
Staff Superannuation Fund (*commenced* 1924) :						
Investments, at cost :						
£300 3½% War Stock	305	16	3			
£75 3% Defence Bonds	75	0	0			
£450 3% Savings Bonds, 1960/70	450	0	0			
£4,350 2½% Consolidated Stock	4,100	16	0			
£450 2½% Defence Bonds (Conversion) ...	450	0	0			
£1,455 2½% National War Bonds, 1949/51...	1,501	8	8			
Insurance Policies :						
Premiums on fully paid Policies £1,110 6 7						
Other premiums £1,647 10 3						
Less Staff Contributions ... 295 0 0						
1,352 10 3						
	2,462	16	10			
Post Office Savings Bank	1,005	6	7			
Cash at Bank	215	7	4			
				10,566	11	8
'A. G. Charleton Prize' Trust Fund :						
£691 10s. 7d. 3½% War Stock, at cost ...	699	15	9			
Cash at Bank	24	4	0			
				723	19	9
Furniture and Library as at 31st Dec., 1947 ...	775	6	5			
Additions during 1948	376	0	11			
	1,151	7	4			
Less Depreciation at 12½%	143	18	4			
				1,007	9	0
Sundry Debtors				168	0	0
Cash at Bank	1,248	6	6			
Less on account of—						
'A. G. Charleton Prize' Trust Fund ...	24	4	0			
				1,224	2	6
Cash in hands of Secretary				200	0	0
				£120,479	12	4

nkers' Certificates, and certify it to be in accordance therewith. The value
ong the assets.

WOODTHORPE, BEVAN & CO.,
CHARTERED ACCOUNTANTS, } Auditors.

Dr. *Receipts and Expenditure for*

1947. £	s.	d.		EXPENDITURE.	£	s.	d.	£	s.	d.
			To	Cost of Printing Publications: *Transactions*, *Bulletin*, advance copies of Papers, reports of Discussions and on account of Volume 56, *less* receipts for the Advertisement Section of the *Bulletin*				2,309	11	9
2,119	4	1								
477	6	5	,,	Printing and Stationery				669	6	11
588	1	11	,,	Expenses of Meetings				296	18	0
804	1	7	,,	Conference on Silicosis				56	4	6
50	4	9	,,	Insurance				75	2	6
1,593	7	2	,,	Rent, Heating, Lighting, Cleaning, etc.				1,641	3	2
42	2	10	,,	Telephone				41	19	0
89	5	0	,,	Audit Fee				89	5	0
4,539	5	0	,,	Salaries	4,418	19	6			
1,000	0	0	,,	Retiring Allowances	1,187	10	0			
								5,606	9	6
479	12	11	,,	Staff Superannuation Fund ...				495	3	5
352	5	9	,,	Postage and Receipt Stamps ...				359	7	4
150	8	9	,,	Sundry Accounts				423	8	1
–	–	–	,,	Legal Expenses				66	19	6
			,,	Subscriptions:						
52	10	0		Ross Institute of Tropical Hygiene				52	10	0
–	–	–		British Standards Institution...				50	0	0
50	0	0		*International Geological Congress*				–	–	–
26	5	0		Parliamentary and Scientific Committee				15	15	0
25	0	0		Mond Nickel Fellowships ...				25	0	0
			,,	National Certificates in Metallurgy:						
150	0	0		Proportion of Expenses ...	125	0	0			
100	0	0		Contribution to Prize Fund	100	0	0			
								225	0	0
			,,	Joint Committee on Metallurgical Education:						
50	0	0		Proportion of Expenses ...				50	0	0
22	15	5	,,	Repairs and Renewals				37	18	3
42	0	0	,,	*'Consolidated Gold Fields of South Africa Ltd.', Premium for 1946*				–	–	–
17	4	8	,,	Interest on Post Graduate Grants Fund				17	15	0
			,,	Library:						
15	1	8		Newspapers, etc.	14	11	11			
75	3	1		Depreciation	94	4	0			
								108	15	11
35	12	1	,,	Depreciation of Furniture ...				49	14	4
£12,946	18	1						£12,763	7	2

s Year ended 31*st December*, 1948 CR.

RECEIPTS.	£	s.	d.	£	s.	d.	*1947.* £	*s.*	*d.*
y Entrance Fees and Subscriptions:									
Entrance Fees	286	13	0				*339*	*3*	*0*
Subscriptions (including £810 12s. 0d. received in advance)	7,127	6	6				*7,339*	*16*	*8*
Life Compositions	252	0	0				*63*	*0*	*0*
				7,665	19	6	*7,741*	*19*	*8*
Sales of *Transactions*				866	0	9	*793*	*1*	*3*
Interest and Dividends, including repayment of Income Tax for 1947–48				3,300	14	6	*3,232*	*1*	*7*
'*Consolidated Gold Fields of South Africa Ltd.*', *Premium for 1946*				–	–	–	*42*	*0*	*0*
Transfer from Reserve for Contingencies and Post-War Expenditure				–	–	–	*800*	*0*	*0*
, Excess of Expenditure over Receipts				930	12	5	*337*	*15*	*7*
				£12,763	7	2	£12,946	18	1

THE INSTITUTION OF MINING AND METALLURGY

SIXTH ORDINARY GENERAL MEETING of the 58TH SESSION held in the Rooms of the Geological Society of London, Burlington House, Piccadilly, W.1, on Thursday, 17th March, 1949.

Mr. DONALD GILL, *Vice-President*, in the Chair

The Chairman announced that the President, Mr. Taylor, was absent owing to an attack of influenza. Good wishes for his speedy recovery would be conveyed to him.

The revised Bye-laws of the Institution had been allowed by His Majesty's Privy Council and were now in force, and copies of the new Bye-laws would shortly be circulated. In the meantime he asked Associate Members, who were formerly known as Associates, to take note of their new title, the authorized abbreviation of which would be 'A.M.I.M.M.' The abbreviation for Members was 'M.I.M.M.' Students would be known as 'Student I.M.M.'

DISCUSSION ON

Geophysics and Economic Geology

By J. McG. BRUCKSHAW, Ph.D., M.Sc., D.I.C.

The Chairman, introducing the author, said that Dr. Bruckshaw was University Lecturer in Physics at the Imperial College of Science and Technology. He had himself read the paper with pleasure and instruction. He was glad that Dr. Bruckshaw had succeeded in treating a highly mathematical subject without using too much mathematical language, which showed what a good grasp of the subject he had. They were fortunate in having Dr. Bruckshaw to present his paper in person.

Dr. J. M. Bruckshaw said that in 1934 a paper was read before the Institution by Mr. S. H. Shaw on geophysical prospecting for water in Rhodesia and, in his introductory remarks, Mr. Shaw likened geophysics, which at that time had a flavour of novelty, to a vigorous mewling infant in whose every gesture the parents—in this case the proponents of geophysics—saw potential genius, while other observers, more detached in their attitude, saw just a naturally vigorous infant who, in the fulness of time, would attain a useful position in society.

After nearly 20 years the speaker had been called upon to review before the Institution the usefulness of the 'infant', now practically in maturity. The task was rather a difficult one. The main reason was that one had to consider not only the fundamental principles on which the various methods were based, but also the limiting factors which controlled the usefulness of those methods when they were applied. It was rather unfortunate that, in a paper of that description, it was necessary to dwell at great length on the limitations rather than on the successes which had been achieved. To quote a list of successes by themselves, without details of the actual specific conditions obtaining in each particular case would be meaningless. The most important part of presenting the potential usefulness of geophysical prospecting was to be able to assess the factors which controlled its application and limited the use of the various methods.

In the short time at his disposal he did not propose to go into very much detail, but rather to emphasize one or two of the major points which would be found outlined at greater length in the paper itself. The first important point was to appreciate that, when a geophysical survey was made, a series of physical measurements was actually made on the surface of the ground, the particular type of observation being selected in such a way that it was influenced by changes in some physical property of the rocks below. It was fairly obvious that from those physical observations deductions could be made only concerning the distribution of some physical property. For example, if a gravity survey were made and resulted in a gravity 'high' it would be justifiable to say that there were dense rocks (and if in gravity 'low', there were light rocks) near the surface. That was the major deduction. As a general rule, however, the picture observed at the surface was not one of specific characteristics of the rocks, but rather of a change in their character. As long as the rocks were uniform throughout there were no peculiarities to observe, so that the picture obtained on the surface was not a picture of large density or small density, but of the change from a large density to a small or vice versa.

If in ideal circumstances one could—and that in general was impossible—make a direct analysis from the observations and obtain a picture of how the density differences were distributed, it would still be impossible to identify the rocks that were producing the difference. Even if he quoted the density of a rock, others would be able to mention a large number of rocks possessing that particular density. There was no direct relation between the observations at the surface and the kind of material producing the disturbances observed. In other words, the identification of the particular mineral being sought by the investigation was not possible. That was one of the most important points to appreciate.

Another important fact was that, if the distribution of densities, the magnetic or electrical properties were known, and in the last case an electrical current were passed through the ground in a known way, then in theory it would be possible to calculate precisely the nature of, and variations in, the observations at the surface. In actual practice one had the reverse. There the observations were given and it was necessary to interpret them in terms of a distribution of physical property which had a geological significance.

There was an infinite number of interpretations of any one particular picture observed at the surface. Hence, from the geophysical observations alone, it was impossible to proceed directly to the unambiguous geological structure one was attempting to elucidate. It was essential to take into account all the known relevant geology in order to limit the number of possible interpretations of the geophysical data. Those interpretations should not only be sufficient to explain the geophysical observations, but should also fit the known geology. Even then, of course, there might be more than one picture, and it seemed to him that, on the basis of those tentative pictures, more exploratory work would be done with the gradual elimination of all but one, which would be consistent with all observations, both geological and physical. One of the most important points in the application of geophysics was that in the interpretation the geology should be considered on equal terms with the physical observations.

Briefly, the main limitations of geophysical prospecting in general were the ambiguity of the interpretation and the impossibility of identifying specifically the types of rock entering into the problem.

It was worth while making a further point. There were, as a rule, three possible ways in which geophysical methods could be applied. Sometimes the particular mineral which was of interest had its own specific physical characteristics. He specified, as an example, galena, which had one or two outstanding properties. It was dense, it was a good conductor of electricity, and, furthermore, such a mineral occasionally occurred in quite massive deposits, sufficiently large to allow the direct location of such a deposit by its own specific characteristics. Hence, one looked for a good conductor by electrical methods, and for a dense body by gravitational methods. More often the particular mineral, of economic importance, had not any outstanding characteristic which would permit its detection directly. Sometimes, however, it had associated with it other minerals whose properties were sufficiently significant to allow an indirect attack. As an example he quoted the search for gold by looking for the quartz reef in which it occurred. The concentration of gold was so small that it had no significant effect on the physical properties of the reef, but the insulating properties of the quartz occasionally allowed the location of the gold indirectly. Thus, there were available the direct and the indirect method, but there was also a third approach, —far more indirect than the other two, and one which was likely to be more useful than any of the others. It was a question of examining the geological features which were associated with the mineral deposit. An outstanding example was the search for oil. There the attack was indirect. The search was actually for a geological structure which was favourable for the accumulation of oil.

As a simple example of how that method might be applied in the case of base metal mining, the search for placer deposits might be considered. There it was possible to trace the topography of an old land surface and to determine the configuration of the denuded valleys in which the placer deposits might be found.

Quite a number of mineral deposits were associated with igneous activity. It might be that the disposition of the igneous rocks was of some significance in elucidating the geological structure, adding to the knowledge and leading ultimately to the discovery of the mineral. There were other examples, of course, but the members of his audience were already familiar with the problems of mining geology.

Before he concluded his short introduction he wished to point out that the paper under discussion portrayed a very grim picture of geophysical prospecting. That was because nearly all the discussion related to the limitations of geophysics, but he would conclude by pointing out that it had been found useful in the past and he was convinced that, by its suitable application under the necessary conditions, it would be found of increasing use in many different applications. Although, at present, all problems relating to economic geology could not be solved by geophysical prospecting —and it was fairly obvious that some types of deposit were not found under the conditions required for the use of the methods at present available—there was no reason to doubt that, in the future, new and possibly different techniques might bring more mineral deposits into the scope of geophysics.

Dr. A. A. Fitch* said that the present day was not a period of spectacular mineral discoveries in which geophysical methods played a large part. The rate of discovery by any technique was not especially high—in spite of the considerable shortages of particular metals which could be expected to stimulate exploration.

It might be of interest to examine briefly the technical matters which might be responsible for such a state of affairs. If the literature of mining geophysics were examined, most of the projects described were found to have been conducted on a rather small scale. The problem assigned was usually that of finding the extension of an orebody or searching for further orebodies near to those which were being developed. Such discoveries had, of course, to be checked by drilling or mining. The decision which management had often to make was whether to employ a geophysical method or to resort directly to drilling boreholes sited on geological inferences from what was already known. Many factors would affect such a decision, but at best the geophysical method became a tool of exploitation of a known field, rather than a method of exploration which might yield spectacular discoveries.

By far the greater part of the ores mined today came from fields which were discovered from evidence at the surface. The next phase in mineral exploration was to discover the fields which presented no evidence at surface. That required that the economic

* Seismograph Service, Ltd.

geologist specified as precisely as he could the geological environment in which he expected to find the ore with which he was concerned; such a statement required long and careful study, and that was perhaps the phase of the work of collaboration between economic geologist and geophysicist which had received least attention. Next, the problem of searching for such a geological environment was considered by the geophysicist; if a suitable method were available it would be employed in the field. Any suitable environment discovered by the geophysical survey would be examined by drilling or by more detailed geophysical work, including observations in the boreholes. The function of management in all that, apart from directing the operations involved, was to evaluate the chance of success at each stage, and to decide whether the project was justified. What was the chance of locating ore in the environment specified, and what was the chance of the geophysicist discovering such an environment? What size of orebody was likely to be discovered, and what was it likely to cost to mine the ore? On the bases of such estimates the risk would have to be calculated.

He asked the author whether there were examples of that indirect approach to the problem of finding new mineral fields.

Professor A. Hubert Cox* said that there was now, of course, no question of the importance of geophysical processes when used in suitable hands and in suitable areas. It was necessary only to mention the extent to which geophysics was used in the search for oil. It had always struck him that one of the cases Dr. Bruckshaw had mentioned—the tracing of the magnetic shales in the Witwatersrand system, where they were under the cover of other rocks—was a very brilliant achievement. There the surface structures gave no hint whatever of what was beneath, and it would have taken a very long process of drilling to prove all that was effected by geophysical methods.

One difficulty in that subject had always seemed to the speaker to be the different methods of approach. There was the approach by the physicist on the one side, and on the other the approach by the mining engineer and the geologist, and often they spoke different languages. The physicist, of course, liked his equations, and if the physicist were asked how a certain material was likely to behave, he at once enquired what was the size of the body, how deep it was, and what was its susceptibility or conductivity. To this the reply often had to be made that the size and depth were not known, and that the susceptibility probably varied enormously, owing to variations in the composition of the body. He himself was fortunate in being associated with a physicist whose help was very valuable, since he realized that in any case the observations he was making were only approximations, and so he did not mind if the mathematical results had to be manipulated in order to get some sort of general picture.

* Department of Geology, University College, Cardiff.

It was in all cases important that observations should be checked by the geologist or the mining engineer. In earlier years of those studies the subject fell into disrepute in some parts of Canada by the appearance of certain companies who undertook to make a survey of an area, using in many cases a rather quick high-frequency electrical method, and handed in their results saying, 'Here is the disturbed area, and it is up to you to test it'. In several cases what was found was nothing more than a bog.

The speaker illustrated on the blackboard one example of the manner in which a magnetic survey had been conducted for experimental purposes. The man who had done that had identified the high spots scattered about in the area. That, of course, was very simple. He had then proceeded to draw contour lines around those high spots. When the area came to be investigated geologically it was found that the principal peaks had no physical connection one with the other at all. It proved to be an area where there had been great horizontal thrusts of alpine type and consequently there might be one orebody at a high level giving rise to one of those magnetic peaks, and far down below another body with no structural connection with the higher mass. The original magnetic picture was most deceptive from the point of view of actual prospecting, though a later magnetic survey, taking the geological structures into consideration, proved most valuable.

With regard to the difficulty of the physicist and the mining engineer speaking different languages, he had found it difficult in the case of the earlier text-books on geophysical surveying to discover any book he could recommend to students to study the subject quickly. The books were apt to be devoted to the theories on which the apparatus was constructed. But one did not always need to know the theory on which an apparatus worked in order to use it. It was not necessary for the motor-car driver to understand the theory on which the car was run. In the field one soon found where 'anomalies' occurred; and one could test them against the local geology and then proceed further.

Mr. Gilbert McPherson said that geophysics was rather a controversial subject and at the same time one of wide interest. Having just returned from abroad he had not had time to prepare his considered views and his remarks might therefore be rather abrupt. While congratulating the author on bringing forward the paper he thought that the treatment was not really in sufficiently sombre tones. Geophysics had grown up from its early stage of infancy and it was surely time to consider coldly how far the claims made on its behalf were justified as regards the metalliferous mining industry. Field exploration by geophysical methods was an expensive matter and, after all, the mining engineer's essential purpose in life was not geology or physics but to find and exploit profitable ores and to find them by the cheapest methods. In his view it was extremely rarely that profitable orebodies had been found by geophysical methods.

For 25 years he had watched the progress of geophysics fairly closely, noting not only the actual discoveries of value made by those methods but also the discoveries which proved to be negative or non-economic. As a result of his observations he concluded that, save in very special instances, geophysics had not proved to be useful. They would all remember, for example, the discovery of the Rhodesian copper fields. A great deal of expensive geophysical work was carried on there, but he did not know of any useful discovery there due to that type of prospecting. On the other hand, several bodies of graphite schist were found and there was one discovery of serpentine which had not previously been located for several hundred miles in any direction. In certain special cases geophysics could be particularly useful, where there was a strongly magnetic or good conducting orebody covered by a few tens of feet of gravel, moraine, or other similar shallow cover. The lead-zinc mines of Newfoundland seemed to be an instance of that. Similarly, in Sweden certain valuable lenses of auriferous copper sulphides and certain iron bodies had been traced or located successfully under shallow cover. But over hundreds of cases it seemed that geophysical methods had mostly failed to yield dividends and in the ultimate analysis it was dividends that must be sought.

He spoke of one of the most prominent cases quoted in recent years—namely, the tracing of magnetic shales on the Rand to indicate the position of auriferous banket beds. It had been claimed that magnetic methods had established the location of valuable deposits, but those engineers who were old enough and knew the circumstances would remember that that particular area had its general structure revealed by ordinary geological methods and by certain drill-holes long before the geophysical methods were used. It seemed to him that the magnetic methods there had been a useful auxiliary. Another field in which geophysical methods had been widely used was in the new goldfield in that neighbourhood. The geophysical methods there initially scored a bull, right in the centre of the target, for which ample credit should be given, but the negative side had had little mention—namely, that large areas, later shown to be probably of high economic value, had been dropped on consideration of the geophysical data.

He did not know of any ordinary type of orebody containing only lead and zinc sulphide minerals occurring under normal conditions in which the orebody could be definitely located by geophysical methods. Last year he had read a report of the tonnage of such ores which had been proved by geophysical methods, but he did not know how it was done and was inclined to think that the use of the word 'proved' was incorrect in that connection. He had noticed that in Canada geophysical methods had shown a literally enormous number of fissure zones, but those present knew that it was only after finding a fissure zone that the real trouble and expense in exploration commenced. One had still to find the orebody. As a further example he mentioned a case in Jugoslavia where geophysical methods were applied as a test on one property over a known orebody which was correctly indicated, but, when applied to a neighbouring property, with certain similarities, proved to give fallacious indications. Drilling of the strongest anomaly showed a greenstone mass.

He emphasized his belief that for most instances the ordinary methods of prospecting by geological examination combined with trenching and shallow underground work followed by drilling and deeper underground work was the cheapest method. The only possible result from geophysical work was a map showing anomalies, sometimes masses of anomalies, but one could not mine anomalies. It was necessary then to start in by the usual methods to find out what those anomalies meant.

He could quote a number of cases but he would only say that while in oil exploration there was no question whatsoever of the value of geophysical methods, in mining, in looking for orebodies, those methods were a doubtful weapon. There had been hundreds of cases of geophysical work being carried out with few successes. He considered that geophysical methods should only be employed under the control of an experienced geologist who thoroughly understood their limitations.

Mr. W. Bullerwell* said that he had not intended to participate in the discussion, but the last speaker had stung him to action. Geophysicists would agree that orebodies could not be distinguished from materials which had similar physical properties and which might occur close to the ore. Dr. Bruckshaw had explained the lack of specific recognition of any material by physical measurements at the surface. Nevertheless, the determination of structures could be made. If it were known, for example, that an ore-field was terminated by a boundary fault, it was probable that use of a geophysical method would locate the fault. Again, if it were known that deposits were related to an anticline it might be possible by geophysical methods to trace the axis of that anticline. He was convinced that in mining geology, as in oil geology, the accurate determination of such structures was of great importance and in many cases could be done by geophysics.

On some problems it was important to use all the geophysical methods applicable. The gravitational method, for example, gave an ambiguity of interpretation. The electrical method also gave an ambiguity. Nevertheless, the nature of the ambiguities in the two cases might not be identical, and therefore it might be possible to resolve them by comparing the anomalies obtained using the two different methods. He thought that very often attempts had been made to carry out mining explorations using only one geophysical method, perhaps leaving all its ambiguities to be resolved by a single bore-hole, when a sounder interpretation could have been made had more geophysical methods been employed. Quite

* Geophysicist, Geological Survey.

frequently in making surveys he himself had wished to re-examine an anomaly by a second method. In some cases that had been done successfully, but there would be cases where ambiguities had to be resolved by drilling.

He thought that all geophysics in the field should commence in a locality of accurate geological information, and that as the survey was developed every effort should be directed to keeping that information up to date so that the interpretation was made in the light of the best geological information available.

Many geophysicists themselves treated a survey primarily as a method of revealing anomalies, thus indicating, as Dr. Bruckshaw had said, areas of unusual physical behaviour, and those should then be tested by drilling. Often, because of the ambiguities, a general survey to reveal the major anomalies was advisable. The main anomaly should be drilled or otherwise investigated, and further geophysical work depended on the results. The first drill might show how the anomaly was related to the ore being sought or to some other feature, and the survey might then be carried on with modified detail.

Sir Lewis Fermor thought the author had been unusually gloomy at the end of his paper and he was sorry Mr. McPherson should have thought it necessary to increase that gloom. By pointing out the drawbacks of geophysical methods the author was really strengthening the position of geophysics by making them all sympathetic towards his subject and leading them to consider in what way geology could help geophysics, realizing, of course, that action and reaction were equal and opposite and that consequently geophysics would help geology.

Dr. Bruckshaw had ascribed to the geophysics ' baby ' an age of only 20 years.

Dr. Bruckshaw: Roughly.

Sir Lewis Fermor thought that he was unaware of the history of geophysics as a whole, especially of the magnetic methods. In 1910 the speaker was in Sweden and visited the iron ore fields of Lapland. At Kirunavara, which was the biggest deposit, he was shown a map depicting the results of a magnetometric survey of the underground continuation of the ore deposit. He had not heard that those magnetometric surveys had disappointed the Swedes in their results.

He then went on to describe how, on returning to India, he decided to see what could be done with the magnetometric testing of manganese ore deposits in the Central Provinces: for he had already discovered that certain manganese minerals were magnetic -e.g. braunite, sitaparite, and vredenburgite. He went to the Survey of India to see what instruments they had, and from them he extracted a dipping needle which he took to the Central Provinces. He took readings both on manganese ore deposits and on adjoining outcrops of gneisses and schists, and found the most surprising differences. He had, however, never published an account of the work, because he could not get the instrument to read consistently, due to the balance weight of the needle always getting out of adjustment on the railway journey from Calcutta to the field. Nevertheless, a geophysical method was thus applied to the manganese ore deposits of the Central Provinces as long ago as 1911–12. He had long thought that there was scope for the use of magnetometric methods in the study of such deposits, and the Geological Survey of India and the Survey of India had in recent years used magnetic methods to test the extension of manganese deposits in the Central Provinces.*

He himself had in mind the following type of experiment. One very valuable manganese ore deposit had been cut off by an igneous intrusion. The mine was now apparently finished. He felt that that was a case for a magnetometric survey ; should anomalies be thereby revealed, the known geological facts would justify a boring to discover whether the anomalies pointed to the position of the remainder of the deposit.

Water could, of course, be found by the use of resistivity methods, and in Southern Rhodesia the Irrigation Department, which was a section of the Agricultural Department, was now using resistivity tests to discover sites for wells for farmers. The farmer was charged one fee if the result was successful, and a smaller fee if it was unsuccessful, the Department thus displaying its experience of the usefulness of this method of prospecting for water.

Mr. J. E. R. Wood† said that, as Dr. Bruckshaw had stated, nearly all geophysical methods had been developed to the state where the general background was recorded. In some circumstances a part of this background was removable from the observations. The example quoted for electrical methods, and illustrated in Fig. 8, was applicable to a certain extent to both magnetic and gravity methods. If the background were due to an unwanted body which gave an anomaly of long period compared with that of the wanted body, then the former could be largely eliminated.

In the magnetic method a part of the background might be due to near-surface variations of magnetic properties which were unpredictable, but their effect might be considerably reduced by the employment of an airborne magnetometer rather than a surface magnetometer.

When, however, the un-removable background level had been reached no further increase in the sensitivity of the instruments

* *See* Dessau, G. Past and future of exploration geophysics in India. *Trans. Min. Geol. Metall. Inst. Ind.*, Vol. 43, 1947, pp. 41–66, for a review of work done on this subject in India. From this the speaker discovers that the records of the Geological Survey of India show that his first attempt to apply magnetic methods of study to Indian manganese deposits dates from 1906, when the dipping needle mentioned was ordered on his recommendation.

† Anglo-Iranian Oil Co., Ltd.

employed was of any value and new methods must be sought to obtain advances in prospecting. In that connection an article by W. M. Barret in *World Petroleum*, for March, 1949, was of interest. It described a small radio transmitter, operated on the surface of the ground, which yielded a strongly-received signal in a salt mine, after passing through 1,200 ft. of sedimentary rocks. That could lead to the development of prospecting for orebodies by the reflection from them of electro-magnetic waves.

Dr. T. Pickering said that he approached the subject from a mining geologist's angle. He was not a geophysicist, but had had practical experience of geophysics and was of the opinion that a good deal could be said for the methods in the search for metals when they were properly allied with geology. If one succeeded in concentrating the area of search by geophysical means, one was cutting down the cost of prospecting. That was all that geophysics was doing for oil and it was being done, on a smaller scale, in the search for minerals. It was very rare that metallic minerals could be directly located by geophysics, but many cases could be quoted where considerable saving in prospecting costs could be attributed to the information which it had given to the geologist.

He had carried out a number of geophysical surveys recently and had come to several conclusions as to the way they should be conducted. First, the survey should be controlled and supervised by a geologist. The survey should, generally speaking, include several geophysical methods, for it was found in practice that each method contributed to the sum total of information, or cross-checked the results of other methods. With regard to the first point, the ideal geophysicist would be one who was also a geologist, but that combination was exceedingly rare. Failing that, it was necessary to have a geologist trained in the practice, interpretation, and—as far as possible—the theory of geophysics. He should be the man to direct the survey.

On the question of costs—a field party that he had handled recently consisted of one geophysicist, three field observers, two surveyors, and a geologist in charge of the operation. That survey cost £400 a month in salaries, a low figure because it was possible to use inexperienced men as field observers, with as little as two months' training in the operation of their instruments. The prime cost of instruments for a magnetic, electrical, and gravimetric survey was about £3,000 and, depreciating that over a two-year programme of field work, a company operating in an area with cheap native labour, and for an all-in cost of about £900 per month, could produce a monthly output of some 2,500 magnetic stations, 1,000 electric, and 700 gravimetric stations, together with the geological interpretation and maps. He felt certain that the companies concerned with the work he had carried out did not regret the expenditure, for the economic results had been encouraging, although several years might be needed before the full assessment of their value could be made.

Dr. K. C. Dunham said that he was not a geophysicist, but geophysical investigations had been carried out in connection with some of his earlier geological work, and it might be of some interest to the Institution to hear something of work done in Great Britain. In the Furness district it would be recalled that haematite deposits occurred in depressions in the Carboniferous Limestone, sometimes as much as 600 ft. in depth, with a central core of soft sand. Those deposits offered a useful target for the indirect method of geophysical investigation, by working out the boundary between the hard rock—in that case the haematite—and the soft sand. He thought that had seismic methods been applied to that district at an early stage they would have had considerable success. As it was, they were not applied until about 1936, and, although two depressions were found, when they were drilled it was discovered that they were merely lined with sand. The experience showed, however, that for that type of deposit an indirect method could have been used successfully. Later on magnetic observations were carried out over the haematite deposits by Dr. A. F. Hallimond. Mr. A. J. Butler, and others, using very small anomalies, and it was found that a proved body could be outlined by that method with a cover of only about 50 ft. of drift. The method was applied successfully in locating a small but economic orebody at the Newton mine. A considerable amount of additional work was done in war-time, but it was not all followed up, and in some cases the results were unsuccessful.

With regard to lead and zinc, he referred to resistivity work carried out in East Allendale for Weardale Lead Co. by a German firm. That unfortunately was faced with too complicated a problem to be capable of solution by the method used and was a warning against the application of comparatively simple methods to complex problems. The country rocks were an alternating series of limestones, sandstones, and shales, traversed by a deep glacial washout. Lead ore occurred in veins and metasomatic flats in the limestone. What the resistivity method indicated as possible orebodies were afterwards discovered to be approximations to the course of the washout. He agreed with Mr. McPherson that in that instance the experiment proved a costly one.

About the same time there was in the Derbyshire district some geophysical work being done in the Mill Close area. He believed that some form of high-frequency electrical method was used there, but he did not think it was ever established whether it was really a success. It led to the abortive Mawston exploration, north-west of Mill Close.

More recently still, in the northern Pennines, the Geological Survey had used the magnetic method primarily for mapping purposes. The object was strictly to find the courses of faults to which were related lead-zinc and barytes deposits. The use of that method depended on the shift of the Whin Sill quartz-dolerite, which revealed the fault-lines by striking anomalies in the magnetic field. The faults could be followed considerable distances by means of geophysics where the surface geology was completely obscured by drift. In the case of the south-westward extension of the Rotherhope Fell vein geophysics gave unambiguous evidence that the fault did go on. Another case was the Closehouse barytes property, an account of the geophysical investigation of which, by Dr. Hallimond and Mr. Butler, was shortly to appear. But, of course, none of those investigations led directly to the discovery of ore. They merely amplified the geological picture.

He would contribute one point to the argument raised by Mr. McPherson. At the International Geological Congress last year, in the course of the discussion on the geological results of applied geophysics, an account had been given of the location by gravity methods of an economic, although not large, lead-zinc orebody at Pinos Altos, New Mexico.

Dr. A. W. Groves said that, although the science of geophysics was still comparatively new, the body of literature attaching to it was already quite considerable. Many papers had been largely concerned with the search for oil, the application of particular methods, or recording the results of geophysical surveys, but apparently few had been addressed specifically to reviewing the relationship of geophysics to geology and mining. However, of papers in that latter group the one they were discussing that evening was one of the very best, and mining engineers and geologists in general would be indebted to Dr. Bruckshaw. The author had been quick to emphasize that a collection of geophysical data alone was of small value and that combined with all the geological knowledge available for the district it was of much greater value; but it was only when, in addition, mining engineers and economic geologists got together and tested, either by drill or underground exploration, the further problems that might then be indicated that they were going to reap the full harvest.

Although Dr. Bruckshaw was dealing with fundamental matters, he had proved himself expert in confining his attention to those essential to his argument. The paper was of particular value not only in giving to those mining engineers who were relatively unacquainted with geophysics some insight into the principal factors controlling the choice of methods most suitable for a certain problem, but also in drawing their attention to the shortcomings and difficulties of interpretation of the results.

That there might sometimes be interpretative difficulties, even when a fair amount of geological information was available, emphasized the futility of attempting to interpret geophysical results alone. Nevertheless, failure in the past to realize some of the shortcomings and ambiguities of those methods, or the use of methods ill-suited to the problem in hand, had sometimes led to the opposite extreme of unjustifiable scepticism. Once the geophysicist had located a sizable anomaly for which no indubitable explanation could be given, it was up to the economic geologist and mining engineer to ascertain the cause. It was often easy to indicate the presence of something different, but so difficult to prognosticate its identity with any degree of certainty.

Turning to the magnetic method, the one with which the speaker had had field experience and the one usually regarded as the cheapest and most easily applied, it was good to see that Dr. Bruckshaw stressed the complications arising from the combined effects of the permanent and induced magnetism that were both noticeably present not only in magnetic orebodies but in many igneous rock bodies. It was perhaps the geologist who was apt to overlook the effects of induced magnetism arising from the earth's field, while the geophysicist might be more apt to fail to link up the abnormalities of an anomaly with the partial dislocation and faulting of an orebody. Was it not possible that the upper end of an orebody might be broken into segments by earth movements, and each segment displaced the same way with respect to each other so that in the result the axis of elongation of the orebody, say of magnetite, had been rotated appreciably away from its original position? In such a case might not the difference in orientation of the main axis of the magnetic anomaly and that of the outcrop or upper end of the orebody become considerable? Only on some such lines it would seem could the case of a magnetite orebody in Shetland be explained. That magnetite orebody which he had surveyed in detail for the Ministry of Supply—but the results of which had not been published—had a N.–S. outcrop some 200 ft. long (slightly but progressively side-stepped by faults of small displacement) and averaged about 10 ft. in width and dipped at 60°. When the outcrop was stripped of its partial cover of peat and drift it was found that the outcrop and innermost zones of the anomaly, when plotted on the same sheet, crossed one another near the middle like the blades of a half-open pair of scissors! The difficulties in intercepting similar but concealed orebodies by boreholes based on the indications of magnetic anomalies alone were manifest.

Dr. Bruckshaw also mentioned other difficulties sometimes attaching to the interpretation of magnetic anomalies and emphasized that they were more difficult to interpret than gravity anomalies, to which they were related. Obviously the number of strongly magnetic orebodies was strictly limited to a very few minerals. In other cases the ore sought might contain a certain amount of strongly magnetic mineral, or the ore might be associated with or related in a definite manner to a magnetic geological horizon. For these reasons the author was right in concluding that the magnetic method was mainly indirect in its application; nevertheless the tracing of a magnetic or marker horizon could be of the greatest value in outlining concealed geological structures.

With regard to speed of operation, the author stated (p. 19) that 'in open country some 20 to 30 magnetic stations at close intervals (25–50 ft.) can be made per hour by a skilled man'. Dr. Bruckshaw was here presumably referring to the magnetic balance type of instrument, and was assuming that the traverse lines had already been laid out, and the station intervals marked. It was remarkable, however, the way gorse, brambles, or peat increased the time taken in levelling the magnetometer and so increased operating time generally. Under such conditions 12 to 16 stations an hour might be very good going. The Thalen-Tiberg magnetometer could be operated somewhat more rapidly but, owing to its much lower sensitivity, its usefulness was largely limited to strongly magnetic subjects.

Mr. T. Dewhurst said that the remarks of Sir Lewis Fermor had reminded him of the early introduction of geophysical methods to prospecting for oil. The Egbell oilfield, situated in the Vienna basin, was discovered in 1917, and the application of geophysics to the search for oil dates from the recognition that that oilfield was situated on a torsion balance ' high '. Professor de Bockh was the first to recognize this coincidence and to appreciate its significance, and after the 1914–1918 war he visited Great Britain to explain to the speaker that new weapon in the armoury of the petroleum geologist. The torsion balance was soon widely used in the search for oil: it was followed in due course by the introduction of the seismic refraction method and, later, by the seismic reflection method. That war gave a tremendous impetus to the search for oil and the three geophysical methods were pressed into service, being well established by 1924. It is noteworthy that the introduction of each method was followed by a spate of new oilfield discoveries.

The speaker then referred at some length to several incidents which indicated that in those early days somewhat extravagant claims were made for geophysical methods, which were expected to supersede and displace geological methods of search. However, experience in due course demonstrated that the best results were obtained by the closest possible collaboration between geologists and geophysicists. There could be no doubt that geophysical methods had been of immense value in the search for petroleum deposits and that fortunate result was probably due largely to the fact that the structural and stratigraphical conditions favourable for oil accumulation were relatively simple and that usually the targets were of considerable size. He had no experience of the application of geophysical methods to mineral prospecting, but it appeared that the geological environment of mineral occurrence was far more complex than was the case with oil and that the application of geophysical methods was, therefore, much more difficult and uncertain. That consideration doubtless explained the somewhat pessimistic treatment of the subject in the paper, which he had read with great interest.

He was interested in Fig. 6 of the paper, as that presented a simple and straightforward picture of the results of a torsion balance survey carried out in the Orange Free State. He had been somewhat impressed with the torsion balance and would like to ask Dr.

Bruckshaw to what extent the results had been confirmed by subsequent drilling operations.

Dr. Bruckshaw said he was speaking from memory, so that he did not wish to be dogmatic, but he thought that the drill-hole put down on the gravitational low entered the important sequence of rock at a depth of 900–1,000 ft., in comparison with a thickness of several thousand feet elsewhere.

Mr. A. E. Gunther said that he hoped he was not drawing a red herring across a highly technical discussion, but he would like to refer to a recent letter in a technical journal in which Dr. Bruckshaw made an unanswerable case for the expansion of geophysical surveys in the Colonies. It was gratifying to read from time to time of the interest shown by Government officials in geophysical surveys, but he thought that progress had been slow.

It was clear from the past that Government departments were not always competent to handle highly technical problems. The question in that case seemed rather to be one of the appropriate authority in the country. Who, for example, was responsible for seeing that geophysics was employed in Colonial Surveys? In the last year an Assistant Director (Air) had been appointed to the Directorate of Colonial Geological Surveys. The speaker would welcome the appointment of an Assistant Director for Geophysics.

It would, of course, be a mistake to create a separate institution for that purpose, because it was essential that geology and geophysics should go hand in hand. The ideal solution would be integration between the British and Colonial Geological Surveys and a Department of Geophysics established to serve both. That opinion seemed to be increasingly shared among Colonial geologists, who were in any case unanimous in desiring the spread of geophysical surveys. Unless the matter was considered seriously in the near future it might become necessary to call on Marshall Aid for geophysical work as we had already called on it for 25 geologists!

Dr. G. V. Hobson* said that he had had the privilege of discussing with Dr. Bruckshaw the relation between geophysical methods and the problems of his (the speaker's) own department, and he noticed with the greatest interest that Dr. Bruckshaw suggested that it would be possible in connection with placer deposits to plot an ancient land surface. That, he felt, might be of great assistance in a slightly different connection. They were at present proposing to make experiments along those lines, and while they remained unconvinced at the moment at least they were setting out hopefully.

One of the bugbears with which they had to deal was ice action, which could be extremely tiresome in relation to the discovery of shallow deposits of coal to be worked by opencast methods. Where a shallow deposit existed it would be subject, if glacial action had taken place, to extreme variation of erosion. If the old land surface could be plotted by means of geophysical methods they would

* Director of Opencast Coal Production, Ministry of Fuel and Power.

achieve something of value in eliminating the risk of opening up a deposit where they expected to find a considerable tonnage of shallow coal, only to find that, in spite of what they had done, ice action had removed much of the coal along the sub-outcrop. That was a problem in which they felt they might derive quite useful help from geophysical methods. He just mentioned it as one possible instance where geophysical tests, coupled with the information obtained from drilling, might assist them in avoiding the pitfalls which had occurred in the past owing to the unpredictable behaviour of the surface of the land which had been left by glacial action.

Dr. Bruckshaw said that he would prefer to have the whole report of the discussion before him before making any reply.

The Chairman proposed a hearty vote of thanks to Dr. Bruckshaw for his paper, and this was carried by acclamation.

CONTRIBUTED REMARKS

Dr. James Phemister*: I note that Dr. Bruckshaw says (p. 10) that the Eötvös torsion balance 'is, for most practical purposes, obsolete'. I do not agree with this view and I consider that this instrument is able to contribute very valuable information on local anomalies in amplification of the results obtained by the gravity meter. Its employment, however, requires a considerable organization and is comparatively expensive.

Dr. G. A. Schnellmann: With some reluctance I strike what may be a jarring note. Strong, however, in the experience that adverse criticism can provide a powerful stimulus, I suggest that mining geophysics is a disappointment—not because its results have been negligible, but because it seems to have made little headway in overcoming its limitations and expanding its sphere of usefulness. Recent progress in geophysics appears to have been solely in the direction of time-saving. No doubt this is a case of the tune being called by the man who pays the piper. The oil industry is the principal user, and for its particular problems the advantages of the gravimeter as compared with the torsion balance, or of flying a magnetometer over inaccessible swamps and open sea, are obvious. So far as the typical mining problem is concerned, time is less important and the greatest need is for improvement in discriminating power and depth penetration of the standard mining methods ; or, of course, an entirely new method. In these fundamental directions it appears to me (and I shall be very glad to be assured that I am wrong) that mining geophysics has made little progress in the last 20 years—since, that is, the work on which the Broughton Edge and Laby report of 1931 was based. The spectacular case of the Orange Free State is essentially an instance

* **Assistant Director, Geological Survey and Museum.**

not of *mining* geophysics but of *oil* geophysics—cf. the scale and the interpretation. The geophysical investigation did not give a positive indication of the sought orebody, but proved a structure some 15 square miles in extent. The presence of auriferous conglomerate in this was inferred from stratigraphical knowledge. This type of problem may well prove to be of increasing frequency in mining geophysics. So far the main problem confronting both the mining geologist and the mining geophysicist has been that of locating individual *orebodies*, usually in or adjacent to proved mineralized areas. In the foreseeable future the problem of locating new *ore-fields* is likely to command increasing attention, and here the methods and scale of *oil*-geophysics will play an important role.

Lest any of the foregoing remarks be misinterpreted as uncompromising opposition, may I conclude by saying that I have employed mining geophysics, and shall certainly continue to do so when appropriate problems arise. My only hope is that the methods may be so developed that they can be applied to a greater proportion of problems than is now the case. Dr. Bruckshaw's paper, useful and important in itself, provides also the basis of a useful stocktaking.

Professor John T. Whetton: This paper appears to be an elementary sketch of the various geophysical methods now in vogue. Research and field application have reached a stage when specific cases could be considered and discussed with advantage to the Institution.

I would suggest that fairly accurate estimates of costs of geophysical surveys can, and in fact are, being made. Most of the well-known oil companies employ their own geophysicists and from their experience the costs are easily obtainable. Furthermore, estimates are submitted to the oil companies by contracting geophysical firms and these estimates are checked very closely before acceptance.

The author states that 20–30 magnetic observations may be made in an hour. With the vertical force variometer, a good rate for accurate work is 12–15 stations an hour for close intervals. It is impossible to state a definite distance of electrical traverse which may be done in a day, as speed varies mostly with different electrode-spacing and with intervals between readings.

Mr. S. H. Shaw: The author is to be congratulated on a concise and up-to-date account of the principles and limitations of the three geophysical methods most likely to be of use in mineral exploration. That he feels, however, that some apology, as it were, for the intrusion of anything so scientific into prospecting is still necessary can be inferred from his reference to the picture being painted in sombre tones and from the statement of faith in his last sentence. This I think is unnecessary.

Geophysical methods are now in general use, and have been for some years, and the mining community is not a philanthropic body that would be content to sponsor the activities of geophysicists if, on balance, those activities were not of value and profit to the miner. The advent of geophysics has not altered the old rule that only one out of every 100 prospects (or whatever the true figure is) will turn into a payable mine. The proportion of payable mines discoverable by geophysics must be even smaller, since not all prospects will be suited to examination by those methods. Just as the one good mine justifies the examination of the other 99 prospects, so the occasional outstanding geophysical success will justify indifferent or disappointing results in many others.

A point that I should have liked to have seen dealt with by the author concerns the best method of undertaking geophysical work. For the seismic and gravitational methods the specialist organization is no doubt the only practical solution, but for the electrical and magnetic methods, where, as the author says, the instruments are cheap, the answer is not so certain. There is no doubt that the specialist organization has the advantage of wide experience and constant contact with geophysical problems, but it is relatively expensive and often would not be called in except for a major problem or, perhaps, like specialists in other spheres, until the patient is past recovery. The conclusion reached by such an organization may also be that with wider application and more detailed work the problem may be usefully solved. Although the mere purchase of equipment will achieve nothing unless it is used in a rational way the routine collection of geophysical data might in some instances serve a valuable purpose.

In conclusion I should like to ask the author whether the ordinary graduate in mining or mining geology is acquainted with the handling of the simpler geophysical instruments or whether such instruction as he has had is in principles only.

Dr. A. C. Skerl: This thoughtful paper by Dr. Bruckshaw is a timely accounting of the status of geophysical prospecting as applied to mining geology. Many members will have compared with interest the useful review by A. A. Brant on the same subject* published a few months earlier. Both authors point out the limitations of the methods and how results are often inconclusive, but the American author is blunter in his criticism of many geophysicists for glossing over such shortcomings.

There appears to be a tendency towards the belief that the main use for geophysical work is the finding of ore hidden at considerable depths. I feel sure, however, that in the majority of cases the problem involves a cover of overburden less than 100 ft. thick. A recent example in British Columbia was the detecting of high-grade silver-lead veins only 1–2 ft. wide under a cover of boulder clay up to 30 ft. thick, using the self-potential method.

* Brant, A. A. Some limiting factors and problems of mining geophysics. *Geophysics*, Vol. 13, No. 4, Oct. 1948, pp. 556–81. (Abstract in *Min. Engng.*, Vol. 1, Jan. 1949, Sec. 1, pp. 28–32.)

In the Philippines in 1940 it was possible to outline the shape of auriferous orebodies with a high sulphide content in narrow, steep veins using the resistivity method. By setting up at regular intervals along the lines of ancient workings and increasing the electrode spacing, successively deeper penetrations were obtained as far down as 700 ft. The resistivity values were plotted on a vertical projection to give a pattern that was checked closely by subsequent underground exploration and stoping. Experimental work in 1941 showed that similar results could be obtained over lenses of chromite on the island of Mindanao, but the further testing by drilling and mining has been delayed until now by the Japanese invasion and its aftermath.

The mining geologist is able to supplement his experience considerably by the published accounts of numerous mineral deposits but the literature normally available to him contains few descriptions of geophysical prospecting from his point of view. I would, therefore, suggest that this excellent paper by Dr. Bruckshaw is followed up by a series of authentic short papers on various recent geophysical explorations, successful and otherwise, emphasizing the geological settings and their effect on the choice and application of the methods used. If sufficient response is obtained, then the publication of a separate volume could be considered.

Mr. Donald F. Foster: Previous to 1902 the Daft-Williams system of sending impulses of induced electric current into the earth was proposed. During trials it was found that certain earth anomalies interfered with transmission and that pipes—metallic or porcelain—interfered with the induced currents. It was from this point that prospecting for economic minerals was evolved.

The method of introducing the electric impulses into the earth was similar to that used for the transmission of early wireless messages—electric storage batteries, an induction coil, and a condenser with a regular ‘ make and break ’ being used.

Further experiments in Cumberland and Wales proved that known reefs or lodes could be plotted by such induced electrical currents. The induced current tended, it was found, to concentrate on the margins of the reef, lode, or dyke. During three years’ field experience there was no proof that the current ever flowed through the earth. In a few cases we were able to measure the depth to the cap of the lode or reef and it was found that one third of the length of the base line gave the depth of the cap.

During a survey on a large stockwerk in Australia, made with the object of picking out the more valuable areas, it was found that impulses concentrated on the quartz stringers. The whole of the stockwerk outcrop was electrified, and the quartz stringers—even the tiniest—gave loud reactions in the microphones. On a porphyry blow the margins gave loud reaction, but across the outcrop things were normal.

Trial over three years proved the system to be of very little economic use, although it did prove that once the outcrop of a reef or lode was discovered its course or strike could be traced; that faults could be located; but that at no time could the value of the lode, kind of anomaly, or kind of lode be determined by electrical prospecting alone.

There are some occurrences, however, where geophysical prospecting might have proved profitable. For instance there are several areas on the Gold Coast where considerable pitting has been done and which the average engineer refers to as large alluvial workings, but which in reality are old outcrop workings and washing pits. To an observer the $\frac{1}{2}$-in. to $\frac{5}{8}$-in. pieces of quartz show where the reef has been broken and washed, and a trench 6 ft. deep cut through these will show the conical washing pits. Here, then, electrical prospecting might be useful in tracing the hidden lode.

AUTHOR'S REPLY TO DISCUSSION* ON

A New Form of Thermal Precipitator

By P. H. Kitto

Mr. P. H. Kitto: Before replying to the various points raised in the discussion concerning this instrument I should like to express my thanks to Mr. Annan for undertaking to introduce the paper at short notice, owing to the unavoidable absence of Professor Ritson.

Mr. Watson mentioned the possibility of spraying a film of collodion on to the particles on the 3-in. by 1-in. slide to enable it to be examined with a 2-mm. oil immersion objective, but if this is done it will be found that a large number of the smallest particles become invisible, owing to the smaller difference in refractive index between the particles and the surrounding medium, and the results will not be as good as those obtained with a 4-mm. apochromatic objective.

Both Mr. Watson and Mr. Walton pointed out that deposition of large particles would take place in the inlet tube, and it would be an improvement to mount the instrument close to the outside of the box so that the length of this tube was reduced to a minimum. In practice, however, it is found that particles larger than 2·5 microns are found on the record, and it is probable that convection currents keep some of these particles from falling out. On these mines the percentage number of particles above this size is usually very small, and comparative size distributions with samples taken on the standard thermal precipitator showed very little difference.

Mr. Davies suggests that the 85 per cent efficiency might be due to a lower temperature gradient between the surface of the glass slide and the wire, but this was tested by taking samples with the slide stationary. Any inefficiency of deposition would show up immediately by a spreading of the particles, particularly the large particles, along the slide, and there would be no well-defined edge to the record, but no such spreading was observed.

With reference to the use of a standard thermal precipitator at 2 amp., our experience has been that, providing the wire is heated only when the metal plugs are in position, the maximum damage to the bakelite facing pieces is a slight charring or blackening round the wire which does not affect the performance of the instrument in any way.

In reply to the remarks by the President about the big variation in percentage reduction of dust counts after acid treatment, I should like to point out that this variation is due mainly to variation in the percentage of acid-soluble particles in the air derived from the atomization of water by the machine drills and to a lesser

* *Bull.* 499, June, 1948.

extent from other sources. The big loss of mineral particles from konimeter slides on acid treatment which was noticed by the President on his visit to the Rand in 1933 is due to the fact that the slides were acid-treated before ignition, and this is known to result in the loss of about half the mineral particles when the correct thickness of jelly is used on the slides. Thermal precipitator samples and konimeter samples which are ignited *before* acid treatment lose much less of the insoluble material, the particles being very much more firmly attached to the glass surface, but the percentage lost increases rapidly with increase in particle size. Experiments carried out in this department indicated that the average overall loss of rock particles from typical mine dust samples is about 10 per cent, but with particles between 2 and 5 microns in diameter the loss is very much greater, of the order of 50 per cent.

Finally, I should like to thank all those who took part in this discussion for their encouraging and stimulating remarks.

AUTHOR'S REPLY TO DISCUSSION* ON

The Sinking of No. 5 Shaft, Van Dyk Consolidated Mines, Ltd.

By T. L. Blunt

Mr. T. L. Blunt: The following remarks refer to the points raised by various contributors to the discussion.

Mr. Jack Spalding: We later found there was no necessity to keep the ends in advance of the rest of the course.

At the end of the sinking the water was measured and found to be 4,800 g.p.h. By June, 1948, the measured figure was 1,490 g.p.h., and at the present time (March, 1949) the figure is 1,300 g.p.h., and the walls for the first 3,000 ft. are quite dry.

The fan was a 48-in. Sirocco type built by Brown of Springs, Transvaal.

With reference to the dividing wall, it is my opinion that extra time would be taken if it were cast during the sinking and inasmuch as man is averse to change, even at work, placing of shuttering or precast beams would take longer per foot when interposing it with other work than if it were done as one job from the bottom to the top. The throwing of concrete would also take longer with two separate compartments. Thus, since I think it would take longer per foot during sinking, the shaft would have to carry all costs including the sinking crew during this period, whereas doing the job at the end of the sinking meant that the cost of the equippers and winding drivers only had to be paid.

I think that had we carried the central wall down with us it would have taken at least 6 months longer to sink the shaft at an overall cost of £15,000 per month, whereas we estimate to complete the wall in 2½ months after the shaft is down, at an overall cost of £3,000 per month (both figures exclude the cost of the wall). The fan pipes are put in by the stage crew while mucking is in progress.

Prof. Ritson: The shaft is intended to hoist up to 150,000 tons per month and to pass 600,000 c.f.m. of air.

With regard to ropes, a copy of the makers' specification and the Government test laboratory certificate for breaking load are shown overleaf. Ropes in South Africa have to be cut every six months and the bottom end submitted to the laboratory for examination and destruction tests. A report similar to that shown is forwarded to the mine for each test. With 14-ft. drums there were 2¾ layers of rope on the drum when the bucket was at the surface. To have one layer only would require a drum approximately 12½ ft. wide. The twist on the inner strands of the rope and that on the outer

* *Bull.* 506, January, 1949.

strands were in opposite directions. The wire cores did not break into short lengths. This did happen on triangular wire and triangular core ropes, but it does not with fabricated six-wire cores. Extra special improved plough steel, of 128/140 short tons breaking stress for wires, is in common use in South Africa. It would be interesting to learn if this is so elsewhere. Rope regulations are reasonably severe here, and ropes have to be discarded when the factor of safety becomes 6 : 1. Sheave wheel treads were correctly fitted and aligned.

Mr. J. B. Richardson : The clam shell for cleaning the shaft bottom was built to be operated by air, but during my absence a conversion to electricity took place. The experimental work was done on surface and it was found that the ordinary clam shell did not dig into the Rand quartzites, hence a modification had to be made in the clam shell to obtain a 30° digging angle, which is the optimum digging angle of scoops for these quartzites.

Mr. H. C. T. Brown : Quartzites and shales on the Witwatersrand are of such strength that the lining of a shaft is mainly to support buntons and bearers. There is no necessity to line a shaft for

---

SPECIMEN :

HAGGIE, SON & LOVE (1936), LTD.
Cleveland, Tvl. *27th February, 1947.*

SPECIFICATION OF 'JUPITER' STEEL WIRE ROPE

Supplied *VAN DYK CONSOLIDATED MINES, LIMITED*
Order No. *V.D. 1005/Job H.208 of 3.12.1946*
Coil No. *158126*
W. Order No. *53408*

DIMENSIONS OF ROPE— *NON-SPIN*
Length *5,000 feet*
Diam. *1½ in.*
Circum. *4¾ in.*

CONSTRUCTION OF ROPE—
Lay *LANGS*
No. of Strands *15 (9 over 6)*
Class of Core *Wire Main Core*

CONSTRUCTION OF STRANDS— *Specially Lubricated*
No. of Wires *10 (8 over 2) outer* *10 (8 over 2) inner*
Diam. of Wires (Decimals of an inch) *·096 ·101 outer, ·080 ·084 inner*
Class of Core *2 Steel Wires*

CLASS OF STEEL OF WHICH WIRE IS MADE *EXTRA SPECIAL IMPROVED PLOUGH—Iscor Special*

BREAKING STRESS OF STEEL *128/140* Tons (2,000 lb.)

WEIGHT OF ROPE PER FOOT *4·295* lbs.

ACTUAL BREAKING LOAD OF ROPE AS PER GOVERNMENT TEST CERTIFICATE No. *77998* ATTACHED *123·6* Tons (2,000 lb.)

DATE OF MANUFACTURE *11th February, 1947*

DELIVERED *Mine*

To *The Resident Engineer,*
Van Dyk Consolidated Mines, Limited,
P. O. VAN DYKMYN.

SPECIMEN :

M.D. 203.

Appl. Received : *13.2.47* Certificate No. *77998*
Rope Received : *13.2.47* Date of Test : *14.2.47*

DEPARTMENT OF MINES

CERTIFICATE OF TEST CONDUCTED IN THE MECHANICAL LABORATORY

Specimen Supplied by : *HAGGIE, SON & LOVE LTD.*
Shaft : —
Compartment : —
Coil Number : *158126* Original Length of Rope : *5,000 ft.*
Name of Manufacturer : *Haggie, Son & Love Ltd.*
Date of Manufacture : *11.2.47* Date Rope was put on :
Weight of Rope per Ft.: *4·295 lb.* Class of Steel used for the Wires : *Extra Spec. Impr. Plough*
Tensile Strength of the Steel Wire used : *128/140* Tons (2,000 lb.) p.s.i.
Original Breaking Load : — Tons (2,000 lb.)
Breaking Load at Last Test : — Tons (2,000 lb.)

PARTICULARS OF SPECIMEN

	As supplied by Applicant	From Examination
Diameter ...	*1½ in.*	*1·50 in.*
Construction	*Non-Spin*	*Non-Spin*
Kind of Lay and R. or L.H.	*Lang's*	*Lang's*
Length of Lay		*13 in.*
Number of Strands	*15 (9/6)*	*9/6*
No. of Wires in Strand...	*10 (8/2) 10 (8/2)*	*8/2 — 8/2*
Diameter of all Wire used (unworn)	*·096 in. ·101 in. ·080 in. ·084 in.*	*·096 in. ·101 in. ·080 in. ·084 in.*
Class of Heart of Rope...	*Wire main core (1 × 30)*	*Wire main core (1 × 30)*
Class of Strand Core	—	—

Length of Specimen Supplied by Applicant : *11 ft.*
Length of Test Specimen : *9 ft.*

RESULT OF TEST AND EXAMINATION.

Breaking Load of Rope : *123·6* Tons (2,000 lb.)

CONDITION OF ROPE.

Least Dimension of Most Worn Outer Wires : *New rope*
Corrosion : *None*
Lubrication : *Good*
Appearance at Fracture of Wires : *Ductile*
Number of Strands Broken : *All except 2 outer strands*
Approximate Position of Fracture : *4 ft. from metal*
Remarks :
Date : *16th February, 1947.*

GOVERNMENT MECHANICAL LABORATORY,
COTTESLOE,
JOHANNESBURG. Testing Officer. Director.

strength as, apart from scaling and isolated cases of weathering dykes, no trouble is experienced with crumbling shaft walls. Where Witwatersrand series is overlain by younger rocks of the Karroo system, such as dolomites, Dwyka conglomerate, etc., care has to be, and is, taken with concrete in order to obtain the required strength, and more often than not the concrete is reinforced. Inasmuch as this shaft went into quartzite at 120 ft. and remained in this to its final depth, strength to resist pressure was not required. However, since it is as easy to place good concrete as poor concrete the quantities in the mix were as follows—cement 3½ bags, sharp washed sand 12¾ cu. ft., aggregate 25¼ cu. ft., water 26 gal., with a wash of 1 gal. to run the concrete down the chute from the mixers to the bottom discharge Blaw Knox bucket. Slump of a maximum of 5½ in. from the mixer was aimed at. Inasmuch as aggregate is obtained by mechanical screening (in our case a Tyrock screen) variation from sand to 1½-in. rock was obtained. Twenty to twenty-five natives were employed on the stage for placing bricks and while concrete was being poured nothing pleased these boys more than to have charging sticks in their hands and to ram the concrete at all times. It would be almost impossible to stop these natives ramming concrete, as they are like children in their mental outlook.

The octopus was set 19 ft. above the top deck of the stage on the north pair of stage ropes (*vide* 'Procedure'). As the concrete came through the rubber hoses of the octopus, it struck the side wall before reaching the actual depositing position, due to the inclination of the hoses. This and the ramming was sufficient to remix the segregated constituents properly.

Circular shafts lined with bricks with concrete poured behind them have been in operation on the Rand for 30 years. These are up to 22 ft. in diameter, some being equipped with guide ropes and some with buntons and guides. No trouble has been experienced with them and therefore it can safely be anticipated that no trouble should be experienced in this shaft. In these brick-lined circular shafts no undue difficulties have been experienced even after the shaft pillar has been extracted.*

Comparisons can be made here with two other elliptical shafts, one with steel shuttering bolted together and the other with a sliding steel shuttering pulled up by turn buckles on chains anchored to eye bolts let into the walls of the shaft above the lift of concrete being placed. Neither of these attained the speed of No. 5 Shaft, and therefore their costs must automatically be greater. Concrete was placed round the periphery of these by a sheet steel portable launder, which is also inclined to give a classification of the concrete.

The volume of ground to be excavated would not be affected by the method of placing the concrete, but rather by the type of

* *See* DUGGAN, G. H. Extraction of shaft pillar at No. 1 Circular Shaft, New Modderfontein Gold Mining Co., Ltd. *Ass. Mine Mgrs. Transvaal Papers* 1937–38, p. 163.

ground and the care with which the sinkers direct their cropper holes. Overbreak was kept to a minimum; 85·8s. per foot was the cost of the bricks, and it was attempted to cut the excavation to 12 in. from the face of the bricks. In good ground this and better was obtained, but if faulting, bad slips or poor ground were encountered overbreak might be as high as 3 or 4 ft., when quantities of plums were brought down and placed in these sections.

The mucking machine worked originally on compressed air, but as compressed air motors were practically unobtainable in the requisite sizes, electric power was substituted. The lower end does rest on the spoil and when undercut it only requires a small lift on the surface hoist to replace it in correct position. It is suspended suitably for conditions at all times by its own hoisting rope and hoist. The driver has a sheet steel cover over himself and the three hoists for operating the grab. This was not shown in the drawing as it was folded back when the draughtsman put the machine on paper.

AUTHOR'S REPLY TO DISCUSSION* ON

A Note on 'Steel' Galena

By G. A. Schnellmann, *Associate*

Dr. G. A. Schnellmann: This note was a record of an investigation which had not been—and because of changed circumstances was unlikely to be—completed. It was submitted for publication on the principle that a partial record is better than no record and can conceivably be of use to others. These remarks are by way of explaining my inability to give conclusive answers to some of the questions asked, particularly by Mr. Eastwood.

Microscopic examination was restricted to polished surfaces of steel galena, and no strain phenomena (by which I take Mr. Eastwood to mean anomalous polarization) were observed. Barytes, pyrite and quartz are for practical purposes unknown in the Halkyn ores. Steel galena there occurs typically in a clay gouge, and I cannot recollect having seen it with gangue minerals. It was for this reason that Dr. Smith's account of the Sipton material was of particular interest. Contrary to Mr. Eastwood's belief, comparative analyses of 'steel' and common galena from the Halkyn veins do show the steel variety to be generally richer in silver, but this is not universally true, as Dr. Williams notes. Mr. Eastwood's reference to silver values in galena from the Lake District veins is well illustrated in the Greenside mine, where the silver content definitely seems to be falling off in depth—i.e., with increasing distance from the barytic part of the lode. Having made no systematic field or laboratory study of the phenomenon, I am diffident about offering an explanation, but suggest two possible reasons: (*a*) *Primary*; the phase of barytes deposition may represent conditions which also favour the precipitation of silver minerals: (*b*) *Secondary*; the exposed barytes zones are quite shallow, not extending more than a very few hundred feet below present-day surface, which suggests the possibility of supergene enrichment.

The data quoted by Dr. Williams on the synthesis of material resembling steel galena, and on the field associations of this material at Sonora, Mexico, strongly support my conclusions, and I am duly grateful for them. I am glad also to have his confirmation of the absence of discrete silver minerals, and his evidence that these are in solid solution in the galena. While I have always tacitly assumed this to be so in general, it is worth noting that in the shallower lodes of the Halkyn district Finlayson found specimens of galena in which silver minerals occurred along the cleavage planes.

* *Bull.* 507, February, 1949.

There is definite evidence suggesting the migration of silver in the way suggested by Dr. Williams, e.g.—

Common galena	*'Steel' galena*	*Remarks*
4·65	5·93	Two varieties contiguous
4·80	7·35	ditto
3·00	7·50	Specimens from different parts of the same lode.
2·25	9·30	ditto
4·60	7·80	ditto

(Figures represent oz. silver per ton of galena.)

I know of only one case in which the two varieties had the same content, 6·15, and one in which the steel galena had a lower content, 3·60 as against 4·25. In neither of these cases were the two varieties contiguous, though each pair was from a single lode.

The gangue minerals in the photomicrographs are certainly carbonates, and I agree that an examination of the kind suggested might be revealing, but here again the work was not carried to completion.

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NOTICE OF GENERAL MEETING

The Eighth Ordinary General Meeting of the Fifty-Eighth Session of the Institution of Mining and Metallurgy will be held, by kind permission, in the Apartments of the Geological Society, Burlington House, Piccadilly, London, W. 1, on **Thursday, 16th June, 1949,** at 5 o'clock p.m.

The Meeting will be devoted to a discussion of practical postgraduate training, which will be opened by Professor J. A. S. Ritson, O.B.E., D.S.O., M.C., *Member.* Notes on the subject for discussion have been prepared by Professor Ritson and are printed on pp. 1–7 of this *Bulletin*. It is hoped that a large number of members will attend and express their views.

Light refreshments will be provided at 4.15 p.m. for members and visitors attending the Meeting.

The Council invite written contributions to the discussion from members who may be unable to be present at the Meeting. The Council reserve the right to edit and condense such contributions.

THE SIR JULIUS WERNHER MEMORIAL LECTURE

As announced last month, Dr. C. H. Desch, F.R.S., will deliver the second Sir Julius Wernher Memorial Lecture of the Institution on Wednesday, 6th July, at 5 p.m., at *The Royal Institution*, 21, *Albemarle Street, London, W.* 1. He will speak on **The Effect of Impurities on the Properties of Metals.**

Admission to the Lecture is free, without ticket, and visitors will be welcomed. Tea and light refreshments will be served from 4.30 p.m.

THE REFINING OF NON-FERROUS METALS

The Symposium on the Refining of Non-Ferrous Metals will be held on Thursday and Friday, 7th and 8th July, from 10 a.m. to 5 p.m. on each day, at *The Royal Institution of Chartered Surveyors*, 14, *Great George Street, Westminster, S.W.* 1.

Full particulars of the papers to be submitted have already been circulated, and a programme of the proceedings will be sent as soon as possible to those who have applied for tickets of admission.

INSTITUTION NOTES

Council of the Institution

The ballot for Members of Council for the Session 1949–50 took place at the Annual General Meeting of the Institution held on 19th May, 1949. The new Council is as follows:

President: Mr. W. A. C. Newman.

Hon. Treasurer: Mr. Robert Annan.

Vice-Presidents: Messrs. A. L. Butler, Tom Eastwood, Donald Gill, Vernon Harbord, L. C. Hill, and Sir Arthur Smout.

Members of Council: Messrs. J. C. Allan, G. Keith Allen, A. T. Climas (*West Africa*), Prof. C. W. Dannatt, Messrs. J. B. Dennison, R. W. Diamond (*Canada*), Sir Lewis L. Fermor, Sir Paul Gueterbock, Mr. H. R. Holmes, Dr. N. R. Junner, Messrs. E. G. Lawford, C. O. Lindberg (*U.S.A.*), E. A. Loring, R. S. Mackilligin, J. D. Mead (*Malaya*), R. G. K. Morrison (*India*), Geoffrey Musgrave (*Rhodesia*), J. B. Richardson, Kenneth Richardson (*South Africa*), Prof. J. A. S. Ritson, Messrs. Stanley Robson, R. H. Skelton, Sir Edmund O. Teale, Dr. J. H. Watson, Messrs. G. A. Whitworth, A. R. O. Williams, Dr. David Williams, and Mr. O. H. Woodward (*Australia*).

Ex-Officio Members of Council (Past-Presidents): Messrs. J. Allen Howe and C. F. Laycock, Prof. W. R. Jones, and Mr. S. E. Taylor.

Election of Members of Council for the Session 1950-51

As previously announced, the new Bye-laws governing the constitution of the Council and its mode of election will affect nominations for the election of Council for the session 1950–51. Nominations for the election of Ordinary Members of Council [*see* Bye-law 27 (iv)] and Overseas Members of Council [*see* Bye-law 28 (iii)] should be sent to the Secretary of the Institution to reach him not later than 1st November, 1949.

Annual General Meeting

The 58th Annual General Meeting of the Institution was held on Thursday, 19th May, 1949, at Burlington House, when Mr. W. A. C. Newman delivered his Presidential Address on 'The role of the Institution in present-day educational developments'. The Address and a full report of the Meeting will be published in the July issue of the *Bulletin*.

Army Officers

A leaflet has been issued by the War Office, entitled 'The Reserves of Officers and How to Join Them'. It sets out the opportunities for released officers to join the Regular Army Reserve of Officers (R.A.R.O.) or the Army Officers Emergency

Reserve (A.O.E.R.) in addition to describing the separate arrangements for the Supplementary Reserve of Officers and Territorial Army Reserve of Officers (T.A.R.O.). The invitation to join is at present extended only to officers who have held commissions, other than Supplementary Reserve or Territorial Army commissions, on full pay. Ex-officer members of the Institution may be interested to read the leaflet, copies of which may be obtained from the War Office, London, S.W. 1.

Candidates for Admission

The Council welcome communications to assist them in deciding whether the qualifications of candidates for admission into the Institution fulfil the requirements of the Bye-laws. The application forms of candidates (other than those for Studentship) will be open for inspection at the office of the Institution for a period of at least two months from the date of the Bulletin *in which their applications are announced.*

The following have applied for transfer since 12th May, 1949 :

To Membership—

Ben Lightfoot (*Maidenhead, Berkshire*).

Frank Pinney Longmire (*Wareham, Dorset*).

To Associate Membership—

John Anthony Desmond Bell (*Prestea, Gold Coast Colony*).

Anthony Vernon Bradshaw (*Alemtejo, Portugal*).

John Henry Knapp (*Bukuru, Northern Nigeria*).

David Ronald Mitchell (*Crowborough, Sussex*).

The following have applied for admission since 12th May, 1949 :

To Associate Membership—

Ernest Thomas Edward Andrews (*Barberton, Transvaal*).

Syed Kazim (*Saifabad, Hyderabad, India*).

Victor Edwin Peterson (*Mufulira, Northern Rhodesia*).

P. N. Vijaya Raghavan (*Oorgaum, Mysore State, India*).

To Studentship—

David John Ivor Evans (*Warlingham, Surrey*).

William Bernard Hall (*N'kana, Northern Rhodesia*).

Giles Freathey Oats (*Hove, Sussex*).

Kenneth Bernard Platt (*Camborne, Cornwall*).

News of Members

Members, Associate Members and Students are invited to supply the Secretary with personal news for publication under this heading.

Mr. J. R. Beech, *Student*, has left England for the Gold Coast.

Mr. C. W. Cayzer, *Member*, has been appointed general manager of the Emperor Gold Mining Co., Ltd., Fiji.

Mr. C. O. Champion, *Student*, is returning to England on leave from Nigeria.

Mr. W. W. Connor, *Associate Member*, has returned from Australia and has joined the staff of the Zinc Corporation, Ltd., in England.

Mr. L. A. Crozier, *Associate Member*, has left Bolivia on his appointment as mine superintendent, Loloma (Fiji) Gold Mines, N.L.

Mr. C. B. Curtis, *Student*, has returned to England from the Gold Coast.

Mr. John Davey, *Member*, has arrived in England from Chile.

Mr. G. A. W. Dove, *Student*, has returned to England from British Guiana.

Mr. S. J. Earl, *Student*, has left England for the Gold Coast.

Mr. E. F. Elkan, *Member*, has returned to Malaya.

Mr. E. A. Fisher, *Associate Member*, has arrived in England on leave from Malaya.

Mr. F. H. Fitch, *Associate Member*, is returning to England on leave from Malaya.

Mr. C. C. Freeman, *Member*, has arrived in England from Adelaide and will be here until October.

Mr. S. F. Gandar, *Student*, has left England for Northern Rhodesia.

Mr. A. I. George, *Associate Member*, has been appointed general mines manager of the Ashanti Goldfields Corporation, Ltd.

Mr. Donald Gill, *M.C.*, *Member*, has left England on a visit to Nigeria.

Mr. J. H. Hohnen, *Associate Member*, has been appointed field manager of New Guinea Goldfields, Ltd.

Mr. W. P. Horne, *Associate Member*, is returning to England on leave from Kenya.

Mr. R. E. W. Hughes, *Member*, has left Johannesburg and is now in South West Africa.

Mr. L. G. Hutchinson, *Member*, has left England for the Gold Coast.

Mr. H. D. M. Jager, ***Associate Member,*** **who was formerly in India, is now in Vancouver, B.C.**

Mr. A. L. Job, ***Associate Member,*** **has left Ipoh and is now in Selangor, Malaya.**

Mr. G. R. Jones, ***Associate Member,*** **has returned to England from India.**

Mr. B. S. Lamba, ***Associate Member,*** **has been appointed Deputy Director, Indian Bureau of Mines, Delhi.**

Mr. J. K. Lindsay, ***Member,*** **has arrived in England on leave from India.**

Mr. C. B. Loubser, ***Associate Member,*** **has left Johannesburg and is now employed by Groothoek Chrome Mines, North-eastern Transvaal.**

Mr. D. A. Mackay, ***Member,*** **is returning to England from Malaya.**

Mr. G. F. Mead, ***Associate Member,*** **has left Western Australia to take up an appointment in Melbourne.**

Mr. D. R. Mitchell, ***Student,*** **has arrived in England on leave from Malaya.**

Mr. L. E. T. Parker, ***Associate Member,*** **has arrived in England on leave from Iran.**

Mr. H. Pascoe, ***Associate Member,*** **has returned to England on leave from South Africa.**

Mr. R. A. Purvis, ***Member,*** **is arriving in England in June on leave from the Gold Coast.**

Mr. R. H. Skelton, ***Member,*** **has left England on a visit to Ceylon.**

Mr. A. Spargo, ***Associate Member,*** **has returned to England on leave from Nigeria.**

Mr. D. V. G. Tregaskis, ***Student,*** **has arrived in England from Nigeria.**

Mr. J. K. Walker, ***Associate Member,*** **has arrived in England on leave from India.**

Mr. H. B. Watson, ***Associate Member,*** **has joined the staff of the Ministry of Fuel and Power.**

Mr. P. N. White, ***Associate Member,*** **has left Northern Rhodesia for England on leave.**

Mr. L. Wilton, ***Student,*** **has left England for India.**

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Notes on Mining Education and Postgraduate Training

By J. A. S. Ritson, O.B.E., D.S.O., M.C., *Member*

It was not the writer's original intention to write a paper but, with the permission of the President and Council, to open a debate on this subject. Subsequently it was suggested that the discussion would draw contributions from overseas members if a short summary were published in the *Bulletin* prior to the debate.

The following notes are the writer's own ideas and nobody realizes better than he that they will not meet with universal approval. They are written to draw readers into expressing their own views as vehemently as they desire. From the resultant clash of opinion some extremely useful ideas should emerge.

During the past 25 years the practice of mining engineering has altered. Manual labour has been superseded largely by machines, and rule of thumb methods are becoming extinct. This has led to the institution of a number of specialized sub-departments, which renders it difficult for a young man to obtain a general training in practical mining.

This change is partly due to the readily accessible, rich and easily-worked deposits becoming depleted and the development with huge outputs of less favourably placed mines.

There is a tendency for a young man to specialize in one phase of a modern mines organization and thus fail, while he is young, to obtain a broad general picture of the whole operation.

The training of a young mining engineer can be divided into two parts—theoretical and practical. The theory part is the duty of the schools, while industry must take care of the practical.

A young mining engineer must have a sound training in the fundamental and applied sciences together with a practical knowledge of mining machinery and mining methods. Equally important is experience in the art of handling men—or 'labour relations' as it is now called. All this has to be related to the economics of the situation because whatever mineral is being worked the engineer is really mining 'money'—that is, extracting the mineral so that there is the greatest possible difference between what it costs to get and what can be got for it in the open market; not for a year or two but over the life of the mine. Further, this 'money' must be obtained with the least possible risk to life and limb.

The necessary training cannot be obtained wholly at a mining school or at a mine; the two are complementary to each other. The mining schools and the mining industry must be partners in this enterprise. The graduate, immediately he leaves college, does not become overnight a mining engineer. The soil has been well dug and good seed sown, but the care of the seedlings belongs to the industry if a satisfactory harvest is to be reaped. To use another analogy, the college is the foundry where sound castings are made and are subsequently machined and polished in the post-graduate university of life and experience.

Mining is a tough business and a good mining engineer must be able to get on well with his fellows. The ability to suffer fools gladly but not too gladly is essential. A young man must be encouraged to take responsibility and should not be dealt with too hardly if he makes a mistake.

It follows that from the time a young man enters a mining school until he becomes a fully qualified mining engineer his training must be systematic and supervised. His work at the school and his early years at the mine must be properly integrated.

Industry must recognize—as indeed it does in many cases—that a young man immediately after graduating is *not* a mining engineer. In no engineering or learned profession is a newly-hatched graduate considered fit to take charge of responsible work. In fact, in every profession—be it medicine, civil engineering, law or accountancy—a young man has to spend from five to seven years, including the university period, as a pupil before he is eligible for responsible employment. A mining engineer should therefore have at least five years' training before he can be considered qualified. This is especially true in highly mechanized mines. As in all other professions the responsibility for training should be divided between the university and the employer.

The days are rapidly disappearing when a boy is apprenticed to a trade and receives only a nominal salary while serving his articles. To-day a young graduate expects to be paid in his first job and paid handsomely too. This is not, in my opinion, always to his ultimate advantage, because an employer, quite understandably, wants a dividend on an outlay of £40 to £50 a month. Consequently, the young man is often set to perform useful tasks, well within his compass, but fails to gain general experience.

The job of the mining school is to give a budding engineer a sound knowledge of the principles of his future profession. It is not the function of a school to turn out a trained engineer. No school can do this and it is a waste of energy to try. On the other hand, a school can and should so train its students that they can earn their keep in one or other of the ancilliary jobs in mining.

A graduate of a good school should be able to take his place at once on the survey staff of a mine, or in the assay office. These are bread-and-butter subjects but they do not often lead to a manager's job *if persisted in for too long.*

The making of a mine manager is the industry's job and under to-day's conditions it is going to cost money. Too often when a keen youngster joins a mine, the manager or a member of his staff finds his knowledge useful and places him into a job where he may stay put—such as collecting dust samples, routine surveys, measuring air velocities, etc. He is earning a profit for the company but is not learning to be a mining engineer. Many an excellent young graduate joins a mine and seeks another occupation at the end of his contract solely because he considers he is being directed into a narrow groove and is not getting a real opportunity to obtain a detailed knowledge of the art of mining.

University Training

A University course should consist of three post-intermediate years. During the intermediate period which follows matriculation or matriculation exemption a young man should be thoroughly grounded in the fundamental sciences—chemistry, physics, mathematics, including geometry. If the timetables allowed a course in English or one of the Arts it would be most useful, because an engineer ought to have a broad general education as well as be a good technician.

The first post-intermediate year should be devoted to geology, applied mechanics (solids and fluids), applied mathematics, and applied physics (heat, electricity and magnetism). These form an introduction to the technical subjects of the two last years which include principles underlying the practice of mining, machine design and mining machinery ; economics as applied to mining ; elementary metallurgy and ore dressing ; while a more detailed training in surveying and assaying should enable a graduate to make these branches of mining his profession if he so wishes.

Postgraduate Training

A graduate, on leaving College to take up his first job, is usually keen and willing to do anything. He realizes he is lacking in practical experience of the various jobs about a mine, many of them manual, and is ready to learn. It is in his own interest and that of his employers ultimately if this deficiency is made good, and it is suggested that a two-year course of systematic practical training should be given to every new entrant of the graduate class. Two years' hard physical work can do no harm to any young man and the advantages accruing are manifold.

Before suggesting in detail such a scheme it is advocated very strongly that a young graduate on arrival at the mine should be recognized as a potential official, i.e., like a member of an O.C.T.U., and be granted and retain the privilege of an official. If, during this training period, he does day-wage work he should not be paid day-wages and be graded as a day-wage man. This is a source of annoyance which causes some men to leave their company.

Every mine of any size ought to have a graduate training scheme into which every 'tenderfoot' should enter. There are such schemes in operation at many mines or individual mines of a large group. Such as really operate are excellent, but in many cases they operate on paper only. Time and time again the writer has discussed with a young man why he has thrown up his job and has received the answer, 'Well, I expected to get two years' practical experience doing all sorts of jobs but I have spent the whole time surveying (or sampling or measuring the ventilation)'. This is one reason why many young men are going to small mines; they think that in such mines they will have a chance of getting more diverse experience.

Before a coal mining graduate can obtain his first class Certificate of Competency, i.e., Colliery Manager's Certificate, he has to spend at least 18 months doing manual work in or about the mine and another 18 months in direct supervision of such work. In practice he does more manual than supervisory work. The manual work is varied and should cover the whole range of operations in a colliery. Such a diet can do no harm to a potential metal mine manager. Where this idea is adopted, a young engineer learns the details of the simple manual tasks and mechanical devices used in a mine. Too long a period spent on one specific detail is a waste of time. Every young graduate is a potential manager and not a drill runner.

A difficulty may arise when all labouring work is carried out by non-European labour but this can be overcome by having a training sector—it need only be quite small—in a central mine of a group or corporation where all trainees can learn the rudiments.

The young man also learns by personal experience what work an average man can do in a day. He gets closer to understanding the meaning of human relations than he would if he was always a 'boss'; he sees at first-hand the work and dodges of a junior official such as a shift boss or a miner where native labour is employed; and he enriches his mining vocabulary with words both good and hearty.

He knows what a good track looks like and how it is laid. He understands the value of a good piece of timbering and can fix a set himself. He learns how to distinguish safe ground from dangerous ground, how and where not to site and drill a shot-hole. and many other things besides. It may be argued that he can learn all this by watching others, but not so well as if he has done it himself. Besides, if a man can do the job himself it gives him confidence whilst supervising other men.

It is not suggested he need spend many months on any job. Rather should he spend a few weeks on each of many jobs. His school of mines training will enable him to pick up the essential point of any operation very quickly.

So that something concrete may be offered, Table I is an attempt to devise the kind of manual and technical training or experience every graduate should undergo when first employed by a mining company after graduation.

TABLE I

	Weeks
Examining surface layout	1
Travelling mine with underground official	2
Track laying	2
Tramming and loading	2
Timbering and road repairs	8
Shaft maintenance	2
Ventilation measurement	4
Dust sampling	4
Erecting ventilation tubes, etc.	2
Main haulage	2
Shaft station	1
Pipe laying	1
Making pump or engine houses	2
Running scraper loaders, etc.	2
Development	8
Stoping	8
Winzing or raising	4
Shaft sinking	2
Underground surveying	12
Sampling	4
Assay office	2
Mechanical engineer shops	4
Electrical engineer shops...	4
Compound management	2
Wages accounts	2
Planning department	4
Stores department	2
Mill	4
Leave	8
	105

(Any time spent on these tasks before graduating should count.)

In the Appendix are included details of schemes in operation at various mines or groups of mines in South Africa.

If any of these schemes are strictly adhered to then a young graduate will become a mining engineer in due course. He will stay in the industry because he knows he is being looked after and will not be anxious to transfer to such other engineering professions as, for instance, public works.

APPENDIX

A. Witwatersrand: Scheme 1

	Weeks
Sampling	6
Surveying	4
Ventilation and dust sampling	4
Study department	6
General mining (stoping, development) / Drive timbering, pipes and tracks / Shaft timbering	17
Reduction works	6
White and native time offices (1 week in each)	2
Native compound	2
Main store and timber yard	1
Assay office	1
Rock drill fitting	2
Drill sharpening	1
Total	52

B. Witwatersrand: Scheme 2

	Months
Sampling	6
Surveying	6
Stoping and development (under supervision of a miner)	3
Pipe laying and track laying	1
Engineering shops	1
Ventilation	1
Timbering, including shafts	2
Haulage, including vertical shafts	2
Study department	1
2 annual leaves of 1 month each	2

Thereafter a graduate would probably be given the following additional training:

Compound and native control	1
Stoping and development (on his own)	5
Economy department	1
Study department	6
Leave	1

C. Northern Rhodesia

(This is a composite scheme made up from information received from several mines.)

	Months
Sampling	3
Surveying	3
Ventilation	3
Mine planning	3

	Month
Underground work—	
Lashing	½
Timbering	½
Development	½
Shaft sinking...	½
Tramming	1
Scraping	1
Pipe fitting	1
Diamond drill	2
General stoping	5

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THE INSTITUTION OF MINING AND METALLURGY

A Review of Pneumokoniosis and Dust Suppression in Mines

Part I—Medical Aspects of Pneumokoniosis*

By A. G. Heppleston, M.B., M.R.C.P.†

Introduction

Chronic respiratory disease is known to have afflicted the workers in dusty trades from prehistoric times (Rosen, 1943) and the desirability of avoiding the inhalation of mine dust was appreciated by the Romans (Pliny), but it remained for Agricola (*De re metallica*, 1556) to associate clearly the inhalation of dust with the occurrence of pulmonary disease in miners, especially where the mines were dry. Following the Roman practice he advocated the use of a simple respirator as a preventative measure. Since those early times the idea that dust caused pulmonary disease has gradually gained general acceptance, more particularly since Zenker (1867) demonstrated beyond doubt that atmospheric dust could be deposited in the depths of the lungs.

Miners' sickness, as this respiratory disease was originally called, bore certain clinical resemblances to pulmonary tuberculosis, such as destruction of the lungs and emaciation (Agricola, ibid.). Many of the earlier workers, however, were careful to distinguish the two conditions. Although detailed studies of the pathology of miners' lungs are relatively recent, the question of the relationship of dust accumulation to tuberculosis was raised in some of the earlier investigations (e.g. Graham, 1834 ; Cox, 1857 ; Peacock, 1860–61 ; Begbie, 1866). Proof of such an association was wanting for many years owing to lack of technical methods, but with their advent it became clear that tuberculosis, in one form or another, was an important factor in the production of pulmonary disease in individuals exposed to certain occupational dust hazards.

The magnitude of the problem of pneumokoniosis as it exists to-day can be judged from the following facts. In South Wales approximately 18,000 coal workers have been disabled by pneumokoniosis during the past 17 years (Jenkins, 1948), whilst for the years 1941–44 the incidence of new cases of silicosis, with and without tuberculosis, in European miners on the Witwatersrand was 9·53 per 1,000 examined (Smith, 1941–44). Nevertheless, much

* This review discusses the present state of medical knowledge and has been prepared at the request of the Joint Committee of The Institution of Mining Engineers and The Institution of Mining and Metallurgy, by whom the Conference on Silicosis, Pneumokoniosis and Dust Suppression in Mines, London, April, 1947, was organized. At a later date it is hoped to publish reviews of the mining and engineering aspects of the problem.

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has been done to prevent the disease, as the corresponding incidence on the Rand for 1917–20 shows, namely 28·11 cases per 1,000 examined (ibid).

Two forms of pneumokoniosis can be distinguished, a simple type due to dust acting alone and an infected type due to the combined effect of dust and infection, particularly tuberculosis (e.g. Strachan, 1947 ; Gough, 1947 ; Caplan, 1947 ; Craw, 1947).

Simple Pneumokoniosis

For a dust to produce disease a sufficient amount must collect in the lungs, what Mavrogordato (1926) called the effective occupation of the lungs by dust. This accumulation depends on several factors, and of these the size of the particles reaching the substance of the lungs appears to be the most important. It is generally agreed that particles above a size of 5μ are rarely found in pneumokoniotic lungs (McCrae, 1913 ; Drinker, 1925 ; Sayers and Jones, 1937) and accordingly only those dusts which exist in the atmosphere in this finely divided state can be considered as potential causes of pulmonary disease. Subsidiary factors such as the concentration of and the duration of exposure to air-borne dust are self-evident, but the part played by individual susceptibility to the accumulation of dust in the lungs is less well known. The efficiency of the pulmonary mechanisms for eliminating inhaled particulate matter seems to vary from one individual to another since under fairly comparable conditions there are often considerable variations in the amounts of dust which different persons collect in their lungs. Of these individual variants nasal filtration (Lehmann, 1933), bronchial movement and secretion, ciliary action, phagocytosis and previous damage to the lung may be mentioned. There may even be a selective action by the lungs on certain types of dust (Fenn, 1920–21, 1922–23 ; Policard, 1938). The effect of deep breathing due to physical exertion whilst exposed to a dusty atmosphere is difficult to assess. Although it might be anticipated that under these conditions more dust would be inhaled, it is quite possible that more would also be eliminated on exhalation. This contention receives some support from the common observation that less dust tends to collect in the lower parts of the lungs, which are usually considered to be better ventilated than the upper parts.

The precise pathological changes produced by dust as it occupies the lungs depend upon the constitution of the dust. When the inhaled dust contains a large proportion of free silica hard, fibrous, ‘ silicotic ’ nodules, measuring up to ½ cm. in diameter, are produced (Strachan and Simson, 1930 ; Simson, Strachan and Irvine, 1931 ; Simson and Strachan, 1935), but with dusts containing only a small amount of free silica the lesions are softer, being only slightly fibrous, and often deeply pigmented (Gough, 1947 ; Craw, 1947). In coal workers the dust lesions consist essentially of numerous black spots, up to ½ cm. diameter, evenly scattered throughout the lungs. The air spaces surrounding these lesions are often onsiderably enlarged and this focal emphysema is much more evere than in classical silicosis (Badham and Taylor, 1936; ough, 1944, 1947; Williams, 1944; Heppleston, 1947). Whether ther dusts of low free silica content produce the focal form of mphysema remains to be demonstrated. Thus we may disinguish a specific type of simple pneumokoniosis, namely silicosis, ue to a specific chemical substance, and a non-specific type due o a mixture of different substances in which there is usually much ess free silica. In this review we are not concerned with asbestosis, specific form of pneumokoniosis due to a fibrous silicate.

That some of these lesions at least are produced by dust itself is hown by the following evidence. Histologically the simple neumokoniosis of coal workers has none of the features of an nfective process. In silicosis, however, whilst we cannot be so efinite on such grounds, bacteriological examination has shown hat typical nodules may occur without any evidence of an ssociated tuberculous infection (Simson and Strachan, 1935). t can be objected that such infection may have been present riginally and may subsequently have healed. Nevertheless, nimal experiments indicate that a lesion closely resembling the uman one can be produced by the inhalation or intravenous jection of finely-divided quartz dust without the aid of any other ctor (Gardner, 1932; Gardner and Cummings, 1933). The early ages of Welsh coal workers' pneumokoniosis have also been produced experimentally by underground exposure to air-borne ust in the absence of infection (Heppleston, to be published).

Disability in simple pneumokoniosis usually presents itself as reathlessness due fundamentally to defective exchange of gases etween the inhaled air and the blood. According to Simson and rachan (1935) fibrous tissue overgrowth is apparent at an early age in the development of the silicotic nodule and this fibrosis eadily extends as the lesion matures. Whilst there is no doubt me replacement of functioning lung tissue by the nodules, their ain effect according to Simson (1935) is to constrict the small passages around which they develop. Such focal emphysema occurs will contribute to the respiratory incapacity. As to the tors underlying the disability in the non-specific pneumokonioses can as yet speak only of the condition in Welsh coal workers. re focal emphysema appears to be the primary factor, but its chanism has not been fully elucidated. Obliteration of small eas of lung tissue by the dust foci themselves may play a bsidiary role. In both these forms of pneumokoniosis general physema, apparently occurring as an independent entity, may mplicate the picture. Occasionally simple pneumokoniosis is ponsible for death as a consequence of congestive heart failure.

Several theories have been advanced to explain how dusts as such t on the lungs. The idea that dust penetrated into the tissues of e lungs by virtue of its hardness and sharp edges has gradually en discarded since Zenker (1867) showed that particles which were neither very hard nor sharp could pass from the atmosphere into the lung substance. Gardner (1923) refuted the belief that pulmonary fibrosis was caused by irritation from hard, sharp particles when he failed to induce such fibrosis in guinea-pigs by the inhalation of carborundum dust, the particles of which are nearly as hard as those of the diamond and whose abrasive qualities are well known. Following the work of Gye and Purdy (1922, 1924) and Gye and Kettle (1922) many people now maintain that in silicosis the fibrous tissue over-growth is stimulated by slow 'solution' of free silica particles in the body fluids. Although this view was affirmed by the International Conference on Silicosis in 1938 (I.L.O., 1938) the evidence is not conclusive (King, 1938) and has been discussed critically by Gardner (1937*b*). For instance Gardner (1934) states, on the basis of his experimental work, that 'the relatively insoluble quartz particles are definitely more active than the more readily soluble silicate particles and tissue reactions begin to develop so quickly that it is hard to conceive of such a solution of silica having occurred in the weakly alkaline body fluids'. To meet such objections Heffernan (1935, 1944) suggests that the activity of free silica particles on tissues depends upon the electrically unsatisfied oxygen atoms of the freshly produced silicon–oxygen tetrahedra. Silicate particles on the other hand are electrochemically inert even when fresh. Gardner (1938), however, points out that, whilst freshly fractured free silica may be more potent than silica which has aged, the ageing process is not progressive since silica remains active for long periods, even when it is suspended in dilute aqueous sodium chloride solution. In coal workers' pneumokoniosis free silica has again been incriminated since coal dust contains a small amount of this substance (Gardner. 1934, 1935, 1937*b*, 1939; Sayers et al., 1935; Cummins and Sladden, 1930; Belt and Ferris, 1942). The adherents of this view explain the different pathological appearances of classical silicosis and coal workers' pneumokoniosis as being due to modification of the action of the free silica by the other components of the coal dust. The results of *in vitro* and *in vivo* experiments based on this idea have not provided completely satisfactory support (King and Nagelschmidt, 1945; King, 1945; Belt and King, 1945), and the *direct* effect of the major portion of the dust has received insufficient attention (Badham and Taylor, 1936). The disease in haematite workers has also been explained as a modified silica effect (Gardner, 1934, 1935, 1937*b*, 1939; Craw, 1947). Thus, whilst silicosis may well arise as a consequence of a chemical or physico-chemical reaction between the lung tissue and the silica dust, the non-specific disease of Welsh coal workers is more readily explicable as the result of simple overloading of the lungs with dust irrespective of its nature rather than as a modified silica effect. It cannot be denied that free silica may play some part in the genesis of the disease in coal workers, but its contribution appears to be of much less importance than that of the other components

[the coal dust. In these men it appears that as more dust is eposited than can be removed it collects into small aggregates 'hich then undergo a very mild form of fibrosis, thereby preventing ırther dispersal of the dust. As a consequence of the aggregation nd fibrosis focal emphysema develops (Heppleston, 1947). 'olicard (1947) also believes that pneumokoniosis may arise from he mechanical presence of large amounts of inert foreign material.

Infected Pneumokoniosis

This type occurs as a complication of simple pneumokoniosis ıd consists essentially of pigmented masses of hard fibrous tissue nbedded in the lung substance. In size they vary from one to ›veral cm. across and are usually located in the upper or middle ortions of the lung and somewhat posteriorly. Cavitation may ccur in massive lesions, appearing either as an area of liquefaction ue to loss of blood supply or as a typical tuberculous cavity. .assive lesions occasionally show other evidence of tuberculosis ıd a rapid terminal involvement of the whole lung by tuberculosis ıay originate in them (see Gardner, 1937*a*; Hale, 1946; Craw, 947). Bullous emphysema, sometimes of severe degree, may ccur in relation to the massive lesions, whilst general emphysema ıay be present independently.

There is good reason for believing that these massive areas of brosis are tuberculous in origin. They occur in places which often orrespond to the common sites for the development of chronic ulmonary tuberculosis in the general population (Gardner, 1939). Iicroscopically massive lesions usually bear evidence of a chronic ıfective process and in a proportion of cases this is tuberculous. n silicosis Gardner (1940) found microscopical signs of tuberculosis ı 60 per cent of massive lesions, but in coal workers' pneumokoniosis ıe incidence is approximately 30 per cent (Gough, 1947). Further, imson and Strachan (1935) isolated tubercle bacilli from certain of ıeir silicotic massive lesions, and Rogers (1946) claims to have emonstrated this organism in 75 per cent of such lesions from a ›ries of 831 Welsh coal workers. The absence of histological and ıcteriological evidence of tuberculosis in some cases may mean ıat the technical methods are inadequate, that the infection has een overcome leaving a healed lesion, or that the massive fibrosis ›presents the end result of a non-tuberculous inflammatory rocess such as a localized pneumonia. Animal experiments show ıat tuberculous infection of the lungs combined with the inhalation [various dusts, including free silica, can induce extensive fibrosis ardner, 1937*b*, 1938), but non-tuberculous infection of the lungs [silicotic rabbits failed to do so (Vorwald, Delahant and Dworski. 940). If we accept the view that tuberculosis is the underlying ıuse of infected lesions in pneumokoniosis we must admit that ıe tuberculosis usually occurs in a modified form, one which ıggests a retardation of its rate of progress and a limitation of its ctent.

It is comparatively easy to appreciate that the obliteration of a large area of lung tissue by massive fibrosis will interfere with respiratory function and produce disability, especially when aggravated by bullous emphysema occurring as a consequence of the fibrosis. Severe strain may be placed on the heart by these two factors and congestive heart failure may ensue, often proceeding to a fatal issue. This termination appears to be commoner than active tuberculosis in Welsh coal workers (Gough, 1947), whereas the reverse obtains in classical silicosis (Strachan, 1947). Death is sometimes due to causes other than pneumokoniosis.

Assessment of the Degree of Disability

This assessment provides the basis for compensation and also for recommendations as to suitable alternative employment. Conditions simulating pneumokoniosis must first be excluded.

Simple Pneumokoniosis.—The clinical history and examination may be quite misleading owing to the subjective factor and the absence of diagnostic features. The degree of clinical disability in silicosis cannot be correlated accurately with the extent of the radiological changes (George, 1938 ; Irvine, 1938 ; Crombie, Blaisdell and MacPherson, 1944), and the same is true for Welsh coal workers. In the latter Gough (1947 and to be published) has demonstrated that the X-ray appearance of 'reticulation' (Hart and Aslett, 1942) is produced by the superimposition of the shadows of the dust foci, no matter whether these are accompanied by focal emphysema or not, yet it is precisely the focal emphysema which is believed to be mainly responsible for the disability. The physiological tests so far applied to this problem have not proved very satisfactory. Many of them are open to subjective errors, are time consuming, lack delicacy and seem incapable of distinguishing incapacity due to pneumokoniosis from that due to associated but not necessarily related pulmonary or cardiac conditions (Irvine, 1938 : McMichael, Hart and Aslett, 1942 ; Crombie, Blaisdell and McPherson, 1944). For practical purposes a simple and reliable objective test has still be to found.

Infected Pneumokoniosis.—Similar considerations apply here. Whilst there is perhaps a closer correlation between the degree of clinical disability and the extent of the radiological opacities in the infected than in the simple form of pneumokoniosis, this is by no means absolute and there may even be no disability at all. The discrepancy probably depends upon the coexistence of bullous emphysema, generalized emphysema, simple pneumokoniosis or active infection, singly or in any combination. The separation of disability due to pneumokoniosis from that due to associated diseases may again prove difficult. The presence of active tuberculosis in massive lesions is often revealed as 'fluffy' shadows on the X-ray film, but clinical evidence of such infection may not be apparent until the later stages. Occasionally bacteriological examination demonstrates tubercle bacilli in the sputum.

Although the assessment of disability on the basis of clinical or X-ray examination leaves much to be desired, the current practice of combining the two methods does achieve a reasonable degree of accuracy in experienced hands. If such examinations are carried out prior to and at intervals during the occupational exposure to dust a continuous record is provided for each man and, from the South African experience of silicosis, it appears that many errors of assessment can thereby be reduced (Orenstein, 1938). The supplement of a simple objective physiological test would provide a measure of disability instead of relying on personal judgement. Because a miner is physically disabled by pneumokoniosis it does not necessarily mean that he is incapable of any other work. If he can work it is most desirable that he should do so in his own as well as in the national interest. The rehabilitation of disabled men entails a complete knowledge of their physical capacities together with the type and maximum physical requirements of the alternative occupation. These factors must be equated in each case.

The Control of Pneumokoniosis

There are two main aspects to this problem—engineering and medical.

Engineering.—The basic requirement for prevention of the disease is the suppression of dust, preferably at its source, combined with adequate ventilation. Since it appears that dusts of very different composition can induce disease, the practical aim must be to eliminate all forms of dust and especially the fractions below the size of 5μ. Apparently more attention should be directed to the smallest fractions, since Orenstein (1947) suggests that water may fail to remove the dangerous very fine particles from the air.

Medical.—Short of prevention the only means of arresting simple pneumokoniosis is removal from the dust hazard. So far as the worker's health is concerned the ideal is removal at the earliest sign of incapacity but economic considerations may necessitate a modification of this objective.

The eradication of infection, particularly tuberculosis, is of the greatest importance. Ideally tuberculosis must be excluded in every entrant into an occupation with a dust hazard, and a man developing this infection whilst exposed to dust must be removed from his occupation forthwith, both for his own sake and to avoid the possibility of infecting others.

These objects can only be achieved by the institution of initial and periodical medical examinations for all workers in dusty trades. Such examinations should include a full industrial history and be conducted clinically, radiologically and, when practicable, physiologically. Laboratory investigation may be needed in certain cases. To interpret the results of these examinations correctly it is essential to correlate them closely with the pathological features seen *post-mortem*. Much has already been done in this direction, mainly with reference to silicosis (Irvine, 1935–1938; Smith, 1941–1944; Sayers, 1938; Crozier, Martin and Policard, 1938), and Gough is proceeding on these lines in connection with Welsh coal workers. In South Africa, in haematite mining and in certain other British industries clinical and radiological measures have been applied systematically with appreciable success (Irvine, 1938; Craw, 1947). The initial examination may prove to be the most important part of the scheme by excluding those men whose physical features suggest that they are particularly liable to contract pneumokoniosis or tuberculosis (Smith, 1941–1944; Craw, 1947).

Initial and periodical medical examinations of all workers exposed to a dust hazard provide information as to the incidence of simple and infected pneumokoniosis in the population at risk and enable exact comparisons to be made between the incidence of pulmonary tuberculosis in these men and in the remainder of the working population. The importance of such knowledge lies in the fact that the value of any particular method of control, whether engineering or medical, must ultimately be assessed by its effects upon the incidence of the disease in both its forms. As penumokoniosis develops slowly medical supervision will be necessary for many years before this information is forthcoming. It is desirable that medical supervision of affected workers should continue after exposure to dust has ceased. South African experience of simple silicosis shows that this form may progress after leaving the mines (Irvine, 1935–1938; Smith, 1947), as might be expected on the 'silica solubility' theory. In Welsh coal workers, where overloading seems to be the most important factor in the simple form of the disease, such progression appears less likely to occur, but this contention requires substantiation. A more important reason for continued supervision of cases of simple pneumokoniosis after cessation of employment is the detection of infective complications, particularly tuberculosis. Infected pneumokoniosis, whenever it arises, is a progressive condition and accordingly men suffering from it should be followed up indefinitely after their compulsory suspension from work. When tuberculosis does develop the members of the man's family should be examined to exclude infection in them.

The position of aluminium in the prevention and treatment of human pneumokoniosis is far from settled. The experimental work of Denny, Robson and Irwin (1937, 1939) and of Gardner, Dworski and Delahant (1944) leaves no doubt that silicosis in animals can be prevented or arrested by the inhalation of aluminium or alumina dust. Gardner et al. nevertheless believe that alumina may influence unfavourably resistance to tuberculosis. Clinical trials in Canada, using aluminium powder, on 34 silicotic men produced subjective benefit in 55 per cent. and 32 per cent showed objective evidence of improvement (Crombie, Blaisdell, and MacPherson, 1944). These authors believe that treatment with aluminium powder of respirable size offers every prospect of preventing the development of human silicosis. Other workers, however, have not achieved the same success in the treatment of the disease. It is very doubtful whether aluminium therapy has ever caused the regression of established simple silicosis as judged by X-rays, and as a preventative measure the method remains unproven. Its use must therefore be subjected to the strictest medical control over a prolonged period. In silicosis aluminium is believed to act by coating the silica particles with an insoluble and impermeable layer of aluminium hydrate, so preventing ‘ solution ’ of the silica in the body fluids. On Heffernan's theory, however, aluminium combines with the unsatisfied oxygen atoms of the silica tetrahedra to form a surface layer of alumina, so rendering the silica inert (Denny, Robson and Irwin, 1937). It has been suggested that the benefit derived from aluminium treatment may depend on quite a different mechanism, namely the relief of bronchial spasm. No evidence has been adduced in support of this contention, which must therefore be regarded as speculative. In the simple pneumokonioses due to dusts with a low free silica content the chemical or physico-chemical theories of silica action may not be applicable, so that aluminium may have no beneficial effect and even do harm by adding yet more dust to that already overloading the lung. Aluminium therapy must on no account be regarded as a substitute for the accepted methods of dust control.

Primary Cancer of the Lung in Relation to Pneumokoniosis

In South Africa there is no evidence to suggest that pulmonary cancer is more frequent in silicotic miners than in non-silicotic miners or in a similar body of adult males in the general population (Irvine, 1935–1938). Vorwald and Karr (1938) from America state that there is no radiological, post mortem or experimental evidence to indicate that the inhalation of dust (except recognized carcinogens such as radium and tar) is associated with an increased incidence of pulmonary cancer. Experience in England and South Wales, although not yet based on statistics, appears to support this conclusion in regard to both silicosis and coal workers' pneumokoniosis.

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THE INSTITUTION OF MINING AND METALLURGY

Report of Speeches at the Annual Dinner held on Thursday, 5th May, 1949

The Annual Dinner of the Institution of Mining and Metallurgy was held at the Savoy Hotel, London, on Thursday, 5th May, 1949. The Chair was taken by the President of the Institution, **Mr. S. E. Taylor,** and the company of members and guests numbered about 200.

The toast of 'The King' was proposed by the President and loyally honoured.

'The Institution of Mining and Metallurgy'

The Rt. Hon. Sir John Anderson, in proposing the toast of 'The Institution', spoke as follows: I need hardly say that I regard it as a great honour to be asked to propose this toast. When your President sent me the invitation I accepted it with alacrity because I realized the importance of the occasion. Then, when I came to reflect, I felt that I had perhaps been a little rash, possessing none—or very little—of the scientific background of others, such as the late Sir William Bragg, who, I observe, proposed this toast on a previous occasion. However, I have knocked about a good deal and picked up a few odds and ends which may perhaps form the basis of some remarks to submit to you. I should like to say at the very outset that this very famous Institution has never been more important, has never had greater opportunities of rendering valuable service to the community, than at the present time.

In what I am going to say I think I had better make a separation of the two subjects which are included in your title—mining and metallurgy. I confess, looking round these tables, I should find it a little difficult to distinguish the miners from the metallurgists. My excuse for referring to some of you as miners is that as I came into this hotel I heard the doorkeeper saying to everyone, 'Miners downstairs'. But you all look the same, fairly cheerful—though that possibly may be a postprandial condition.

To come to mining; was there ever a time in our country when the spirit of adventure, the spirit that leads people to go out into the world and prospect, was more necessary for the restoration of our economy than it is at the present time? Of course, you all know very well, better than I do, that prospecting in the mining field is a matter not only of difficulty but of risk, and not only of personal but of economic risk. If people are expected to take that sort of risk they have to be lured on by the expectation of a commensurate reward. (Applause.) I rather thought that would evoke a reaction. That is where taxation hits you hard. But you will perhaps give me—if you are really the kindly people you seem to be—some little credit for having tried to do as much as I could in a former capacity to ease that position, and I am glad to think that in that respect, at any rate, I have a successor as Chancellor of the Exchequer who has followed in my footsteps and sought to improve on what I was able to do.

I cannot speak on either mining or metallurgy as an expert. If I pretended to do so the President of the Royal Society, sitting two paces away from me, would think very ill of me, or less well of me than I hope he does now. But I do know just a little. I know that mining engineers have not yet by any means exhausted the possibilities of the mineral wealth of the world. I recall that when I was in the Colonial Office nearly half a century ago the first geologists were sent out to tropical Africa—to the Gold Coast and Nigeria—and practical results followed very speedily in your particular field. Diamonds were found in Sierra Leone, the Udi coalfield was discovered in Nigeria, and the Bauchi tinfield was opened up by a light railway. More recently important new discoveries have taken place in the field of mining. There is, for example, titanium—a metal of growing importance—in the Quebec Province of Canada.

A metal of great interest, of course, is uranium, and I could not let an occasion like this pass by and omit a reference to it. That brings me to a matter which is cognate, namely, the importance of developing new techniques in dealing with mineral ores, especially low-grade ores. When I was in Canada not very long ago I saw there how a particular ore—consisting of an intimate mixture of sulphides of iron, lead, zinc, copper, and antimony, with a little silver and gold—was dealt with by a process which, I believe, was called the flotation process, devised by one of your own members and a friend of mine. When we come to uranium, this occurs in massive formations only in very few places in the world, notably in the Belgian Congo and in Canada. In South Africa there are sources which probably greatly exceed in total quantity the possible production from the fields I have mentioned, but it is a very low-grade material and a great deal must depend on the development of new and efficient methods of recovering the uranium from such material. All that is exceedingly important, not only for mining engineers or mining companies, but for the whole human community, and that is a matter in which you can make tremendous contributions.

If I may now switch over to metallurgy; on this I know even less than about geology and mining. I see the Director of the Geological Survey in front of me and his presence reminds me that there was a time when I spent some years in trying to learn about geology. But about metallurgy I can honestly claim to know nothing. Nevertheless, I am going to tell you something about it, thereby following the example of politicians, who are always telling the public about things of which they themselves know precious little. Their success depends upon the skill with which they manage to conceal their ignorance. I hope I have disarmed you by that confession at the outset, but I do know this that until quite recently metallurgy was almost wholly empirical, it was an art rather than a science. During quite recent years science has brought new light upon metallurgy; new methods, new instruments have become available. I need only refer to the electron microscope, to X-ray analysis, and, even more recently to what I believe is called X-ray diffraction. All this has turned a vast amount of light on what was until then a very obscure field. I am sure that metallurgists now understand as they never did before the significance of the various phenomena connected with metals. They know more about 'creep' and about the age-hardening of alloys—a very interesting topic. Certain alloys, as you all know, develop hardness at ordinary temperatures and more rapidly at temperatures somewhat above normal in the mere process of time. It is now known that this is due to changes of phase in the crystal formation and in that field your Institution has obviously a very great contribution to make by encouraging study and by the development of colleges of technology.

That brings me to a subject which is common to both mining and metallurgy, namely, the provision of better facilities for technical education. It is very easy to exaggerate the extent to which we in this country have failed to keep pace with other countries, notably America, in the matter of technical education. I say that it is very easy to exaggerate that. After all, we are not so bad in this country, but I am sure we could do with more and better facilities, and your Institution by providing scholarships and giving encouragement to students and helping more directly in the development of technical colleges can make a most significant and valuable contribution to the well-being of our country. We do require to make the most of our natural aptitudes and we do require all the help that science can afford. In pure science we can challenge comparison with any country in the world, but as Sir Henry Tizard has pointed out more than once, what we require is not only increased scientific knowledge but, even more, dissemination of that knowledge and facilities for its application to practical affairs.

That is all I will venture to say to you. Some of you may disagree with some of the things I have said, but, on the whole, I think most of you will be in accord.

I have to couple with this toast the name of your President, Mr. Taylor. I have not had the pleasure of knowing him until this evening, but he has been telling me something of the activities in which he has been engaged. Like me, he has spent some time in India, and like me he must be much interested in political developments in that great country. There is one thing that can be said with perfect confidence about India, that the men and women of our race who have gone to India have come away with an immense feeling of affection for the people of that great country. They have taken a road of their own choosing and we wish them well in their passage along it. I say that both of India and of Pakistan, and I have many personal friends in both parts of geographical India.

May I just add this, that if the President has anything he can tell us about the gold standard, I, as an ex-Chancellor of the Exchequer, will be much interested to hear it.

The President said: Sir John Anderson, my Lords and Gentlemen, in responding to this toast so admirably proposed by Sir John, I must, first of all, on behalf of the members of our Institution—and I feel sure I can include our guests—thank him sincerely for the encouraging and interesting way in which he has proposed our health. He said that he had felt somewhat rash in accepting my invitation, and we can only say that we are delighted that he is so rash a man.

This Annual Dinner is a great occasion for the Institution of Mining and Metallurgy because it gives an opportunity to our members to meet and make themselves better known to those whom they serve and who are so dependent on them, and upon many of whom they in their turn are dependent. Even we ourselves are apt to forget what an important and world-wide profession we are. No harm will be done if on this occasion I remind you who we are and what we are doing.

I must not steal the thunder of your President-Elect who will shortly be indulging in the pleasurable task of welcoming our guests, but I cannot resist the temptation—indeed, I should be failing in my duty—if I did not say how delighted we are to have so many distinguished friends with us and also so many members and friends from overseas.

We are a comparatively small society, our members number only 2,200, and more than half of them are scattered about the world in seventy different countries. Though few in numbers a great responsibility rests on our profession. We are charged with the great task of maintaining the supply of metals and minerals in the quantity and form that they are required by innumerable industries and individuals. These metals and minerals are indispensable raw materials and they must continue to be made available despite the wastage of resources or the difficulties of working, extraction and refining.

Rather than speak of the Institution as a body or profession I would like this evening to speak of the individual members, for all the qualities, status, attainments and shortcomings of our Institution are in fact those of our members.

The first thing about our members is the variety of their skill and knowledge, defying classification. I find this gives me great pleasure, for I must say that instinctively I object to classifications and designations, I suppose because they tend to submerge individuality. This individuality is a very characteristic feature; for we are frequently called upon to form opinions when the only evidence available is distinctly slender. Personal opinion can become a highly developed art. It may vary from intelligent guesswork to scientific reasoning sometimes giving to the uninitiated the appearance of uncanny intuition or almost second sight. Probably without realizing it this quality becomes highly developed by continually probing the unknown and unexpected. For the composition of the earth's crust and the properties of metals and minerals are for ever springing surprises and sometimes seem to disobey our rules for the game.

Our members include mining engineers, geologists and metallurgists and each of these branches covers a wide variety and they merge into one another. Frequently an engineer in charge of a small mine is called upon to cover all branches of the profession and be like the captain of a ship, a law unto himself and his own doctor, electrician and accountant as well.

Further, it is just because this wide variety amongst our members is still not wide enough that we have just taken a most important step in our history by opening our doors of membership still wider and creating a new class of Affiliates for members of other professions —such as mechanical and electrical engineers and doctors—whose work is intimately connected with and essential to the mining and metallurgical industries. Here I would say in anticipation of the many applications I am sure we shall receive for Affiliateship that we shall welcome you and we need you and we have something to give that you need. This part of my remarks is a recruiting speech for which I make no apology and I hope all members will follow my example and help to swell the Affiliate class. Perhaps some of you who are here as guests tonight are qualified to join; if so my welcome is direct and to you personally. The application forms are now ready.

The next thing about our members is the high standard of integrity which tradition imbues and our profession demands. By the nature of our work this is essential for we may make discoveries both good or bad which can be of great financial import, or we may be called upon to form an opinion on a proposal involving large expenditure and considerable risks.

We claim no monopoly in professional integrity, for happily it is common to members of other technical societies, so many of whose representatives we are pleased to have with us this evening.

I suppose when we do manage to find time to reflect on our problems, our profession or our industry, we tend to look forward and try and peer into the future, which is most necessary and desirable. In this profitable exercise of our imagination, however, we are apt to forget the past and underestimate the value of the experience of our predecessors.

For this reason and also for personal reasons I am going to try and give you a glimpse of a mining engineer 100 to 150 years ago. Early in 1799 my great-grandfather, John Taylor, at the age of 19, and after an early training in land surveying in Norfolk, was given charge of the Wheal Friendship mine near Tavistock. His employers started him at a salary of £100 a year and in a letter dated 18th April, 1799, he says: 'I thanked them for this un-commonly handsome allowance and expressed the wish and hope that my services might be of the value to them they ought to expect '.

He was appointed superintendent with the special object of reforming abuses which had crept into the administration. One who knew him intimately wrote: 'His judgement, uprightness and firm but considerate management of the men under his charge soon achieved success '. Within a few years he was entrusted with the conduct of many other mining enterprises in Cornwall, Wales and the north of England.

In 1803 he was faced with a transport problem and solved it by constructing a canal 5 miles long from Tavistock to the river Tamar, including a 2-mile tunnel through Morwelham Down. The work took 13 years to complete. The ore from the mine was taken by barge along the canal to the works and the project was a great success.

A few years later he solved another transport problem by organizing and constructing the Redruth and Chacewater Railway which operated successfully for 55 years.

In 1812 he came to London as the best place to control his many interests, which by then extended to chemical manufacture and the technical societies in addition to the management of mines.

In 1810 he was awarded the silver medal of the Royal Society of Arts for a paper entitled 'Method of ventilating mines or hospitals by extracting foul air from them '.

In 1815 he was granted a patent for a process for producing 'inflammable air or olefiant gas applicable to the process of giving light ' from oil, fat, bitumen or resin. A plant was installed and the gas was used to illuminate the Covent Garden Opera House.

In 1816 he became Treasurer of the Geological Society and held that office for 27 years.

In 1824 he undertook the formation of a company to work the silver mines of Real del Monte in Mexico and was thus one of the first mining engineers to direct overseas mining enterprises from a London office.

His main problems in this case were heavy water, for which he ordered and shipped a large Cornish pumping plant from Hayle, and the length of time letters took which made distant management through agents most difficult.

About this time there was much speculation in various joint stock enterprises including foreign mining. In 1825 he made the following pronouncement: 'My inference from the whole is that mining is neither, as the public seem to think a few months since, a certain source of immeasurable wealth, to be obtained by everyone who was lucky enough to get a share in any mine, in any place, and under any kind of management, nor is mining, as it seems now the fashion to designate it, all a bubble, cheat and delusion. I maintain that British capital may be applied to it with fair chance of competent profit, if the means properly adapted to the end be used and steadily continued '.

In all he had the control and management of some 92 mines.

He was always ready to assist any project that had for its object the advancement of learning. He was a Fellow of the Royal Society, a founder member of the British Association, one of the founders of the London University, a member of the Royal Society of Arts and of the Institution of Civil Engineers.

In 1825 he wrote a prospectus for the establishment of a School of Mines giving detailed plans for its curriculum, government and maintenance. In it he says : ' The mines of distant countries are passing into the hands of English possessors, and they must rely more or less for their success upon the talents of the agents they employ. A School of Mines will not only be the means of instructing such agents, but it will be a place where character will be developed, and, as it were, put upon record '. —Apropos my earlier remarks about members of our profession he says : ' Miners in general are a superior class of men, and, in the deep mines particularly, the constant exercise of judgement and thought which is necessary produces a proportionate degree of intelligence '. But he was ahead of his time and the proposal for a School of Mines did not meet with encouragement.

However, he trained his two sons in the latest mining and metallurgical practices, sending them to Germany and elsewhere. In due course they joined him in his business which thus became known as John Taylor & Sons.

In all there have been 13 members of the family in the firm extending over 5 generations in 150 years.

I hope this brief reference to John Taylor will serve as a tribute to one who can claim to be one of the founders of our profession and at the same time stimulate a study of the work and wisdom of our predecessors. Such a study will prove profitable as an inspiration and encouragement in tackling our present-day problems and in framing our future policies.

One of the activities of John Taylor that I mentioned was the direction of overseas mining ventures from London.

It would make a fascinating study to trace the development of he City of London's overseas mining activities from 1824 onwards. For 50 years or more mining companies have held a prominent position in the City. But to-day it seems, with growing nationalism everywhere, the unique facilities of the City of London, which make it the best place from which to direct the complex, world-wide overseas mining and metallurgical industry, are too highly priced. Through the medium of excessive taxation this country is demanding too high a fee for the service it renders to overseas industries. In consequence the City is losing and will continue to lose valuable business. The direct and indirect benefits which this country derives from the overseas mining industry are not fully appreciated in some quarters. Our overseas mining interests are a valuable national asset that should not lightly be cast away. It is true that after patient efforts some measure of relief has been granted, but it is not enough, and a much more realistic attitude is required towards the taxation of the overseas mining and metallurgical industry if the City is to retain its position as the best place for directing overseas mining activities.

I am permitted on this occasion to express my own personal views on topical and even controversial subjects and so will take the risk of treading on dangerous ground and say a word or two about the price of gold.

I am well aware of the important economic and political influences surrounding this question about which I am not competent to speak. However, the production and sale of gold goes on from day to day and in consequence vast sums of money are put into circulation and thereby purchasing power and demand are created and stimulated.

I may be wrong, but it seems to me it is a good thing to sustain this source of purchasing power and that it would be a bad thing if this flow of new wealth dried up.

Over the ages the price of gold has tended to rise along with the cost of living and it seems to me that it is about time the price of gold was allowed to rise again. The benefits that would arise from an increase in the flow of new wealth by reason of a rise in gold price would, I should have thought, outweigh the objections.

I should like now to refer to India for two reasons; in the first place I have had a long and close association with India and, secondly, the happy outcome of the recent conference of Commonwealth Prime Ministers is fresh in our minds. Many of our members have played their part in the development of the mineral resources of India and we have some 100 members in India, including 30 Indian members, so we are directly interested in that great country. Our members have ties of friendship arising from common endeavour and many others from this country have had close association with India, Pakistan and Ceylon, whether it be through Government service both administrative and military or by professional and business work. There exists a bond of friendship between many individuals and there is a traditional and national friendship. These deep feelings spring from the fact that we have helped one another in good times and in ill.

We therefore welcome the clear recognition by the conference that these deep feelings of friendship exist and will continue.

Before closing I must express the thanks of the Institution to the Nuffield Foundation and through the foundation to Lord Nuffield, to Messrs. Capper Pass and to the Mond Nickel Company for their generous and far-sighted help and encouragement in the training of metallurgists by means of Scholarships and Fellowships. I feel sure they must all derive much satisfaction from the success of the scheme.

I am very pleased to announce that the Council of the Institution has shown its deep appreciation of Lord Nuffield's part in this valuable work by conferring upon him Honorary Membership of the Institution.

'OUR GUESTS'

Mr. W. A. C. Newman (President-Elect), in proposing the health of 'The Guests', said: This time-honoured toast is drunk with equal zest whether in a small company of two or three or at an annual festival such as the present. As the President has said, the company numbers many important guests covering a wide field of interests with which the Institution is directly or indirectly concerned. Their presence emphasizes the convergence of such interests and their interdependence, which becomes more pronounced as the days go by.

It is impossible for me to mention every guest by name as I would like to do, and to describe the part which each plays in the public, industrial or academic life of this country, and I ask indulgence for any errors of omission. I feel sure our other guests will not misunderstand me if I refer to Sir John Anderson and Sir Clive Baillieu as our principal guests this evening.

Sir John Anderson was at one time a distinguished Civil Servant, being Permanent Secretary at the Home Office. Subsequently he became Governor of Bengal, and then assumed very high offices in the Government at home, finally becoming Chancellor of the Exchequer. Throughout his life and especially during the war years, aided by his wide knowledge of men and affairs, which included not a little of the scientific outlook, he has contributed greatly to the welfare of this country.

British parliamentary institutions are respected and envied throughout the world, and it was hoped to have a fuller representation of both Houses of the Legislature present this evening. Unfortunately, Lord McGowan and Mr. Oliver Lyttleton were not able to be with us, but we are very glad to see Viscount Falmouth, who is Chairman of the Governing Body of the Imperial College of which the Royal School of Mines is an integral part.

There are present a large number of chairmen and directors of mining companies in the City of London that, collectively, have mining and metallurgical interests covering the whole face of the earth. We are glad to learn from time to time that the work of the Institution has found favour with the mining community, and that they look with confidence upon the hallmark of qualification which membership of this Institution confers on those who receive it.

It is very rare on an occasion like this that the Fighting Services are not directly represented, but we are in that unhappy position this evening. But the circumstance brings into relief the fact that the other Service—the Civil Service—is represented by many distinguished members. I have already referred to Sir John Anderson in his role of Civil Servant. We are glad to welcome too, Sir Ben Lockspeiser, Secretary of the Department of Scientific and

Industrial Research, Mr. D. J. Wardley, Deputy Master of the Royal Mint, Mr. A. M. Bryan, Chief Inspector of Mines, and Dr. W. F. P. McLintock, Director of the Geological Survey of Great Britain. Others whom we welcome most cordially are Sir Charles Goodeve, Director of the British Iron and Steel Research Association, and Mr. G. L. Bailey, Director of the Non-Ferrous Metals Research Association.

The Dominions and Colonies are represented by the Acting High Commissioner for Southern Rhodesia, the Deputy High Commissioner for the Union of South Africa, and the Supply Commissioner for Pakistan. We are pleased to have among us also Sir John Calder, Senior Crown Agent for the Colonies.

We are very appreciative of the presence of the Presidents of nine sister Societies headed by the Royal Society, indicating, I think, the increasingly close relationship which exists between them and us in our professional activities. It is an interesting fact that among the nine Presidents, three are also members of the Council of the Institution.

The President has already mentioned the debt which is owing to certain bodies—the Nuffield Foundation, the Mond Nickel Co. and Messrs. Capper Pass and Sons—for the generous donation of large funds in the administration of which the Institution has recently participated.

These funds are to be applied for the advancement of vacational and post-graduate experience in metallurgical fields both here and abroad and for the stimulation of papers on plant and processes used in extraction metallurgy and in the fabrication industries.

Unfortunately Sir William Griffiths, the Chairman of the Mond Nickel Company, is unable to be present, but Mr. Farrer-Brown, Secretary of the Nuffield Foundation and his colleague, Maj.-Gen. Bullen-Smith, and Col. Sir Paul Gueterbock, Managing Director of Messrs. Capper Pass & Sons, by accepting our invitation have given us the opportunity to express to them personally our thanks for the extreme generosity of their respective corporations.

I should like to mention two other guests individually : Dr. C. H. Desch, who might be termed the doyen of physical metallurgy in this country, is to deliver the Sir Julius Wernher Memorial Lecture to the Institution in July next : Col. W. French, who as Superintendent of Technology of the City and Guilds of London Institute has for many years organized a unique system of examinations in a great variety of technological subjects, which have had a great influence on the standard of the teaching of those subjects in the schools and technical colleges of this country. Col. French, I understand, is shortly retiring from his arduous office, and we wish him well in his retirement.

There remains only for me to mention the gentlemen of the Press—on whose good offices we depend for publicity and the faithful recording of our activities—and the guests of members of the Institution, all of whom, no less than the guests I have mentioned, we welcome most heartily.

With great pleasure I associate this toast with the name of Sir Clive Baillieu—a man of many parts and wide sympathies, a great Britisher, a wise and shrewd industrialist. He is Chairman of the Dunlop Rubber Co., and has many important banking and commercial interests. For two years he has been President of the Federation of British Industries and has been at the head of two missions in Washington which dealt with the purchase of war and other materials during the recent conflict. But it is rather in connection with our own industries that we know him best. As chairman of the Central Mining and Investment Corporation and as a director of the Zinc Corporation and the Imperial Smelting Corporation he has done yeoman service. Members of the Institution recall with great gratification that it is just 20 years ago almost to the day that Sir Clive's father, the Hon. W. L. Baillieu, a pioneer in the development of those great mines in the Broken Hill area, and of their associated smelting companies, a dominant and much-beloved personality, was asked jointly with Mr. W. S. Robinson to accept the Gold Medal of the Institution, the highest award in its power to give. It is a great delight to us all that we should be honoured this evening by the presence of such a worthy and distinguished son of such a worthy and distinguished father.

Sir Clive Baillieu, in acknowledging the toast, said: I recall that it was Dr. Johnson himself who once said that when a man is invited to dinner he is disappointed if he doesn't get something good. I can assure our very generous hosts tonight that their guests are suffering from no sense of disappointment. Both fare and company have been excellent and well up to the traditional standards of this annual function which attracts so much interest, and such a distinguished company.

I suppose, following a rather conventional form, I should start by apologizing for finding myself honoured with the response to the toast of 'the Guests'. I have the feeling, however, that if any apology is needed it should be made by those who reposed this task in me. It was Lord Beaconsfield, in his role as author, who made one of his characters that had dined well and in goodly company say: 'I feel a very unusual sensation; if it is not indigestion I think it must be gratitude'. We are very grateful to you Mr. President for your hospitality this evening, and my fellow guests, with myself, are equally grateful to the President-Elect for the warmth of his reference to the many guests here this evening. He has spoken of my father. In some sense, perhaps, I may claim to have been born in 'the purple', for, from my earliest years, through family associations, I found myself in close contact with the personalities and problems of the industry you serve. I recall with special pride the fact that my father and my life-long colleague and friend, Mr. W. S. Robinson, were recipients of the Gold Medal of this Institution. Nothing, I may say, in their careers—which were devoted to the building and strengthening of the non-ferrous industry throughout the Empire—afforded them greater gratification than this high honour.

I am left with certain broad impressions, after listening to the speeches of Sir John and of your President. The members of this and kindred Institutions are the natural frontiersmen of industry, and the spearhead of civilized man's continuous efforts to widen the geographical bounds and to develop the mineral resources of this planet of ours. Your work naturally involves the taking of risks, and with the risks you take go the complementary financial risks involved in the exploration and development of great mineral resources. The special position which the City of London has achieved in relation to world mining enterprise is a reflection of the success which has attended our joint efforts. It is my duty and privilege to sit on the Boards of certain mining enterprises, and there I do not see any lack of the initiative, any lack of the courage or desire to move ahead and to open new fields. Such ambitions which were characteristic of our part still move us to-day ; they are there, working and waiting for their opportunity. Taxation, of course, places a restraining hand on these activities, but the spirit which underlies and animates those controlling or working in great mining enterprises to-day is very similar to that which inspired our forebears. This attitude and approach to mining problems is reflected in the lines of an English poet well loved by my own countrymen :

No game was ever yet worth a rap
For a rational man to play,
In which no accident, no mishap,
Could possibly find its way.

Sir John and the President have made reference to the price of gold. This is a matter of more than passing interest to many who attend this Dinner this evening. May I express the hope that if in the course of time and the permutations of politics, Sir John again finds himself where his successor is to-day he will not forget the word of the President.

The President referred to the great variety and integrity of the members of this Institution, and he gave us a most interesting account of the career of one of its great pioneers, his distinguished ancestor who played a unique role in the early days of mining enterprise in this country and in the Empire.

It is well for us to study the past, for the past is so often prologue.

We are so apt to forget the lessons of the past, even those of the very immediate past. The President-Elect, in proposing the health of the guests this evening, alluded to my duties in Washington during the war. Amongst other things I then had the job, on the Raw Materials Board, of harnessing the combined mineral resources of the allies for the purposes of war. One of the greatest difficulties which my American colleague, Mr. William Batt, and I faced, was to secure an intelligible balance sheet of our resources and requirements. Without such a balance sheet we found great ifficulty in doing anything like a workmanlike job. In the course, owever, of two or three years techniques were developed and a ide mass of information and knowledge obtained that was xtrémely useful, and it is fair to say that before the end of the ar this particular job was ticking over reasonably well. Following ie end of the war, however, the machinery so patiently erected as dismantled and the personnel dispersed. Perhaps this may ave been inevitable, and I know that efforts have been, and are eing, made to keep some of the work then started moving ahead, id that a lot of information is being collected and collated in arious quarters. I wonder, however, whether this is being done ı a form that is entirely adequate, both for current needs and for ossible future requirements should we ever again find ourselves ıvolved in a major crisis. At this stage I would enter a plea that ie matter should be brought under review and that we should onsider whether or not we are doing all we should to maintain, ı an up-to-date form, an inventory of the vital mineral resources f the British Empire which would, of course, include some of those ırer metals and minerals which are of such vital importance to ur industrial growth and our national security.

In this Institution and in kindred bodies there exists a unique eserve of technical and professional knowledge which can continue o increase the vital mineral resources of the British Commonwealth.

In conclusion, may I say just a word about the Fourth Empire lining and Metallurgical Congress to be held in this country in uly. I happen to be a working member of the Organizing ommittee of that Congress. This Committee has been living ıborious days and scorning delights in planning and preparing for his Congress. We will assemble in the early days of July in ondon and, after the initial receptions—including, we hope, what ill prove to be a very successful Banquet at the Guildhall—we hall proceed to Oxford for a week, where the technical sessions will e held. Then the Congress will break up into a number of parties nd will make a series of visits to different parts of the country. 'he technical sessions will cover a wide range of subjects, and we elieve will arouse considerable professional and technical interest. Ve are hoping that all our friends, who are coming to us from verseas, will enjoy themselves. A very warm welcome awaits hem and, as a member of the Organizing Committee, I would take his opportunity of thanking all those who, in response to my appeal s Treasurer, have come forward so generously to meet our needs.

THE INSTITUTION OF MINING AND METALLURGY

7TH ORDINARY GENERAL MEETING of the 58TH SESSION held in the rooms of the Geological Society of London, Burlington House, Piccadilly, London, W. 1, on Thursday, 21st April, 1949.

Mr. S. E. TAYLOR, *President*, in the Chair

RESÙMED DISCUSSION ON

Geophysics and Economic Geology

By J. McG. BRUCKSHAW, Ph.D., M.Sc., D.I.C.

The President said that as several members had been unable to speak at the previous General Meeting, it had been decided to resume the discussion on Dr. Bruckshaw's paper.

Mr. G. F. Laycock said that at the last Meeting there was such a galaxy of geophysical and geological talent present—all wishing to take part in the discussion—that the mere miner, with one notable exception, scarcely got an opportunity to put his views forward. It seemed to him that those mining men who had had experience with geophysical methods in the field could perhaps take a more balanced and reasonable view of that highly technical subject. The only mining engineer who had taken part in the discussion previously was Mr. McPherson whose experience with geophysical work had evidently been so unfortunate that his views were very much biased, and as a result he allowed his prejudices to run away with him. It was perhaps hardly the time or place to cross swords with him, as no doubt Dr. Bruckshaw would deal with the criticisms in his reply to the discussion. There were, however, a few comments he would like to make, particularly as he did not want their younger members to get the idea that in the study of geophysics they were wasting their time and that they need not bother any more about it.

It was absurd to say (as had been said during the discussion) that the mineral deposits so far discovered in the world by geophysical methods could be counted on the fingers of one hand. They had perhaps not been as numerous as some people expected or could have wished, but there had been quite a lot of discoveries made and some of them had been of major importance. It was also useless to say that those discoveries which had been made would have been made anyway in the course of time by ordinary prospecting methods ; that was one of those loose statements which could never be proved or disproved. The fact was that they were not discovered by ordinary methods and geophysics was entitled to full credit for them.

Every now and then, when hope of hearing of further discoveries by geophysical methods had almost been given up, some spectacular discovery became known which strengthened one's faith again. He was thinking of the more or less recent discovery of what appeared to be an important orebody at O'Okiep in the Cape Province. There, he understood, a new orebody lying several hundred feet below the surface had been located entirely by geophysical methods and that had been fully confirmed by diamond drilling. That orebody was some distance away from the other known ore deposits in that area. As was well known, mining had been carried on in that district for the past 50 years or more, so apparently ordinary prospecting methods had not been effective as no-one had any inkling of the presence of the new orebody until geophysical methods were employed.

In the last copy of *The Northern Miner* which he had received a few days ago he noticed that there were nineteen diamond drills at work on geophysical anomalies in the Flin Flon district of Manitoba at the present time. They evidently had considerable faith in geophysics there and thought the expenditure fully justified as a result of important discoveries which had been made in that district in the past by those methods.

The speaker said he made no claim to be a geologist or a geophysicist; his interest in the subject had been entirely as an observer from the side-lines. He had merely watched the other fellows doing the work and had benefited very much by their results when they had been successful, or had tried to justify the heavy expenditure involved when the results had been negative—as unfortunately they often were.

He was somewhat disappointed that the author did not go into rather more detail in some respects but the paper had nevertheless given him a great deal of useful information and gave everybody an opportunity of asking questions. He was sorry the author appeared to disparage the metalliferous mining industry as compared with the oil industry in the matter of geophysical prospecting. It was quite true that in the search for metalliferous orebodies operations were not on the same huge scale as those employed by the oil companies but they were quite considerable, and although what the author called 'spectacular results' might be rather rare they were of very great importance when they did occur and made up for many disappointments. He was convinced that there were numerous undiscovered orebodies still lying hidden not very far below the surface which would some day be located by improved geophysical methods in conjunction with the diamond drill.

The author deserved special thanks for putting the science of geophysics in its proper light and for emphasizing its limitations as he did throughout the paper and particularly on p.17. That side of the picture was seldom given. Most geophysicists had to be good salesmen or else nobody would employ them, consequently they were inclined to become super-optimists. It was a sad truth that the use of geophysical methods in the search for metalliferous deposits had not in the past met with the success which at one time seemed likely, with the result that some mining men were still rather sceptical of the whole business. That was a pity because there were undoubtedly many instances where those methods could be of great assistance provided one did not expect too much of them by themselves.

Perhaps he had been particularly fortunate in seeing millions of tons of high-grade ore discovered by geophysical methods in Newfoundland in disconnected orebodies spread over a distance of several miles. Some of the ore was admittedly not far below the surface and might have been discovered eventually by ordinary prospecting methods, but a great deal of it most certainly would not have been so discovered as it was buried under a deposit of glacial drift varying in thickness from 50 to 80 ft. with no rock outcropping anywhere near from which to learn the geology. On top of the glacial drift there were several feet of soft muskeg which necessitated laying platforms of timber corduroy for the diamond drills to work from. Locating ore under that sort of a blanket by geophysical methods could hardly be called a ‘ fluke ’, as it had been described by one of the previous speakers.

There were bound to be many disappointments in all geophysical work, but, he was sure that any mining engineer after reading the paper would be in a much better position to appreciate the odds for and against. It was obvious that to be of any real value geophysical surveys must be carried out on the grand scale. Several different methods would probably have to be tried out before the right one or the right combination of methods was learned. That was bound to be a lengthy and expensive process. A little knowledge was certainly a dangerous thing in geophysics, and unless the work could be carried out most systematically and with the very best equipment and highly trained personnel, it should be left alone.

The author did not say very much about geophysical surveying from the air ; the only reference was on p. 10 where he spoke of the advantages of the airborne magnetometer. The speaker wished to ask if any progress had been made in using electrical methods from the air, with a helicopter, for instance ? That had always sounded to him almost too good to be true but they had been told a year ago by one of their members, who was a well-known geophysicist, that it held out great promise.

He asked the author also to what depth he thought electrical methods were effective. From what he had seen in the field he had come to the conclusion that they were not likely to feel mineral more than a hundred feet or so below the surface. He knew of several instances where they had failed to locate large bodies of massive mixed sulphides which did not come up to the surface and although he had often asked geophysicists that question he had never obtained a satisfactory answer—for the simple reason, he supposed, that they did not really know themselves. He gathered that gravimetric methods which were coming more into use with very much improved instruments, had a much better chance of finding ore at depth than electrical methods in which the field was rather restricted.

The author mentioned the discovery of ore by drilling on an anomaly obtained by gravimetric methods in New Mexico and it would be interesting to know at what depth that discovery was made. It was later stated that with direct current it was possible to obtain any desired penetration but that the power required became prohibitive. He did not quite understand that statement but he supposed that it referred to very great depths, as surely the power consumption would not be so serious at depths up to, say, 500 ft. as to rule the method out altogether.

The author mentioned also that the resistivity of massive galena was relatively low as compared with other rocks and minerals. He had always understood that pure galena was practically inert and did not give any kick at all in electrical methods; was that correct or had he misunderstood something?

As a final note and in the hope that geophysicists would be spurred on to greater efforts in improving their methods he added that the diamond drill was in the opinion of many mining engineers still their best friend in the search for new orebodies. With more reliable methods, however, the combination of geophysics and diamond drilling should prove a very strong team indeed and would, he hoped, help to solve many of their present-day problems as to where future supplies of some of the base metals were to come from.

Mr. W. W. Varvill also congratulated the author on his clear and instructive presentation of a subject which could so readily remain incomprehensible to those not gifted with a knowledge of higher mathematics. A mining engineer like himself who had long forgotten his mathematics, except those related to pounds, shillings and pence, found it a great relief to read a paper which was not obscured in a cloud of symbols and formulae. It was not necessary to know how a motor-car worked in order to drive it and so the mining engineer must be content to accept what he was offered by the geophysicist and apply it to the work which he had to undertake. For that reason it was very important that the mining engineer should know the limitations of geophysical work and in his paper the author set out clearly his own opinion in that matter and could not be accused of the excessive optimism common to geophysicists.

The speaker had during his life been brought into contact with a number of geophysical surveys, but in every case they were hunting for small and sporadic fissure veins of base metals and gold and he did not remember a single instance where the results of the surveys led to anything but wasted expenditure. On the other hand, he admitted that geophysics, particularly with regard to oil and magnetic metals, had produced remarkable results, so remarkable that in recent years the public in general had ascribed to the geophysicist powers which hitherto had been attributed to the diviner. The use of various mysterious instruments during the war for detecting land mines, and the great discoveries on the West Witwatersrand, where the existence of magnetic strata helped the geologists to site their bore-holes successfully, led the general public to think that the mining engineer had been given a magic box which would tell him where the mineral was.

It was one of the duties of the mining engineer to enlighten the uninitiated on the question of the limitations of geophysical work, otherwise geophysicists were liable to bring discredit upon themselves by making claims which they could not substantiate, just as diviners had done in the past. He was not a believer in divining but there was no doubt some influence which could not be explained by science in its present stage of advancement, with the result that diviners in the past had claimed to be able to discover lost dogs, missing bodies, and so on.

At the last Meeting a speaker enquired about the possible use of radio in geophysics, and asked if there had been any such recent developments and to what depths of rock the waves would go. He understood radio waves could penetrate. For instance, he was told on good authority that a submarine lying on the bottom of the sea could pick up the broadcasting service without going up to the surface or sending up a floating aerial. That meant that radio waves penetrated water, which was one of the most difficult things for them to penetrate.

Dr. Dunham at the last Meeting mentioned a geophysical survey at Mill Close mine. He did not know if it was the same survey that was carried out 20 years ago, when the speaker was employed there, by an amateur who was a skilled research worker on wireless. He had some mysterious box with which he was hunting for interference with the B.B.C. broadcasting waves and which might be attributed to masses of galena. At Mill Close there was at that time a large mass of nearly solid galena and everybody knew where it was. The expert got to work and successfully traced the line of the 70-fathom level from the shaft for $\frac{3}{4}$ mile to the north where the deposit was worked. It was true he knew it was there, but he definitely got results, and everybody was excited about it. When he reached the ground over the top of the large deposit he came to an end and could get no signals beyond that point although the ore ran for $\frac{1}{2}$ mile further. One day he lost his signals altogether and that happened at a time when they were overhauling the electric pumps in the mine and the cables were dead, so that there was little doubt that the signals he had received originated in the cables. He would like to ask the author whether those electric waves would travel $\frac{3}{4}$ mile along the 70-fm. level and up the shaft, or did they go through the rock ?

The President said that much of the discussion on that interesting subject had been devoted to the more general aspects. In view of the limitations of the various methods of geophysical prospecting which had been mentioned it was clear that the assistance not only of geological knowledge but also of any of the ordinary forms of prospecting must be brought to bear upon the complex problem of finding ore deposits. The author had referred to some interesting examples of surveys in order to illustrate his points and further examples had been given in the discussion to illustrate other aspects of the problem. There must be many other members, some of whom were abroad, who had had experience of geophysical surveys, and he hoped they would contribute in writing to the discussion, referring particularly to any actual experience they had had in special cases.

As an example of what he had in mind he mentioned two cases in India. In both the rock surfaces were masked, in one case by a deep covering of cultivated soil, and in the other by a laterite capping. The outcrops of lodes were worked by the ancients and the rediscovery of those workings had been the basis of most of the mineral discoveries in the country. They were no more than deep trenches or ditches filled with rubble and in the case mentioned they were completely masked by cultivation. The method employed was a simple combination of electrical prospecting together with rapid trenching to uncover any indications and also a very careful panning of the soil. The soil examination was for particles of quartz. Using that method a very extensive series of old workings was found on three separate lines of lodes, one extending for 5,000 ft., and in no case was there any depression or indication of those old workings on surface.

In the second case it was found that quartz reefs in rotted schist under the laterite gave indications using electrical methods. Deep trenching at the points indicated uncovered a number of quartz reefs. By this means the probable extension of the Kolar Gold Field under the laterite was examined. Unfortunately no values of importance were found.

He gave those two illustrations as simple examples of the use of other methods of prospecting in combination with geophysical prospecting.

He concluded by inviting Dr. Bruckshaw to make further remarks.

Dr. Bruckshaw, in reply, said that when he first attempted to produce the paper under discussion he had as an objective to describe geophysical prospecting clearly and concisely and to show as precisely as possible what geophysics could accomplish. He had hoped that he would be able to demonstrate that there were no miracles attached to geophysics, that it was based on sound scientific foundations and when used with common sense would serve as a very useful tool in the prospecting for base metal ore-bodies.

In his introductory remarks, and also in the paper itself, he stressed what he considered to be the major considerations before undertaking a geophysical survey and also in the interpretation of the results. He stressed, for example, the well-known ambiguity in interpretation and, also, the impossibility of identifying, from physical observations made at the surface, the actual nature of the body producing the observed disturbed field. Later, some surveys were specified by Mr. McPherson in relation to the search for certain orebodies. Unfortunately, geological details of the conditions under which those geophysical surveys were carried out had not been given but he remembered quite well Mr. McPherson saying that geophysics had not discovered orebodies. As far as he could see geophysics did not discover orebodies directly. Geophysics, in fact, produced a map of anomalies and those anomalies must then be interpreted in terms of the known geology and their potential value assessed. To state that an anomaly was due to an orebody went beyond the geophysical evidence ; it was combined geology and geophysics. The examples quoted by Mr. McPherson on that occasion demonstrated very forcibly the point he had attempted to make, namely, that it was impossible to identify a body from its physical effect at the surface, and, in some measure, how not to conclude a geophysical survey.

There was one other fairly general point which he would like to make. He believed that Dr. Pickering, amongst others, stated that geophysics was very good in the hands of the geologist, a statement equivalent to saying that geophysics was a subject which should not be carried out by the geophysicist. He suspected that Dr. Pickering was thinking not of the geophysicist, but rather of the physicist. To the physicist was usually attributed the ability to make measurements ; the geologist must make the interpretation of the measurements. He claimed that when the geologist was acting in that manner he became a geophysicist and was carrying out the functions which could be performed by a competent geophysicist. The latter should possess not only the necessary physical background but also the necessary geological background.

In carrying out a survey a geophysicist could not be as familiar with the geology of any area as the geologist who had examined it in greater detail prior to his arrival. It was only natural, and desirable, that he should maintain close contact with that geologist. Nevertheless, the geophysicist should appreciate the full significance of the geological data which was presented to him. Thus, to state that the geophysicist was not the person to carry out geophysical surveys was an anomaly.

It would appear that, as a result of the discussion, the general conclusion was that geophysics was quite a useful item of the prospecting methods. There were occasions on which it had failed, and it was fairly obvious that it was not going to be 100 per cent successful. There would always be the disappointing cases, but, nevertheless, geophysics was a useful established tool and would maintain its successes at a high level in the future.

The President said that the Institution was greatly indebted to Dr. Bruckshaw for his interesting paper and his concluding remarks.

DISCUSSION ON

Recovery of Sulphur from Smelter Gases by the Orkla Process at Rio Tinto

By H. R. Potts and E. G. Lawford, *Members*

The President said that the excellent paper for discussion that evening was presented by joint authors, but the Institution was indebted to Mr. Lawford for undertaking the introduction of the paper.

Mr. E. G. Lawford said that, as there might be some present who had not had time to read the paper, he would briefly go over the flow-sheet of the process in order that his subsequent remarks and the general discussion might be more intelligible. He then exhibited a lantern slide of the flow-sheet and gave an outline of the process. He exhibited lantern slides showing open-top and closed-top blast furnaces, sulphur washing autoclave and Cottrell chambers and a sulphur storage monolith. Continuing his introduction, he said that unfortunately, when it was already too late to make alterations, it had become apparent to the authors that the title which they had selected for the paper was by no means a good one. 'The Recovery of Sulphur from Smelter Gases' might well give the impression that the sulphur was already present in the gases, whereas what the authors described in their paper was a method of smelting which gave a gas containing sulphur: the actual recovery of the sulphur from the gas was a comparatively simple matter of condensation, whereas the method of smelting whereby the sulphur containing gas was produced was a much more complex and interesting matter. They would, therefore, have preferred to call the paper 'The Production of Sulphur by the Orkla Process at Rio Tinto'.

He would like to point out, before passing on to other matters, what he considered to be somewhat misleading wording on p. 24, in which it was stated that the ore was prepared by screening on a 1-in. square mesh and that the upper limit was an 8-in. ring. That might give the impression that the ore was prepared by screening on a double-deck screen with a top deck of 8-in. mesh and a lower deck of 1-in. mesh, the undersize from the top deck and the oversize from the bottom deck being the feed for the furnace. No such method of preparation was in fact practised. All the ore screened on the 1-in. square mesh screen was previously passed through a jaw crusher: that made quite sure that no pieces of ore exceeded 8-in. ring in size—and indeed by far the greater part of the ore would be less than 6-in. ring. The reason why the authors mentioned 8-in. ring at all was because nothing larger than that size would pass through the charging bells of the furnace.

One of the great difficulties that had confronted all those connected with the process was the impossibility of ascertaining by sight or by sampling and analysis just exactly what went on inside the furnace. If they would think for a moment of the placid calm of the inside of a reverberatory furnace and its relative accessibility and then compare that with the roaring turbulence of the blast furnace, encased in its four water-jacketted sides in which there was no opening above the tuyères, they would realize just what that difficulty amounted to. In the reverberatory there were abundant openings through which the hearth could be viewed and gas temperatures taken either optically or by other means. Furthermore, the drawing of gas samples presented no insuperable difficulty. In the case of the blast furnace, however, the very nature of the plant itself and the process taking place in it made it impossible to secure accurate temperatures or analyses of the gas.

There was no doubt in his mind that if it could possibly be done they would learn a very great deal by obtaining gas temperatures and analyses at vertical intervals of, say, a foot from the tuyères up to the exit gas ports of the furnace.

Even the sampling and analysis of the furnaces gases after they left the cold Cottrells presented formidable difficulties. They had not yet found any instrument capable of providing an accurate and continuous record of the SO_2 in the exit gas, still less of the total sulphur, and as far as they were aware no such instrument had yet been devised and manufactured.

For all those reasons they had to admit, even to-day after many years of experience, that they did not know with absolute certainty what was a typical analysis of their gases under any given condition of coke consumption, smelting rate and so forth.

In spite of the difficulties and the resulting uncertainty in regard to some of the data, he and his co-author had ventured to put forward, very tentatively, a theory on how the process worked, and they had attempted an actual quantitative analysis of it. They wished to stress, however, that they fully realized the arbitrary nature of some of the assumptions and the fact that the gas analyses which they had used might not be strictly accurate, and that for those reasons no exact results were to be expected. They did think, however, that they were justified in assuming that a fair amount—to use no more precise numerical definition—of reduction was done by solid carbon.

They also believed that it followed from the theory which they had set out that the more coke used the greater the reduction by CO. In other words, they thought that there was sufficient contact time and temperature to burn just so much carbon and no more in SO_2; if the carbon in the charge were increased, more and more would reach the focus and would ultimately react with the SO_2 as CO and not as C.

That brought them to a point which they had stressed in the paper; in operating the process in a matting blast furnace they were attempting to carry out a high degree of oxidation of FeS and then within a few feet, possibly inches even, to turn the whole process into a reducing one. That meant that their pyritic smelting was not at all good and their reduction of SO_2 not particularly efficient. The very essence of true pyritic smelting was absence of carbon at the focus, so that there should be nothing to burn except FeS. The presence of carbon at once limited the oxidation of FeS and this, in their view, accounted for the poor ratio of concentration which they obtained. When those present remembered the ratios of concentration of 15, 16 and 17 : 1 which were made at Mount Lyell in the early days of the present century, and compared them with the concentration of between 3 and 4 : 1 at Rio Tinto, they would fully realize the truth of the statement regarding the inferiority of the pyritic smelting done.

The question might well be asked whether the conventional type of water-jacketted blast furnace was the right kind of plant to use for the Orkla process. They had often wondered whether it would be possible to lengthen the high temperature contact zone by using a furnace which would much more closely resemble an iron blast furnace than a matting furnace. In the conventional copper furnace no attempt was made to conserve heat; in fact the water-jackets were intended to dissipate it. It was interesting to speculate on what might happen in a brick-built furnace 50 or 60 ft. high, in which every attempt was made to generate the maximum temperature at the focus by intensive oxidation of FeS and then to conserve that heat by insulation and like devices, at the same time causing the gases to travel a much greater distance in contact with coke than they do in the present short furnaces.

The use of oxygen-enriched blast in such a furnace in order to generate a very high temperature at the focus was an interesting possibility on which to speculate.

The Norwegian operators, unlike those dealing with arsenical Iberian ores, did not have to worry unduly about the extent to which they effected reduction in the furnace and there was, therefore, not the same inducement in their case to consider departing from the conventional copper blast furnace. In Norway the gases could be efficiently treated by catalysis which had the effect of recovering substantial additional quantities of sulphur from the gases outside the furnace. Unreduced SO_2 was required in order to catalyse the sulphur combined with carbon and hydrogen and, therefore, complete reduction of the SO_2 in the furnace, even if it were possible, would not in their case be desired. With the arsenical Iberian ores, once the gases left the furnace, any further recovery of sulphur—recovery as distinct from mere condensation of existing sulphur vapour—had so far proved impossible for the reasons given in the paper.

In that connection they were afraid that the last paragraph on p. 25 might appear to conflict with the first paragraph on p. 33 and both the paragraphs required some explanation. In the last paragraph on p. 25 the expression 'exit gases' should not have been used as that term should have been reserved for gases leaving the cold Cottrell and passing out of the system, whereas, of course, the paragraph referred to gases leaving the furnace. The first paragraph on p. 38 might give the impression that they had in view some special step whereby arsenic would first be removed from the gases after leaving the cold Cottrell and catalysis then carried out. They regretted the bad wording of the paragraph and suggested the following for substitution: 'Possible treatment of the arsenic-free exit gases by catalysis'. In explanation they would say that substantially all the arsenic reported in the sulphur already condensed and was recovered before the gases left the cold Cottrells, and if it were found possible to catalyse the gases leaving the cold Cottrells and at present passing out of the system, any sulphur recovered in that manner would be substantially arsenic-free.

Catalysis of the gases after they left the cold Cottrells had at one time been considered, but it necessitated reheating those gases to 400°C. and that would have made the whole scheme quite uneconomic. They believed, however, that some catalysis might be possible at a lower temperature and that it might only be necessary to reheat the gases leaving the cold Cottrells to something like 250°C. Whether the yield of sulphur by catalysis at that temperature would cover the cost remained to be ascertained.

In Norway, catalysis of the gases was carried out in two stages. He believed that the first stage was carried out at 400°C. and recovered some 23 g. of sulphur per cu. m., mainly from carbon disulphide and carbon oxy-sulphide, and that the exit gases from the process were reheated to 260°C., again catalysed and a further 20 g. of sulphur recovered. They very much hoped that the engineers in charge at Orkla would contribute in writing to the discussion and would give full details of treatment of their gases.

The authors wished to make a few observations on the general efficiency of the Orkla process.

In spite of the relatively poor reduction of SO_2 the process used very much less coke per ton of sulphur than any of the numerous reduction processes, all of which first burned all the S in the pyritic material to SO_2 and then reduced the SO_2 by C or CO either directly or after first concentrating the SO_2 by absorption, sending a very strong gas forward to the reduction step. The reason was, of course, that the Orkla process recovered all the loose atom sulphur without reduction and, for that reason, judged on the yield of sulphur per ton of coke, it was a very efficient process.

It was much less efficient if the yard-stick applied was the yield of sulphur per ton of pyrites, because, in addition to the inevitable loss of sulphur as SO_2 and carbon compounds in the end gas, there was also the loss of sulphur in the matte. No doubt some of that sulphur could be recovered, as indeed it was in Norway, by concentrating the matte in a closed-top furnace but some sulphur must always be lost in matte and at Rio Tinto it amounted to 10–15 per cent of the total sulphur in the pyrites.

In passing to the sulphur purification, Mr. Lawford said that there was little to add to what the authors had already written. The process was a batch process and suffered from the defects inherent to all batch processes. However, they were actively experimenting with a continuous sulphur washing process and they hoped that, eventually, they would greatly improve the present method.

It would, of course, be noted that there again the presence of arsenic adversely affected recoveries as compared with those obtained from arsenic-free or low-arsenic ores. The arsenical liquors were thio-arsenate compounds and carried away their due proportion of valuable sulphur. With all its expense and disadvantages the purification process could virtually eliminate arsenic and must, therefore, be considered a highly efficient process.

To close the introduction a glance would be taken into the future.

Anyone who had had a close acquaintance with Spanish conditions since the outbreak of the Civil War in 1936, through the World War, and through the last years of partial ostracism of Spain by the United Nations, would realize the impossibility of doing any large-scale experimental work. The first and urgent need had been to keep plant and process running somehow, no matter how inefficiently. There had been recurrent serious shortages of coal and coke, at least one serious drought causing curtailment of operations through a dangerous shortage of water, and great stringency in foreign exchange with which to make purchases abroad ; in addition, shortage of labour had been very serious. Superimposed on that kaleidoscope of shortages had been a spiral of rising costs. All that had made it impossible to contemplate large-scale experimental work. However, they all hoped that conditions would change sooner or later and that experimental work would once again become possible. They did not yet regard the problem of catalysis as impossible of solution and they believed that the reduction of SO_2 within the furnace might be greatly improved.

They hoped, therefore, that in a not too distant future the Institution might have before it a second paper describing improvements and developments in the Orkla process.

Finally, the speaker said he and his co-author wished to acknowledge their immense indebtedness to Mr. L. U. Salkield of the metallurgical staff of the Rio Tinto Company in Spain. To him was largely due the reduction theory that they had presented in the paper in its present form.

Dr. S. I. Levy* said that the mechanism of the process was of great theoretical interest especially since, as Mr. Lawford had clearly pointed out, it was necessarily a compromise between smelting and treatment for sulphur recovery. Those two objects might very well be incompatible, since the furnace had to be kept

*Consulting chemical engineer to the Rio Tinto Co., Ltd.

running and it had a habit of choosing its own conditions of smelting, which might not be the most suitable for sulphur recovery.

Following the burden down the furnace, the first thing that happened when the pyrites was subjected to increasing temperature was that arsenic sulphide was driven off. If pyrites were heated in a glass or silica tube, the first product obtained was arsenic sulphide, and one could see a broad yellow ring formed on the cooler part of the tube as soon as the temperature in the furnace got to 450–500°C. The arsenic sulphide is at first pure, but sulphur begins to come off at a not much higher temperature—500 to 600°C.—and increases up to almost the melting point of the mineral. The melting point of this crude mineral is a good deal below 900°C.; heated above 850°C. it tends to slag with the silica of the tube, so that laboratory experiments had been limited to 850°C. At that temperature a relatively small proportion of the total sulphur was obtained—about 17 per cent by weight of the original mineral, and less than 40 per cent of the original sulphur content.

He had not carried out experiments at higher temperatures but he had been given results of experiments carried out in an induction furnace. Pyrites was not a material which readily accepted heat in an induction furnace, so that it was necessary to heat it in a carbon crucible, which acted as accepter. Very little more sulphur came off until the temperature approached 1200°C., when the rest of the first atom began to come off and was completely driven off between 1500 and 1600°C., a pure FeS being obtained. That became volatile at 1600°C. and began to vaporize, but it did not lose any more sulphur by heating. At 1600°C., however, ferrous sulphide entered into chemical reaction with the carbon of the crucible; carbon disulphide was formed, and the residue was an impure iron containing carbon and some sulphur.

In the Orkla furnace temperatures of 1600° were not approached. Probably a temperature of 1400°C. was obtained at the focus but it seemed likely that by the time the burden came into contact with the oxygen of the air the actual temperature of the pyrites was probably not much above 1200°C. so that it was fair to assume, as the authors had done, that the whole of the first atom sulphur was not driven off, but something of the order of 40 per cent of the total sulphur in the mineral. In that respect he thought, although there was no real experimental evidence to decide it definitely, that the authors were probably correct when they assumed that material of about the composition $Fe_7 S_8$ or $Fe_6 S_7$ reached the reaction zone.

The authors had assumed that there was some reaction of the coke in the burden with the gases. He did not altogether accept the suggested course of such reaction. He did not think carbon disulphide would be formed in the column, though some carbon oxy-sulphide might be formed in the column. However that might be, he thought they were all agreed that the great bulk of the carbon reached the reaction zone just above the tuyères without much change. That view was confirmed by an observation recorded by Mr. Salkield on one occasion when a furnace was stopped suddenly, and the charge when cold was analysed foot by foot for carbon. It was found that as low as 4 ft. above the focus the carbon was still nearly up to the original proportion in the charge, so that the actual reaction of carbon in the column was quite small until the zone of extreme activity just above the tuyères was reached. In that zone practically the whole of the smelting reactions had to take place.

The views as to those reactions which the authors of the paper had put forward could be confirmed to some degree by looking at the oxygen balance. It was quite certain that practically all the oxygen was consumed within 2 or 3 ft. of the tuyères, and that the oxidizing action took place almost at once. It was possible to get an oxygen balance because the products of the reactions were known and an estimate of the oxygen entering the furnace could be made from the figures which the authors had provided. They gave the volume of exit gases per ton of mineral charged, from which it was easy to calculate the volume of air used, since there was no change in volume except in two respects. In the first place a certain amount of oxygen was removed by iron, which went into the slag in the form of ferrous oxide. That could be calculated from the figures on p. 5. Secondly, there was a slight increase in the volume of gas due to formation of carbon monoxide and carbon oxy-sulphide. Allowing for those two changes in volume, the air used per ton of pyrites was about 60 cu. m. more than the volume of the exit gases. Taking the oxygen from that, one could get a figure for the total amount of oxygen entering the furnace.

The authors calculated how much sulphur was burnt at the focus to form SO_2, and they gave the figure for iron in the slag, so that it could be calculated how much oxygen was available at the focus for carbon. It seemed fairly clear that there was enough oxygen to burn the greater part of the carbon to carbon monoxide. Probably something of the order of two thirds to three quarters of the carbon was burned to carbon monoxide in the reaction zone at the focus. The rest of the carbon therefore must be oxidized by sulphur dioxide. The authors had concluded that sulphur dioxide must be reduced to sulphur to some extent by solid carbon, and that conclusion was confirmed by calculation of the oxygen balance, which showed that the whole of the carbon was not converted to carbon monoxide, since there was not enough oxygen; it followed that a certain amount of carbon must be consumed by reaction with sulphur dioxide.

It was clear that the ideal aim would be for sulphur dioxide to be reduced entirely by carbon. One could only bring the iron monosulphide into reaction by burning it with oxygen, therefore sulphur dioxide must be formed from all the FeS which was brought into the reaction zone. That being so, the most effective method of getting sulphur would be to reduce that SO_2 by solid carbon and that could be done, as the authors had pointed out, in the column above the main reaction zone if the temperature were high enough. The reduction of SO_2 by carbon did not proceed quickly much below 1200°C., but at 1200°C. it proceeded quite easily and quickly. If the temperature could be raised sufficiently, therefore, sulphur dioxide could be reduced by carbon in the column above the focus; the carbon would not then descend to the focus and consume oxygen. That would allow a more effective use of air in oxidising the mineral itself and would give a better grade of matte. The whole of the study of those reactions pointed to the conclusion that the only practical way of getting better recovery of sulphur or a lower consumption of carbon for the sulphur recovered would be to get a higher temperature.

Whether or not that was practical was not entirely clear. It was quite certain that a water-jacketted furnace must remove a large quantity of the heat available and must keep the temperature down at the point where it was required to be highest. He saw that brick lining and some insulation of the water jackets had been attempted but had not been very successful. If some effective insulation between the water-jacket and the charge could be devised a higher temperature was bound to follow. Another 100°–150°C. would probably make a very great difference, and might lead to a much better matte grade and much better sulphur recovery, by bringing about reduction of sulphur dioxide in the column with a relatively low use of carbon. It was important to point out that that involved using less air per ton of charge, which in itself seemed a little contradictory. It would be difficult to get higher temperature and at the same time use less air, but if better results were to be obtained it would be necessary to cut down the amount of air entering the furnace per ton of mineral charged.

The next point was the treatment of the exit gases, and particularly the effect of the arsenic present. The gases contained up to 250 g./cu. m. of free sulphur, with 5 to 6 g. of arsenic. The effect of the arsenic was to make catalysis impossible. The only arsenic-sulphur compound which could exist in the vapour state was arsenic trisulphide; that was considerably less volatile than sulphur, and tended to separate at a higher temperature. When it separated with sulphur, an extremely viscous mixture was formed. The only practical way of dealing with it was to keep the gases hot enough to prevent any condensation at all until the whole quantity could be cooled down quickly to such a degree that the bulk of the sulphur separated with the arsenic sulphide and formed a solution sufficiently mobile to flow without too much difficulty. Sulphur containing even a moderate proportion of arsenic was not easily handled in a liquid state, because it was extremely viscous even at temperatures considerably above the melting point. The method adopted at Rio Tinto was therefore to remove mechanical dust from the gases at a temperature well above that at which arsenical sulphur began to deposit. If the temperature was allowed to get much lower than 450°C., extremely viscous deposits of high arsenical sulphur might be formed. The gases were therefore freed from dust at a temperature as near to 450°C. as possible, and then passed through a condenser to bring the temperature down to 120–140°C. At that temperature most of the sulphur separated with practically all the arsenic, so that one got a liquid sulphur containing 3 per cent or less of arsenic. That could be handled fairly comfortably, but the effect was that although the arsenic was removed from the gases without too much mechanical trouble, the gases had to be cooled so much that they would no longer react in the presence of a catalyst. At temperatures as low as 150°C. it was impossible to cause the sulphur dioxide, carbon disulphide and carbon oxy-sulphide in the gases to react to produce sulphur at any useful rate, and all the combined sulphur in the gases was therefore lost.

He would like to say a word or two on sulphur refining. The removal of more than 2 per cent of arsenic in one operation by such a simple process as the authors had described to give a substantially pure sulphur was a remarkable achievement. It was all the more remarkable in that while arsenic sulphide was completely soluble in molten sulphur neither arsenic sulphide nor sulphur itself was soluble in water. The curious thing on which the success of the process depended was that calcium thioarsenate was extremely soluble in water. That was rather unexpected, because the corresponding oxygen compound, calcium arsenate, was practically insoluble in water. Related compounds like the calcium phosphates were also completely insoluble in water. It was surprising to find that calcium thioarsenate was so soluble that even when the solution was concentrated to a high density the thioarsenate would not crystallize. It had only been possible to make it crystallize by concentrating to a high degree and then cooling and adding alcohol. The crystals which then formed on standing were clear and brilliant, but rapidly became yellow and opaque in the air. The thioarsenates were decomposed by carbon dioxide, with separation of sulphur and arsenic sulphide. The latter was non-poisonous by reason of the fact that it would not dissolve to any degree in acid, and so was not attacked in the body but passed through unchanged. The only poison hazard in that process arose from the possible presence of H_2S.

In concluding his remarks the speaker joined the authors in paying a tribute to the valuable work of Mr. Salkield, who was the first to point out that in the blast furnace operation substantially all the oxygen and a great deal of the carbon must be consumed in a relatively small zone just above the tuyères. Mr. Salkield and the late Mr. L. A. Lawrence had to face the great difficulty of bringing a new process into operation on a very large scale with the enormous complications due to the large amount of arsenic which was present. The success of the process at Rio Tinto owed a great deal to their efforts as well as to the work of the authors of the paper.

Col. J. Cross Brown* also thanked the President for his invitation to attend the first presentation to a technical institution of a paper on the Orkla process. Although he was unable to contribute any remarks of technical value he would like to make some general comments.

When, in 1905, he visited the smelters at Rio Tinto with his father—the late Mr. Nicol Brown—who commented on the great loss of brimstone issuing from the chimney stack, he little thought that 20 years later he would become associated with his friend Mr. N. E. Lenander—managing director of the Orkla Company—in the early development of what is now known as the Orkla process.

Mr. Lenander told him of the encouraging results they had obtained at their works at Oscarsham on the Baltic from experiments on the practically non-arsenical pyrites from the Orkla mine and how his company wished to investigate the possibility of extending the scope of the process to pyrites from the Peninsular containing about ten times more arsenic.

As a result of those conversations it was decided to erect a cylindrical furnace at Mina de S. Domingos in Portugal (operated by Mason and Barry, Ltd.) but that was not a success although some useful information was obtained. A few years later Mr. Lenander, to whom all credit must be given for developing the process, started his first pilot plant at the Orkla mine, where he invited representatives of the Rio Tinto and Mason and Barry companies to see some Rio Tinto and San Domingos arsenical ores, which had been shipped from the Peninsular to Norway, put through the pilot plant. Accompanied by Mr. R. M. Preston and the late Mr. G. Wynter Gray he went to Orkla and could assure the audience that it was a great moment when he saw bright yellow brimstone flowing out of the experimental unit.

He remembered on the way home Mr. Gray saying to him: 'We have a smelter at Rio Tinto and we are going on with this process. You have not got a smelter at San Domingos so I suggest we should let you know our experience and if you decide to go ahead we should be only too glad to help you'. He would like to pay a warm tribute to the broad-mindedness of that Member of the Institution.

The reports by the Rio Tinto Co. being satisfactory and there being a regular demand for brimstone for use as an insecticide in the wine industry of Portugal, the first of two brimstone plants was erected at S. Domingos in 1934, the average production being 10,000–11,000 tons annually.

It was a disappointment that the catalyser chamber which was embraced in the first unit at S. Domingos never functioned, as the catalytic mass peeled off owing to the high arsenic present. Had

* Messrs. Mason & Barry, Ltd., London.

it been a success the recovery of brimstone would have exceeded 70 per cent, and he was glad to hear that evening that it was hoped sooner or later to overcome that problem.

He was very interested to hear about the elaborate washing plant employed at Rio Tinto. That was another problem at S. Domingos when designing the first unit—should they adopt such an elaborate washing plant as at Rio Tinto or should they pass the bad coloured viscous brimstone through a tank and wash it with milk of lime as was done at Orkla when dealing with practically non-arsenical pyrites? They took the chance, and except on the first day's operation, when it completely failed, the washery at S. Domingos had never given the slightest trouble, the arsenic in the brimstone exported being reduced to 0·05 per cent. Dr. Levy had mentioned that arsenic sulphide was non-poisonous and so far they had never had any complaint on that score. They were thus relieved not only of large capital expenditure on an elaborate plant but working costs were very much less than in the process used at Rio Tinto. That was not a criticism, for no doubt there were good reasons why Rio Tinto should eliminate arsenic to a minimum.

The fuel mentioned by the authors was coke, but war conditions sometimes demanded alternatives, and at S. Domingos for some years charcoal was used instead of coke, which was unobtainable. As far as a non-technical man could see, it worked quite satisfactorily and consumption per ton smelted was as low as coke. Charcoal, however, was much more brittle than coke and there was a good deal of loss in fines which could not be put into the smelters.

Zinc—always a bugbear to smelters—caused their managers considerable trouble. However, a remedy was found which was not only a remedy but also brought in a satisfactory revenue. He referred to the substitution of gossan, containing approximately 27 per cent of ferric oxide and 63 per cent of silica, for a large proportion of ordinary silica flux containing about 95–98 per cent of silica. That not only helped to purge the zinc but also contributed some gold and silver to the copper matte. There was one more problem which had recently been tackled by the Orkla company and some years ago at S. Domingos. He referred to making the slag—which contained about 40 per cent of iron and very high silica—into a suitable product for the iron works. Just before the war quite a considerable tonnage was shipped from Norway to Germany for iron smelting but the circumstances then were quite exceptional. He hoped that something might be evolved in that respect and reported upon in the next paper referred to by Mr. Lawford.

Prof. C. W. Dannatt said that he assumed the 'sulphur' in the furnace feed (Table I) to be the residue from the sulphur-washing plant, and it would have been preferable to refer to it as the sulphur residue. It was slightly confusing to see in this Table two materials with identical sulphur contents.

He thought the authors had been unfair to themselves in the footnote to Table IV where they suggested two methods for the calculation of recovery. Admittedly the difference between the two figures (0·5 per cent) was almost insignificant, but the higher figure was correct, for any sulphur recovered from the secondaries must have originated in the primary feed. The secondaries merely formed a ' circulating load '.

He had been unable to find any figure for arsenic in the exit gases and he asked whether the authors could indicate its quantity. Presumably it would be far too low to cause any difficulty if those gases were catalysed. He was interested to hear that catalysis of the gases was under consideration for he understood that a double catalytic treatment was given at Orkla and that it was highly successful. The chief problem would appear to be the provision of additional heat to the exit gases before catalysis but, as the temperature required was not high, it might prove possible to arrange some system of heat exchange. In that connection there was another point of interest.

The authors stated (p. 2) that ' the aim of the process is to produce as much sulphur dioxide as possible at the focus '. That could be done by increasing the oxygen supply or by decreasing the coke, but the latter alternative would give less carbon monoxide for reaction with the larger amount of sulphur dioxide. The former alternative would mean stronger blast. On p. 3 the authors stated that ' the high temperature zone does not extend for a sufficient distance above the focus to give efficient reduction '. That zone could be extended by increased blast, possibly using an additional row of tuyères above the existing ones. The objection to the use of stronger blast was that it would give a ' hot top ' and there was ' a limit to the temperature of the gases entering the Cottrells ' (p. 25). The advantages, however, were evident and if the excess heat in the furnace gases could be transferred to the exit gases prior to their catalysis, there should be an important increase in recovery.

Mr. Stanley Robson said that he spoke without the full study of the paper which it deserved. He would, however, like to say how much he appreciated that and the other papers recently presented to the Institution by the Rio Tinto staff, in all of which they had given their methods and descriptions of equipment in detail, and also their own explanation of how the several processes worked. Such papers remained of permanent value even if later conclusions did not agree with the views which were then expressed. In the paper under discussion the authors had ventured to give their ideas on the reactions which took place in a blast furnace and when one remembered the many discordant views as to the reactions which went on in that type of equipment, one realized that some courage had been shown.

That was particularly true in the present case, as the so-called loose or labile atom of sulphur of iron pyrites FeS_2 was capable under suitable temperature conditions of being reduced to FeS and additional chemical possibilities were therefore superimposed on the reactions which occurred in an ordinary iron blast-furnace operation. The authors pointed out the probability that the final product of the thermal dissociation of the FeS_2 in the Rio Tinto blast furnace would not be FeS but a substance of the formula $Fe_nS_n + 1$ where $n = 7$. It was to be noted, however, that a whole range of iron sulphides—Fe_3S_4, Fe_4S_5, Fe_5S_6, Fe_6S_7 up to $Fe_{12}S_{13}$—all of which were said by sponsors of repute to have been separately isolated, might therefore exist at various levels in the furnace, and, of course, in any shaft furnace under blast there was a great irregularity in the temperature and pressure zones, and in the distribution of the products within the equipment even horizontally as well as vertically.

In general he did not feel that a paper in which a precise analysis of the chemical reactions was attempted was complete if it was based solely on gravimetric factors. A thermodynamic survey was necessary to prove the probability of many of the reactions which were suggested, but that was almost impossible in the present paper as the calculations would be too involved, and one had, therefore, to be guided by deductions on such evidence as was available. The authors had realized that limitation very clearly and put many reservations on their conclusions. When one remembered that many years ago even Lothian Bell had wrongly assumed that the ratio of $CO : CO_2$ of 2 : 1 in the exit gas from an iron blast furnace was necessary for best smelting conditions, basing his conclusion on what appeared to be straight-forward chemical reasoning, and that in recent years the latest experience had reduced the $CO : CO_2$ ration to 1·4 : 1, it would be seen that the authors were justified in avoiding more specific claims than they made.

The authors stated that the recovery of 55 per cent of the sulphur available in the pyrites in the form of brimstone was a low figure. That that was the case was clear whatever the chemical explanation might be, but Mr. Lawford, in his introduction to the paper, pointed out that the Rio Tinto Co. was concerned with economic recoveries and it was from that point of view that the metallurgist had directed his attention to the problem. It was true that there were processes which made a much higher recovery of sulphur in which the first stage was the isolation of the SO_2 from the furnace gas. The merit of the Orkla method under Rio Tinto conditions, however, was that a yield of as much as 3 tons of sulphur per ton of coke was obtained, compared with 2 tons of sulphur per ton of coke by those other processes. He, indeed, doubted whether the authors in their modesty had not under-stated the position. The price of coke in many cases being very high, the importance of economy in its use was very great, and the process described took full advantage of that fact.

The main loss of sulphur appeared to be in the final gases. From the analysis given in the paper it seemed that the exit contained about 8 per cent SO_2, 0·4 per cent H_2S, 0·75 per cent CS_2, and 0·2 per cent COS by volume. It occurred to him that some thought should be given to the recovery of the CS_2 as such. Sulphur in that form was much more valuable than sulphur as brimstone, and it seemed possible that some process of absorption might be conceived which would be economic. Carbon bisulphide was used in the processes for making rayon and a very large tonnage of sulphur was used for that purpose. If the CS_2 was present in the exit gases in the quantity reported, and if it was proved that the use of more coke would increase its production, it might be worth while considering an increase in its production by that means as an integral part of the operation.

He would not comment on the means used for the purification of the sulphur other than to express an appreciation of the neatness of the operation. To start with a material containing 2 per cent arsenic and by relatively simple means to finish with a final product containing not more than five parts per million was good work. It was a pretty process and the Rio Tinto Company were to be complimented. The influence of the arsenic on the whole of the processing from the ore to the final sulphur was also of great interest and had been fully described.

He noted that the authors believed that the carbon was removed in the lower zones of the furnace and not in the upper zones and that a lengthened lower zone would help. He was not sure, however, that the steps which were forecast for the improvement of the process covered all possibilities. He would have thought that one way of increasing the hot zone of the furnace was to put in more heat units by pre-heating the blast. Also, while he appreciated that it was impossible to pre-heat the pyrites itself to any extent, it might be possible to pre-heat to a relatively low extent, say 200°C., without bringing in all the difficulties which Mr. Lawford had imagined. If these steps were taken they would certainly add more heat units and so increase the hot zone and improve the operation.

Another thing which occurred to him was that the exit gases, after suitable cleaning with or without removal by absorption of materials like CS_2, could be recirculated through appropriate heat exchangers into the furnace itself. There would, of course, have to be a 'bleed off' as simple arithmetic indicated, but it might be possible by that means to increase the final yield of sulphur to an extent comparable with the addition of a catalyst stage capable of a 50 per cent efficiency. He was very conscious that his was a rather scrappy contribution to the discussion of a paper which merited much more close attention than he had been able so far to give it. He also appreciated the reference made to Mr. Salkield with whom he had some association some years ago.

The President said that as it was getting late he hoped the authors would agree to reply in writing later.

The discussion had been extremely interesting and further written contributions would be most welcome. He thanked the authors for their excellent paper.

CONTRIBUTED REMARKS

Mr. E. Rich: Having been associated with the S. Domingos plant from its inception and for a number of years, and on many occasions having had the pleasure of technical discussions with the authors on this process, I am intensely interested in any improvements that may arise in the future, as I feel that the process as a whole presents a very large field for development before attaining eventual perfection as a means of recovering sulphur and other valuable products from massive pyrites.

My days as superintendent of the above plant were spent in a constant struggle to improve the recovery efficiency and in dealing with a multitude of difficulties and problems which otherwise could be regarded as a chemical engineer's paradise! In fact, the advice of a Norwegian friend will always be remembered to the effect that if I looked after the mechanical and electrical sides the chemical would look after itself!—hardly as simple as all that, but true up to a point, which experience showed was the limit we could ever hope to attain with the particular design of plant installed. That limit appeared to be a final maximum recovery efficiency of about 60 per cent of the sulphur in the ore, being based on refined sulphur ready for the market.

Such a figure, though considered good practice in view of the very disturbing effects caused by the presence of zinc, arsenic and lead in the ore, compares with a considerably greater recovery regularly obtained at Orkla, where the most disturbing element of the above three, namely arsenic, is practically absent. As the authors aptly point out, therefore, it has long been felt that the conventional type of water-jacketted copper blast furnace is not the most suitable apparatus in which to treat massive pyrites of the types generally found in the Iberian Peninsula. Nevertheless, that type of furnace has given satisfactory results at Orkla, though a modified type would possibly give the same recovery but at a cheaper cost of producing the sulphur.

The principal reason for such a divergence in the recoveries is to be found in the relative amounts of volatile sulphur to be obtained from each type of ore, this averaging about 170 kg./ton for Orkla ore and up to 220 kg./ton for Iberian pyrites. The former, moreover, contains only traces of arsenic, compared with anything up to 0·8 per cent in the latter, a fact which also exercises a considerable influence on the readiness with which the volatile sulphur is driven off by heating. The distillation of this sulphur in the upper zones of the smelting column constitutes one of the chief phases in the process, and one which profoundly affects the whole series of subsequent chemical reactions taking place in the smelting zone or focus of the furnace. The time necessary and heat required to drive off this sulphur so control the remainder of the process that all efforts to improve the efficiency of reduction of SO_2 by carbon should be preceded by alleviating in the first instance the work of distilling off the free atom sulphur. As far as can be ascertained, experimental evidence indicates that the upper limit of the focus or heat-generating zone of the furnace does not extend to more than 4 ft. above the tuyères, the remaining 12–18 ft. of column being used by the charge in absorbing heat. As the authors indicate, there is therefore insufficient contact time for all the coke to burn in SO_2, or, expressed differently, there is an insufficient temperature prevailing to allow this reaction to proceed briskly. The result is that what reduction of SO_2 there is takes place chiefly by means of CO, a substantial amount escaping reduction altogether and finding its way into the waste gases. Furthermore, a good deal of the coke burns directly to CO_2 and is lost as such; in fact, unless some coke does burn in oxygen at the tuyères, insufficient heat is generated in the focus and recoveries drop along with the throughput. It is therefore almost axiomatic to state that if we can increase the focus temperature, and the contact time of SO_2 and carbon at this temperature, we shall more nearly approach the best recovery possible by using only 1 atom of carbon per atom of sulphur produced, from direct reduction of SO_2; in other words by the simple equation $SO_2 + C = CO_2 + S$.

As the authors have stated, however, the possibility of doing this by increasing the coke on the charge does not pay, as it results in less oxidation of ferrous sulphide owing to the hot carbon having a much greater affinity for the oxygen of the blast than the molten ferrous sulphide. On the other hand, it follows from this that if all the oxygen of the blast is used in oxidizing sulphides and does not contact any carbon in the first instance, then all the sulphur dioxide reduced will be recovered by the above equation, i.e. in the carbon/sulphur ratio of 1 : 1, even if some of the CO_2 produced by that reaction should be subsequently broken down to CO.

The next possibility is to increase the intensity of oxidation by supplying more oxygen, but while that doubtless increases the focus temperature it reduces the contact time owing to the higher gas velocities produced, if air is used as the source of the oxygen. The alternative to prevent the latter evil would be to use an oxygen-enriched blast, but its high cost would probably make such a procedure uneconomical, and not as satisfactory as another possibility referred to later. Furthermore, increasing the blast excessively will result in some of the distilled sulphur being burnt owing to penetration of oxygen high up in the column. Thus, there is an optimum amount of oxygen which any one furnace will take, this figure varying little up or down from 350 kg. per ton of pyrites in the Orkla process to probably as much as 550 kg. for the purely pyritic smelting process employed at Mt. Lyell many years ago. That high figure was no doubt arrived at by trial and error, and explains why it is not possible to recover sulphur by smelting purely pyritically with a minimum of coke, seeing that there is more than enough oxygen to satisfy the amount of coke and ferrous sulphide available. The balance of the oxygen then consumes volatile sulphur, the uncombined sulphur remaining in the large volume of gas emitted being too dilute to be cleaned effectively in the electrostatic precipitators, and thereby fouling the condensers which follow.

Finally, hot blast might be used in an attempt to increase the focus temperature, but that would certainly be accompanied by a substantial increase in gas volume for the same amount of oxygen per ton of ore smelted. That in its turn would cause increased gas velocities and, therefore, reduced contact times. Moreover, the hot blast would no doubt tend to promote simple melting of ferrous monosulphide without supplying the requisite oxygen for smelting it at the same time, the result being an increased flow of dilute matte. One has thus to bear always in mind that the principal smelting action taking place inside the furnace, namely the oxidation of the ferrous sulphide, depends largely on the rate of percolation of that product when molten over the silica skeleton. This is entirely dependent on gravity and cannot be altered, hence blast volumes can only be varied within a moderately narrow range. It is therefore significant that the best results from the Orkla furnace are generally obtained during periods of frosty weather, when air temperatures are below freezing and the moisture content is down to about 3 g./cu. m. Recoveries and throughput during those conditions are normally better than they are during the summer months when the air temperature is up to 40°C. and the moisture up to 10 g./cu. m. A chilled blast is thus to be preferred to a hot one, as, volume for volume, it allows a greater weight of oxygen to be supplied per unit of time and furnace area.

The conditions to be satisfied, therefore, are such that a high focus of heat must be maintained conjointly with a low gas velocity in the column, so that a high percentage of the carbon charged shall reduce the SO_2 formed in the carbon/sulphur ratio of 1 : 1, and not 2 : 1 as happens when CO is the sole reducing agent. It would thus appear that one of the first steps to be taken to achieve this object is to assist as many of the endothermic reactions as possible by external heat and hence to assure the charge entering the focus at a substantially higher temperature than at present. As already mentioned, the distillation of the free atom sulphur is by far the most endothermic reaction in the process and some observations on it may be of interest.

Let us consider, therefore, what happens when a stone of cold pyrites drops into a furnace where the temperature is standing at about 400°C. and the concentration of sulphur vapour runs at about 200 g./cu. m. of gas at N.T.P. There must be an immediate condensation of sulphur on that lump, contaminated to a greater or lesser degree with anything up to 8 per cent of arsenic, or say 6 per cent of the equivalent pentasulphide in which form that element is carried by the gases. This film of arsenical sulphur will adhere to the lump during the whole time that its temperature is being raised, the concentration of the arsenic gradually increasing until a point is reached when the film is almost entirely arsenic pentasulphide, which on further heating continues to lose sulphur until the trisulphide is formed. That film finally gets driven off at about 700°C. It would therefore appear that the transfer of heat to, and the free evolution of the volatile sulphur from the stone are hindered by this film of condensate forming on it. Furthermore, the arsenical sulphur going through a pasty form in the early stages of its condensation no doubt serves as a means of collecting dust particles, which will cause the film to thicken. Conditions are therefore quite adverse to a rapid dissociation of the pyrites, which will necessarily sink well down in the column before all the volatile sulphur has been lost.

In an effort, therefore, to formulate an idea as to how long under varying degrees of heat it would take for the free atom sulphur to leave lump pyrites, I carried out a series of experiments consisting of heating samples of pyrites out of contact with air by being immersed in fine charcoal. The thickness of the charcoal surrounding these specimens averaged 25 mm. and the specimens themselves were taken at random and roughly broken down to 60 mm. cubes, being then placed in thin steel boxes with ample provision for free evolution of gases. The boxes were inserted into an oil-fired furnace, pyrometer controlled, and heated at definite temperatures for known periods. On completion of the tests, the boxes were removed from the furnace and allowed to cool, the ore contents being separated from the charcoal and assayed. It is fully realized that the conditions were not similar to those prevailing in the blast furnace, the emulation of which on a laboratory scale was more than could be managed at the time. On the other hand, when it is realized that the boxes overall measured only 120 mm. cube, that their contents were not subjected to the deposition of an arsenical film, and that there are many lumps of ore in the charge which are equivalent 200 mm. cubes, then it is reasonable to suppose that the box and contents would be equivalent to the average piece of pyrites in the charge.

In every case therefore, the boxes were placed in a cold furnace and gradually brought up to the required temperature, the furnace being then kept at this heat for the determined period, varying from 2 to 6 hours. The first series consisted in heating groups of 4 specimens for 2 hours at temperatures of 600°, 700° and 800°C. respectively, the sulphur assays before and afterwards being as follows :

Temperature	*Per cent S before heating*	*Per cent S after heating*
600°C.	49·8	47·6
700°C.	47·2	44·1
800°C.	46·3	41·0

The second series consisted in heating a further group of 3 specimens to 800°C. for 4 hours, the result being for each individual specimen :

Weight of cold specimen, g.	*Per cent S before heating*	*Per cent S after heating*
1310	47·6	38·2
1885	43·7	37·8
1215	49·6	38·3

The above experiments quickly led to the conclusion that some specimens appeared to give up their volatile sulphur more readily than others, and as this may have been due to variations in their original sulphur contents a further series of tests was then carried out on samples spalled off the same ' mother 'specimens. This therefore gave a greater degree of homogeneity, and the heating was then carried out for 1 hour only on the first specimen, 2 on the second, 3 on the third and 4 hours on the last, the complete group being put into the furnace simultaneously and the time counted from the moment when the temperature had reached the desired limit. The results were as given in Table IX, all the heating being at 800°C.

TABLE IX

	Weight of cold specimen, g.	*Time of heating, hr.*	*Per cent S before heating*	*Per cent S after heating*
3rd series	1043	1	50·4	48·7
	1038	2	49·1	42·6
	1030	3	49·0	38·2
	1028	4	49·1	37·8
4th series	830	1	49·7	49·2
	1018	2	50·0	46·4
	820	3	50·1	39·4
	1010	4	50·5	36·7
5th series	1023	1	50·0	47·2
	988	2	49·6	39·4
	1022	3	49·8	37·6
	1047	4	50·1	36·9

Further work of this nature was then undertaken at lower temperatures, it being revealed that complete removal of all the available volatile suphur at 650°C. would take 8 hours, and 6 hours for its evolution at 750°C. It may therefore be inferred that the complete dissociation of the pyrites in the smelting column will take a minimum of 4 hours and possibly as much as 8 hours in a furnace running 'cold'. With the top then at 400°C. and a focus upper limit at say 1000°C., the mean is about 700°C., so we may conclude that many pieces of pyrites in the charge must be subjected to a full 6 hours' heating before they can effectively lose all volatile sulphur. The average rate of descent of the charge is some 2 ft./hr., hence about 12 ft. of the charge column must be traversed by the ore before all the volatile sulphur is evolved. Practical experience coupled with the low SO_2 reduction efficiency very often encountered leads one to the conclusion that dissociation is in fact taking place all the way down the column and frequently has not ended by the time the upper limit of oxygen penetration has been reached. Burning of volatile sulphur in oxygen of the blast can then take place. It will now be clear that in addition to dissociation taking so long, with the resulting likelihood of burning volatile sulphur, one has to contend with the general cooling.down effect such as endothermic action has on the contents of the whole charge column. This cooling effect causes all the charge constituents to enter the focus comparatively cold, chief amongst them at this stage being the silica, which is insufficiently hot on entering the oxygen zone to react strongly with any ferrous sulphide oxydized in trickling over its surface. That explains why at times silica will be found to accumulate in the furnace, and is one of the direct results of a cold focus. Such is the picture with regard to the dissociating pyrites—to be followed by a similar though less adverse set of conditions established by the dissociation of the limestone. This reaction starts at about 800°C. and is completed at just over 1000°C., the time necessary being about half that for an equal sized piece of pyrites. Hence it would appear that the limestone takes over when the pyrites has finished and thus contributes further towards keeping the upper limit of the focus at a relatively low temperature.

The combination of all these effects, increased by that of normal heat exchange from the rising stream of hot gases into the remainder of the descending charge column, results in that column acquiring a depth which varies little up or down of 17 ft., when the rate of charging is regulated so as not to allow the top temperature to drop below 400°C. Those are the conditions prevailing under normal circumstances with the Iberian ores ; occasionally it is possible to drop the top temperature to 350°C. and thus obtain a slightly improved recovery which results from the column deepening to about 18 ft. and a better heat exchange being obtained. Unforeseen and unpredictable increases in the arsenic content of the ore soon make a change back to the higher temperature necessary, otherwise there is a risk of bringing about an accumulation of condensed arsenic at the top of the charge, resulting in restriction of the gas flow. So far, therefore, moderate increases in the volume of blast commensurate with furnace dimensions have not given very marked improvements in recovery and it would seem that, as already ıentioned, to create a deeper column and thus increase both focus эmperatures and contact times, the endothermic reactions taking lace should be assisted by heat applied from sources other than he focus of the furnace.

To apply any such heating process to the cold charge by means f a retort type of furnace such as is used for coke manufacture 'ould be very costly. We are therefore led to investigate the ossibilities of utilizing the waste heat carried away in the final aseous effluent of the process, and this would appear to be all the ıore attractive when it is realized that such an effluent is almost ntirely arsenic-free and hence would lend itself readily to catalysis 'hen it is heated up to say 350°C. and provided the CS_2 : SO_2 ratio ı suitable. For reasons which the authors have clearly set out, it is ot possible to catalyse any of the gases in their passage from he furnace to the condensers owing to their arsenic content, but ıch gases, once rid of this contaminating element, are readily andled. It would seem, therefore, that the heat to be imparted ɔ them for this purpose—which must come from external sources— ould be utilized in preheating the cold charge rather than being ıore or less wasted in generating steam—this in order to reduce the эmperature to a point where the liberated sulphur may be collected y condensation, and precipitation in an electrostatic filter.

A study of the heat requirements for such a step reveals that all he charge could be warmed up to about 250°C. by merely blowing eated waste gas through it after passage through a catalyser.

This would leave the charge as a final gaseous effluent at about 40°C. or at the same temperature as it leaves the plant now ithout any catalyser. The cold charge would no doubt collect a ery small amount of the precipitated sulphur in the catalysed ases, and if, after so doing, it passed directly into the smelting ırnace proper, such sulphur would be evaporated along with the olatile sulphur. The result, however, would be that with all the harge entering the furnace at 250°C. the column would at once eepen to a depth estimated at between 25 and 30 ft. if the same ırnace exit gas temperature of 400°C. were maintained. The heat arried into the furnace by the warmed charge per ton of ore melted is calculated at 68,000 cal. and compares with 108,000 cal. эquired by the combined heats of dissociation of the pyrites and ıe limestone. More than half the total heat required for the ndothermic reactions taking place would therefore be supplied, nd in addition the complete charge would be dried.

The final effect on the focus temperature of smelting a preheated ıarge and one entirely devoid of moisture would be to raise the ›mperature there considerably, say as much as 150°C. That incre- ıent is all added to what must be at present a small ' differential ' : the driving force behind the affinity of SO_2 for the hot carbon. his affinity should be increased as a result. It is believed that ıch a modification in the process would enable recoveries of) per cent of the sulphur in the ore to be regularly obtained from the furnace alone and the process made less subject than it is at present to variations in the volatile sulphur, arsenic and moisture contents of the charge being smelted. Also—and not by any means a minor consideration—a more active focus would result in a higher grade of matte being produced, the manipulation of the furnace for such a result being easier and the extra sulphur dioxide so released being more actively reduced, owing to the prevailing higher temperatures.

The authors' concluding remarks in their paper are therefore very much to the point in that there is much room for improvement and research in this most interesting process. At the moment we are primarily concerned with improving the furnace recovery of the sulphur and in the application of catalysis to the waste gases. Following on that, there is a substantial amount of sulphur to be recovered from the washery effluent at present going to waste, this basically alkaline solution readily reacting with the acidic final waste gases to extract some of the sulphur carried by each. The eventual economic realization of all these possibilities should enable recoveries of 75 per cent to be obtained, and much interest will be aroused by a further paper on such developments, and on any others in the big field which by-products may occupy in the future.

No. 512 JULY, 1949

BULLETIN OF THE INSTITUTION OF MINING AND METALLURGY

Principal Contents :

THE ROLE OF THE INSTITUTION IN PRESENT-DAY EDUCATIONAL DEVELOPMENTS
(Presidential Address of W. A. C. NEWMAN)

REPORT OF THE PROCEEDINGS AT THE ANNUAL GENERAL MEETING, 19th MAY, 1949

MANAGEMENT IN INDUSTRY
By F. G. HILL, *Member*

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No. 512 **JULY, 1949**

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INSTITUTION NOTES

Members of Council

The Council, at their meeting held on 9th June, 1949, decided to co-opt Mr. E. D. McDermott, Mr. Thomas Pryor, Dr. S. W. Smith and Professor S. J. Truscott, *Past-Presidents*, as Members of Council for the Session 1949–50, under the provisions of Bye-law 23. A full list of Council and Officers of the Institution is published facing the contents page (i).

Election of Members of Council for the Session 1950-51

As previously announced, the new Bye-laws governing the constitution of the Council and its mode of election will affect nominations for the election of Council for the session 1950–51. Nominations for the election of Ordinary Members of Council [*see* Bye-law 27 (iv)] and Overseas Members of Council [*see* Bye-law 28 (iii)] should be sent to the Secretary of the Institution to reach him not later than 1st November, 1949.

Visit of Mining Students to the Gold Coast

For the first time, twelve mining students—six from the Royal School of Mines and six from the School of Metalliferous Mining (Camborne)—are visiting mines on the Gold Coast this month as part of their normal training. They will travel by air and return at the beginning of October. Arrangements for the visit to West Africa have been made by the Gold Coast Chamber of Mines.

June General Meeting

The subject of discussion at the Eighth Ordinary General Meeting of the Institution, held at Burlington House on 16th June, was the practical postgraduate training of mining engineers. Professor J. A. S. Ritson, *Member*, Head of the Department of Mining at the Royal School of Mines, opened the discussion before an audience of about 90 members and visitors, and an interesting debate followed. A report of the discussion will be published in the August issue of the *Bulletin*.

Refining of Non-Ferrous Metals

As this issue of the *Bulletin* goes to press the Institution's Symposium on the Refining of Non-Ferrous Metals will be taking place. Well over 300 tickets of admission have been issued, and it is hoped that useful discussion will result from the meetings. A report of Proceedings of the Symposium will be published, with the 19 papers submitted, as a separate volume, and one copy will be available free of charge to each member provided it is ordered *by 31st July*.

Lecture by Dr. C. H. Desch

The Second Sir Julius Wernher Memorial Lecture of the Institution was delivered by Dr. Desch on 6 July, 1949, at The Royal Institution. The lecture, entitled 'The Effect of Impurities on the Properties of Metals', will be published in the August *Bulletin*.

Meetings in the Session 1949-1950

General Meetings of the Institution during the Fifty-Ninth Session will be held on the third Thursday in each month from October, 1949, to June, 1950. The dates of the Meetings are as follows:

20th October, 1949
17th November, 1949
15th December, 1949
19th January, 1950
16th February, 1950
16th March, 1950
20th April, 1950
18th May, 1950
15th June, 1950

Members from Abroad

The Council are always anxious to meet members who come to England after a long absence abroad, and ask such members to make themselves known to the Secretary when attending General Meetings of the Institution at Burlington House.

Candidates for Admission

The Council welcome communications to assist them in deciding whether the qualifications of candidates for admission into the Institution fulfil the requirements of the Bye-laws. The application forms of candidates (other than those for Studentship) will be open for inspection at the office of the Institution for a period of at least two months from the date of the* Bulletin *in which their applications are announced.

The following have applied for transfer since 9th June, 1949 :

To Membership—

James Herbert Bennetts (*Camborne, Cornwall*).
Richard Bradford McConnell (*Kaduna Junction, Northern Nigeria*).

To Associate Membership—

Robert Laurence Carlton (*Bawdwin, Burma*).
Fadil Kheiry Kabbani (*Jeddah, Saudi Arabia*).
Reginald Clive Pargeter (*Bourne End, Buckinghamshire*).
William Peter Holford Parkinson (*Maryport, Cumberland*).
Ian Graham Pert (*Ghatsila, India*).
John Eric Rockingham (*Kaduna Junction, Northern Nigeria*).

The following have applied for admission since 9th June, 1949 :

To Membership—

Robert Lepsoe (*Trondheim, Norway*).
Harley Uncless Suart (*Broxbourne, Hertfordshire*).

To Associate Membership—

Dowl Foord (*Jos, Northern Nigeria*).
D. Thyagaraja Iyer (*Champion Reefs, India*).
Alexander Munro (*Morro Velho, Nova Lima, Brazil*).

To Affiliateship—

David Faunch (*Wembley, Middlesex*).
Arthur Griffith (*London*).
Harry Albinson Mellor (*Orpington, Kent*).

To Studentship—

Peter John Hedley Rich (*London*).
John Henry Warnock (*Kidderminster, Worcestershire*).

Transfers and Elections

The following were transferred (subject to confirmation in accordance with the Bye-laws) on 9th June, 1949 :

To Membership—

Peter Best (*Champion Reefs, S. India*).
Gordon Brian Mackenzie (*Freetown, Sierra Leone, West Africa*).
Gerald Augustine Patrick Moorhead (*Georgetown, British Guiana*).

To Associate Membership—

George Anthony Jess (*Plymouth, Devon*).
Henry Joseph Martin (*Selukwe, Southern Rhodesia*).
Arthur Theodore Max Mehliss (*Dunnottar, Transvaal*).
William John Palk (*Hangha, Sierra Leone, West Africa*).
Leo Terry (*Filabusi, Southern Rhodesia*) (reinstatement).

The following were elected (subject to confirmation in accordance with the conditions of the Bye-laws) on 9th June, 1949 :

To Membership—

Robert Maurits Peterson (*London*).

To Associate Membership—

Geoffrey Joynt Brittingham (*Port Kembla, N.S.W., Australia*).
Trevor Jestyn Davies (*Stourport-on-Severn, Worcestershire*).
Arthur Fannin Evans (*Port Kembla, N.S.W., Australia*).
Eric Allix Foote (*Mount Isa, Queensland, Australia*).
William Gibson (*South Shields, Co. Durham*).
Joseph Pindor (*Geita, Tanganyika Territory*).

To Studentship—

John William Davies (*Barakin Ladi, Northern Nigeria*).
David John Ivor Evans (*Warlingham, Surrey*).
Michael Andrew Grigg (*Thames Ditton, Surrey*).
John Trevor Hall (*London*).
Giles Freathey Oats (*Hove, Sussex*).
Alan Robert Dundas Orr (*Glasgow*).
Kenneth Bernard Platt (*Camborne, Cornwall*).
Miles Holme Russell (*London*).
Roy Russell (*Hillingdon Heath, Middlesex*).

News of Members

Members, Associate Members and Students are invited to supply the Secretary with personal notes for publication under this heading.

Erratum : A note was published under this heading in the May, *Bulletin*, describing Mr. Leigh W. Bladon as an *Associate Member*. His status is that of *Member*.

Mr. H. Arnall, *Member*, has left England on his return to Nigeria.

Mr. J. G. Berry, *Associate Member*, has returned to India from England.

Mr. A. W. Boustred, *Associate Member*, is returning to England on leave this month, and will return to West Africa later in the year to Taquah and Abosso Mines, Ltd., to which company he has been transferred from Amalgamated Banket Areas, Ltd.

Mr. W. P. Boxall, *Associate Member*, has been appointed Manager of Welgedacht Exploration Co., Ltd., Transvaal.

Mr. Leslie Bristowe, *Member*, will be coming to England on holiday from S.W. Africa at the end of August.

Mr. H. F. Burton, *Student*, has arrived in the Gold Coast from England.

Mr. W. F. Castle, *Associate Member*, has sold his interests in Southern Rhodesia, and is now chairman and managing director of Wildon Estates, in Northern Rhodesia and Tanganyika.

Mr. J. E. Clay, *Associate Member*, has been transferred to the staff of Sub Nigel, Ltd.

Mr. K. E. Daniel, *Student*, has left England for Northern Rhodesia.

Mr. E. L. Day, *Associate Member*, is visiting Canada and the United States.

Mr. J. G. Dennis, *Student*, is returning to England from the Gold Coast on leave.

Dr. J. R. Dinsdale, *Associate Member*, expects to arrive in England this month on six months' leave from India.

Dr. F. Dixey, *O.B.E.*, *Member*, geological adviser to the Secretary of State for the Colonies and Director of Colonial Geological Surveys, was awarded the C.M.G. in the Birthday Honours.

Mr. E. M. El Alfy, *Member*, has been awarded the Order of Industry and Commerce by the King of Egypt.

Mr. K. B. Goode, *Member*, has joined the staff of Mount Isa Mines, Ltd., Queensland.

Mr. O. L. Gray, *Associate Member*, has arrived in England from Nigeria.

Mr. E. P. Hargraves, *Member*, expects to arrive in England from Australia this month.

Dr. G. V. Hobson, *Member*, has resigned his post as Assistant Director (Development) of the Directorate of Opencast Coal Production, to resume private practice as consulting mining geologist.

Mr. H. L. Holloway, *Associate Member*, has returned to Nigeria.

Professor B. W. Holman, *Member*, has been awarded the Order of Maaref by the King of Egypt.

Mr. D. Hosking, *Student*, has arrived in England on leave from the Gold Coast.

Mr. L. S. Jones, *Associate Member*, has joined the staff of the Mines Department, Queensland.

Mr. H. M. Kay, *Associate Member*, has returned to Colombia, South America, after a short visit to the U.S.A. and Great Britain.

Mr. P. F. F. Lancaster-Jones, *Associate Member*, has joined the staff of Western Reefs, Ltd.

Mr. W. D. Leslie, *Associate Member*, has left Salisbury on transfer to Gwanda, Southern Rhodesia.

Mr. D. H. McCall, *Associate Member*, has arrived in Scotland on leave from the Kolar Gold Field.

Mr. G. A. R. McIlhatton, *Student*, is returning to England on leave from Tanganyika.

Mr. F. I. Maritz, *Associate Member*, has left the service of the Southern Rhodesia Government and has joined Shamva Tributors, Ltd.

Mr. J. D. Mead, *Member*, was awarded the C.B.E. in the Birthday Honours for services to Malaya.

Mr. E. P. Meaton, *Associate Member*, has left Randfontein Estates, Ltd., and is now in Johannesburg.

Mr. R. G. K. Morrison, *Member*, superintendent of the Nundydroog mine, Kolar Gold Field, has been appointed to the MacDonald Chair of Mining at McGill University, in succession to Mr. Leigh W. Bladon, *Member*, who is resuming his consulting practice.

Mr. J. J. Pascoe, *Student*, has arrived in India.

Mr. JOHN PENHALE, *Associate Member*, has been transferred from the staff of Sierra Leone Selection Trust to that of Consolidated African Selection Trust, Ltd.

Mr. P. A. PURVIS, *Member*, is now in England on leave until September.

Mr. H. H. ROBOTTOM, *Associate Member*, has returned to Burma after leave in England.

Mr. J. E. ROBSON, *Associate Member*, is now on leave in England from the Gold Coast.

Mr. J. A. ROYCE-EVANS, *Associate Member*, is on his way to England on leave from the Gold Coast.

Dr. G. A. SCHNELLMANN, *Associate Member*, has been appointed consulting geologist to Halkyn District United Mines, Ltd., while retaining an association with the Millom and Askam Hematite Iron Co., Ltd.

Mr. B. G. SKELTON, *Associate Member*, until recently underground manager at Vogelstruisbult, has joined the staff of the New Consolidated Gold Fields, Ltd., in Johannesburg.

Mr. G. GAMLEN THOMAS, *Member*, has been awarded the C.B.E.

Mr. J. W. C. TREEBY, *Associate Member*, is shortly closing his consulting practice in Malaya and will resume in the Caribbean.

Mr. E. H. TREGONING, *Associate Member*, has left England on his return to Burma.

Mr. W. J. TRYTHALL, *Associate Member*, is leaving England shortly for the Gold Coast.

Mr. C. W. WALKER, *Associate Member*, has left England on his return to Sierra Leone.

Mr. T. H. WINSOR, *Associate Member*, has left Southern Rhodesia to accept a post on the Gold Coast.

Mr. S. R. WORTHY, *Student*, will be leaving England shortly to take up a position with Uruwira Minerals, Ltd., Tanganyika.

Mr. J. R. WRIGHT, *Associate Member*, has been transferred from New State Areas to Randfontein Estates Gold Mining Co. (Witwatersrand), Ltd.

Addresses Wanted

A. Armstrong.	R. B. Hicks.
D. S. Broadhurst	G. C. Morgan.
J. A. Cocking.	A. I. Scott.
E. Dickson.	A. Sloss.

The Council regret to announce the decease of GEORGE CARTER, *Member*, on 2nd July, 1948; ARTHUR DELMAR COMBE, *Associate Member*, on 23rd May, 1949; NATHANIEL MALCOLM, *Member*, in 1948; R. ARTHUR THOMAS, *Member*, on 30th May, 1949; AMOS TRELOAR, *Member*, on 25th May, 1949; and HENRY MORLEY WHITE, *Member*, on 1st July, 1949.

ADDITIONS TO JOINT LIBRARY OF THE INSTITUTION AND THE INSTITUTION OF MINING ENGINEERS

*Books (excluding works marked *) may be borrowed by members personally or by post from the Librarian, 424, Salisbury House, London, E.C. 2.*

Books and Pamphlets:

BRITISH STANDARDS INSTITUTION. *Yearbook 1949.* London: The Institution, 1949. 362 p. 5s.

NORTHERN INDUSTRIAL GROUP, RESEARCH STAFF. *North-east coast; a survey of industrial facilities, comprising Northumberland, Durham and the North Riding of Yorkshire.* (Describes the metallic and non-metallic minerals of this area.) Newcastle-upon-Tyne: Andrew Reid, 1949. 146 p., illus., maps, diagrs., tabs., biblios. 45s.

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331 Labour

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ITUTION OF MINING AND METALLURGY

PRESIDENTIAL ADDRESS

Role of The Institution in Present-day Educational Developments

By W. A. C. NEWMAN, O.B.E.

r the 57 years of its existence the Institution has main avenues of activity:

establishment and maintenance of the dignity and of the three professions with which it is concerned—us mining, metallurgy, and mining geology—and the of professional status by enforcing high technical ons for admission to its membership and by regulating sional conduct of its members. It is, and we trust it nue to be, the foremost qualifying body in these branches d science, the representative of authoritative professional and the medium for approach on all relevant matters to nent and other public corporations.

ie advancement of knowledge in the three professions by lication of papers and by discussions comprising valuable tions to technological literature which circulate in every the world.

The furtherance of education in relation to the needs of the ions and their members with which has been coupled in the he award of postgraduate scholarships and assistance in the ishment and encouragement of technological institutions as the Imperial College of Science and Technology and the l of Metalliferous Mining at Camborne.

milestone in connection with the first of these three objects reached and passed when the new Bye-laws, which were ered desirable by changing conditions in the professions, were ntly approved by the Privy Council. It is anticipated by the uncil and by the members of the Institution generally that the ised provisions for entry to Associate Membership by examina- n will materially assist in making the Institution, which is eady strong and influential, a greater and more comprehensive ofessional body, comparable with similar bodies serving allied ofessions. They are hopeful, too, that into the new class of filiates will be drawn men, highly placed in their own spheres, o have a direct interest in our activities but who cannot qualify r corporate membership in accordance with the terms of the arter; and that by co-operation and goodwill, by team work d understanding and by the convergence of effort, greater ogress and efficiency may be achieved.

It is to the third of the objectives which have been mentioned, wever, that I may perhaps direct your attention by passing in review certain aspects of educational activity, especially as

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THE INSTITUTION OF MINING AND METALLURGY

PRESIDENTIAL ADDRESS

The Role of The Institution in Present-day Educational Developments

By W. A. C. Newman, O.B.E.

Throughout the 57 years of its existence the Institution has had three main avenues of activity:

(1) The establishment and maintenance of the dignity and usefulness of the three professions with which it is concerned—metalliferous mining, metallurgy, and mining geology—and the upholding of professional status by enforcing high technical qualifications for admission to its membership and by regulating the professional conduct of its members. It is, and we trust it will continue to be, the foremost qualifying body in these branches of applied science, the representative of authoritative professional opinion and the medium for approach on all relevant matters to Government and other public corporations.

(2) The advancement of knowledge in the three professions by the publication of papers and by discussions comprising valuable contributions to technological literature which circulate in every part of the world.

(3) The furtherance of education in relation to the needs of the professions and their members with which has been coupled in the past the award of postgraduate scholarships and assistance in the establishment and encouragement of technological institutions such as the Imperial College of Science and Technology and the School of Metalliferous Mining at Camborne.

A milestone in connection with the first of these three objects was reached and passed when the new Bye-laws, which were rendered desirable by changing conditions in the professions, were recently approved by the Privy Council. It is anticipated by the Council and by the members of the Institution generally that the revised provisions for entry to Associate Membership by examination will materially assist in making the Institution, which is already strong and influential, a greater and more comprehensive professional body, comparable with similar bodies serving allied professions. They are hopeful, too, that into the new class of Affiliates will be drawn men, highly placed in their own spheres, who have a direct interest in our activities but who cannot qualify for corporate membership in accordance with the terms of the Charter; and that by co-operation and goodwill, by team work and understanding and by the convergence of effort, greater progress and efficiency may be achieved.

It is to the third of the objectives which have been mentioned, however, that I may perhaps direct your attention by passing in brief review certain aspects of educational activity, especially as they affect the Institution and bear upon its role as an educative influence in the training of recruits to the three professions.

The Increasing Interest in Technical Education

At the present time, considerable attention is being given to educational matters, in all the principal countries of the world, and the Council is actively interested in such of these as impinge on the affairs of the Institution in their many phases, particularly in this country and the Empire. In the Dominions educational activity is rapidly expanding. The training of scientific and technical personnel for the mining and metallurgical industry, for instance, is the subject of a recent report by the Council of the Australasian Institute of Mining and Metallurgy, in which reference is made to the formation of the N.S.W. Institute of Technology and to the possible inauguration of a complete mining course attached to the University of Queensland.

The essential need for all young people to receive the type of education and training best suited to their interests and their natural aptitudes is becoming more widely believed and acted upon. In the last few years there has been an increasing insistence on the reorientation of our educational system from school age up to graduation and there is now a stronger public realization than ever of the importance of the Universities and the contribution they can make to the national good.

The stimulus which has prompted this increased interest in educational matters is two-fold.

(1) During the recent wars the scientists and the technical men held key posts and were recognized as contributing largely to the ultimate victories. The frenzy of war and the clash of arms impelled research, discovery, invention and application with far-reaching effects, and the cry then went up—probably even with more emphasis in 1945 than in 1918—that in peace as well as in war the trained scientific mind must be given ample opportunities to contribute to the country's progress. This urge towards a scientific career has been intensified by the greater facilities now available for reaching the highest educational levels.

(2) A further stimulus towards enhanced interest in these matters has been the serious shortage in scientific man-power available in Government Departments, in industry and in the teaching profession, which has been revealed by several Government Committees. Reference to these will be made later.

As a guardian of the interests of metalliferous mining, mining geology and extraction metallurgy, the Institution shares in the growing concern for recruits to applied science and for the further development of educational facilities. It has a particular part to play and becomes naturally, and rightly, interested not only in the professional status of its present members but also in the educational upbringing of those who will be associated with it in the future, in the type of youth who can be attracted into its ranks and the range of training he gets at all stages to fit him for his future career.

In pursuing this interest, however, four important points should be borne in mind—

(*a*) The professions which we serve are essentially applied sciences of which little is normally heard in the schools.

(*b*) They have not the same prominence in this country as other branches of technology, since the mineral resources, apart from coal, are comparatively small.

(*c*) The young student is still in the formative stage. He presents a problem in adaptation to a complex social and industrial system in which education plays the first important part.

(*d*) Developments in any particular branch of technical education, and especially in the ones in which we are directly interested, cannot be assessed properly unless attention is paid at the same time to a background of the whole educational system of the country of which they form a part, and also to their relationships with those other branches of technology which are becoming increasingly important to the professions we represent. Indeed, because of the very nature of our particular applied sciences, a conception of the whole structure, including the pure sciences on which they are based, is necessary before we can determine their relevance. It is quite as important that we should know something of school curricula and their use as it is to know the extent of University courses, for on the efficiency of the general grounding in the earlier stages will depend the success achieved at the later.

Instruction and training in metalliferous mining, extraction metallurgy and mining geology in Britain and also overseas is confined to a relatively small group of centres—Universities, schools of University standing and certain technical colleges. The problems of recruitment to these centres, and of issue from them, are no less acute or less fundamental because of this limitation of numbers. To ensure that the professions will be best served, the best men must be sought early and encouraged to enter an educational sequence, of which, in their own early stages, they may not be aware, but which offers good prospects and many advantages. It is more than ever desirable under the conditions of to-day that the traditions of mining and metallurgy, long established in this country and now increasingly so in the Dominions, shall be maintained and that the best possible types of men in sufficient numbers shall be ready to go, properly trained, into those lands where British names and British capital have meant, and still mean, so much.

All this demands a knowledge by those in the schools of the opportunities that are available. It calls for a preliminary grounding in the fundamental subjects which in some instances might conveniently quote industrial applications as examples. Thus it will be readily realized that if chemistry masters could be induced to give simple metallurgical illustrations of applied chemistry, the interest in metallurgy itself would receive a great impetus. Following the initial grounding there should be a steady insistence at a high level on the provision of the fullest facilities for graduate training in the subjects which concern the Institution. Instruction of this character cannot of course turn out a finished product, a fully competent engineer. Rather must the attempt be made to create a thinking, constructive and imaginative mind, equipped with fundamental knowledge which experience and contact with his fellows will mature.

Education in the Schools

Our interest, then, starts in the schools and, so far as metallurgy is concerned, the Joint Committee on Metallurgical Education, on which the Institution is represented, has given considerable attention to informing teachers and scholars what metallurgy is and what it offers as a career.

By means of a brochure, which describes the subject and indicates the prospects in various fields, by films and by lectures, the Committee has endeavoured to stimulate enquiry in the schools regarding metallurgy and thus to broaden the field of recruitment.

Into these considerations, however, the repercussions of the 1944 Education Act have been projected. During the last ten or fifteen years, objections have been raised to the ordinary School or leaving Certificate and to the Higher School Certificate examinations—the near equivalents of the older Matriculation and Intermediate Degree examinations respectively. The Norwood Committee in 1942 and the Secondary Schools Examinations Council in 1947, both set up by the Governments of their time, reported in favour of a change. Their conclusions were reinforced by a Government Circular (No. 168) issued on the same subject. Both bodies felt that there was need for an examination less rigid in its syllabuses and in the number and choice of papers to be taken. They also recommended, among other things, that there should be fewer—or perhaps no—compulsory papers. The Examinations Council put forward a scheme for a ' General Certificate of Education ' coupled with the formulation of a School record card containing a running commentary on a student's progress and potentialities. In effect these recommendations have been accepted and after a transition period lasting until 1951 the scholar will be able, if over 16, to sit for as many or as few subjects as he desires in Ordinary, Advanced or Scholarship grades of the new examination. His Certificate will be endorsed with the subjects in which he passes and subsequently with those which he may take at a later date. The papers will be set by his own teachers and they may, possibly, be assessed by some accepted local body. Unfortunately, this may result in a lack of uniformity in standard, a difficulty which has already arisen in the development of the scheme for National Certificates in metallurgy and other subjects where a similar system of conducting examinations is employed and where judicial marking and assessment have given great concern.

The General Certificate of Education, together with the record card, will be the sole basis of a youth's entry into the University or into industry. He will have been a free agent in selecting the number and grades of the examinations he has attempted and the fear is that, unless well directed, he may not have acquired a balanced education.

Considerable discussion has naturally ranged round these decisions. The Ministry has ostensibly fixed the minimum age of entry at 16, in order to prevent undue specialization at school. It has been urged that basic principles and disciplines should be firmly established so that they may exert their influence fully during a youth's future life, thus ensuring a soundly trained mental outlook not confused by assorted shreds of knowledge and in addition that greater scope should be given to an individual's natural bent to develop his own personality.

The schools in reply say that the proposed regulations will penalize the bright boy who has hitherto sat for the School Certificate at 14 years of age, and will deprive him of a period in which he could be preparing for the University.

In the Secondary Schools the passage from the fifth to the sixth form occurs at a transitional stage in the average youth's life and outlook. His mind and prospects are developing and he begins to realize for the first time the need for some concentration and more intensive study. Some youths, however, may mature early, say at 14, while others are only ready for the next step at 17 and it is difficult to reconcile these wide variations in a general scheme.

The following alternatives, however, have been advanced—

(1) To continue the present school curriculum and allow the youth who is ready to take the examination, at, say, 15, to extend his study of the five or six necessary subjects during a further year. This would mean a waste of time at the most receptive age and might lead to staleness ; it would decrease the period spent in preparation for the University from three to two years.

(2) To extend the time devoted to preliminary, fundamental subjects and thus bring him on more gradually until he is 16. This would have the advantage of giving greater opportunity for instruction in the humanities, concerning the lack of which much has recently been written and said in many walks of life. There is little time for such work in the technological sides of the Universities ; it is rightly the function of the schools in the first instance.

To this view of early education most bodies connected with technology subscribe. Thus the Council of the Australasian Institute say in the report already referred to : ' The youth should be "educated" as well as "trained". Education in cultural subjects should proceed far enough to develop a broad outlook on social, scientific and economic questions. Education in pure science should aim at the development of exact and systematic habits of thought as well as at knowledge of the subjects. Technical training should be concerned with the practical application of general principles rather than with the minutiae of technique.'

Up to the present, the Universities have said that the new examination will not ensure a sufficiently broad initial education or an adequate preparation for their own courses, and they cannot accept it. Possibly this reflects changing views and even grave doubts about the proper level of University entrance which are now becoming discernible. There has been, for instance, a distinct tendency to relegate work for the Intermediate—or Higher Schools Certificate—examination to the schools, thus leaving the first year in the University free for graduate work. This, in effect, gives another year to study for a degree, but at the same time imposes a large measure of specialization in the later years of school life.

A suggested compromise is to postpone the beginning of sixth-form work for a year, which would result in a higher average age of entry into, and exit from, the University. Alternatively, the schools and the Universities may agree on a modified range of subjects and standards for entrance purposes.

The use of individual records, which is part of the new examination proposals, would be of great benefit if one could be certain that (i) they were kept regularly and fairly, and (ii) they were related to specific standards, so that proper comparisons could be made of records compiled in different parts of the country. Neither of these can be guaranteed, however, and the records may break down because of the diversities in human nature. The Committee of Vice Chancellors and Principals of Universities foresee these same difficulties and hesitate to accept such records which may not have a common denominator.

While we must necessarily be interested in educational trends in the schools, it is to the next stage that our attention should be principally directed, namely, to consideration of the Universities and technological and technical colleges, of full and part-time study, of vocational courses and of postgraduate work. Here again, it is useful to have a general picture of the proposals in their bearing on our own interests. It is also necessary to remember that the efficiency or otherwise of University teaching quickly becomes known in the schools and has a direct influence on the attraction of mining or metallurgy—or indeed any other subject—for the youths passing through them.

Shortage of Scientific Man-power

Some comments have already been made on a revival in interest in educational matters and in scientific careers, which is occurring almost simultaneously with the urgent reminder by the Barlow and other Committees of a serious shortage in scientific man-power.

The Barlow Committee stated in 1946 that at that time, the maximum number of qualified scientists in all branches in the country probably did not exceed 55,000. It was estimated that at least 70,000—of whom at least 30,000 would be teachers in Universities and secondary schools—would be required by 1950. Beyond this an increase of some 40 per cent in the student population of the Universities was predicted, which, with the raising of the school leaving age to 16 and the establishment of County Colleges would involve the appointment of a further 15,000 teachers. It was estimated that by 1955 the overall demand for scientists would have increased to 95,000 and the maximum supply only to some 64,000, a discrepancy which led the Committee to recommend the doubling of the outturn of graduates from the Universities, i.e. 5,000 instead of 2,500 per annum, and the provision of buildings and finance accordingly. Even with these largely increased numbers the proportion passing to mining and metallurgy is not excessive—indeed, in some directions it is insufficient.

The Universities have responded with commendable co-operation and the University Grants Committee last year reported that the numerical expansion scheduled to take ten years has been almost carried out in three; that, in fact, nearly 80,000 of the 90,000 scientists have passed into, or are through, the Universities. The Committee feel too that there is a large body of innate intelligence in the country which will more than fill the present University capacity and provide the scientific quotas required, especially if financial aid is extended in those instances where a bright youth is frustrated by lack of funds. On the other hand there is a fear that with the very largely increased numbers of University entrants the standards of entry are being lowered. A working party which reported to the Minister of Education last year suggested that local authority awards which are given to a large number of University entrants should be made only with the recommendation of a University, on the assumption that examinations alone are not a sufficient basis under present conditions for selecting students. This would, at any rate, enable the Universities to choose their entrants.

The prospect now is that for some years to come the proportion of students of science and technology will be higher than it was before the war, although the distribution in the various faculties may vary. For instance, it is found that in Mechanical, Civil and Electrical Engineering, Physics and Geology the demand will probably not exceed the outturn, whereas the reverse will be the case with Chemistry and Metallurgy. Mining, either of coal or of metals, has not so far come within the purview of these Committees —a fact which is to be deplored. A number of Committees under the direction of Lord Hankey are due to report shortly on the general demand for scientists and technologists in various branches of activity and they will no doubt indicate how any differences in demand can be met.

Grade of Work to be Done at the Universities

Turning now to the Universities which make provision for technological studies, we are confronted at the outset with their changing views on entrance requirements, to which some reference has already been made. A report by the Nuffield College suggests, in addition to the relegation of work of intermediate standard to the schools, that courses in applied sciences should be diverted from the Universities to the technological colleges, leaving the former only with fundamental sciences and research : and above all this there is a threat, in one or two instances, that eventually a University course will not be considered complete if it is of less than five years' duration. The Nuffield College report, in considering these possibilities, feels it to be of primary importance that the character of the Universities shall be preserved and that men issuing from them must be trained in fundamentals without altering the character and the purpose of the University. In relation to this and to the proposed shedding of intermediate work at one end and of applied science courses at the other, it must be pointed out that one of the first effects would be that, so far as technological students are concerned, the humanitarian and liberalizing influence of the University would be constricted and might even be sacrificed. This condition of affairs would be extremely unfortunate since it is very desirable that such students should acquire a broader understanding which personal contacts with colleagues engaged in other courses would do much to stimulate. In this connection Lord Elton has written earlier this week : ' A University should be a universe in miniature, housing all sorts and conditions . . . all of whom, by a process of mental cross fertilization, are likely to benefit from dissimilar contemporaries '.

The Technical Colleges

Among the great problems engaging serious attention at the present time, and one of great importance to the Institution, is the place of technology and of the applied sciences in the education structure. Should they continue to be taught at the Universities in regular courses or in postgraduate work ? What are the proper places and functions of the technical colleges ? Are these colleges sufficiently developed for our national needs and in what relationship should they stand to the Universities, to local authorities and to industry ?

The Percy Committee, appointed by the Government to report on the problems of higher technological education, have been supported in their recommendations by the Parliamentary and Scientific Committee. The Council of the Institution has submitted a memorandum to the latter pointing out its views so far as mining and metallurgy are concerned, and the Joint Committee on Metallurgical Education is preparing memoranda on the same question.

The Percy Committee foresee that the future role of the technical college must be an increasingly important one since emphasis is now being laid to a greater degree on technology and technological research. To meet this it is urged that the status of the technical colleges should be raised, and that to some of the major ones either degree or diploma granting powers should be given, or that they should be linked to neighbouring Universities. In any case, there should be closer contact and interchange of students between all classes of technical colleges and the Universities, and technological courses together with postgraduate work of a high standard should be developed within both. Properly controlled, such an organization would be beneficial and would prevent the narrowness of outlook which arises from isolation ; any unrestrained granting of degrees, however, such as is the case in some instances abroad, would lower their value and their prestige and might very well degrade the value of present University degrees themselves. Some uniformity of standard in graduation should be maintained as far as possible.

To effect these changes it is proposed to set up a National Council of Technology to advise the Minister and the University Grants Committee on the national aspects of regional policies, and later this Council would possibly have an executive function in granting certificates following technological courses of graduate standard. The Council is to consist of representatives of Regional Advisory Councils, of Regional Academic Boards and of Industry. There are, in all, ten regions, in each of which an Advisory Council will co-ordinate technological studies in the Universities and the technical colleges. Complementary to this Advisory Council there will be an Academic Board, formed of representatives of the teaching staffs, to consider the development of studies and related matters and to advise the governing bodies of the various institutions

From the discussions which have ranged round these various reports, three points may be selected for special mention.

(1) It has been suggested that in those cases where, for varying reasons, applied sciences draw only a limited number of students they should be accommodated in special colleges or universities, where they can be developed to the fullest extent, and to which the students concerned should gravitate. In this way, plant, staff and buildings would be conserved. Within this framework the subjects in which we are interested would naturally fall, and, as an Institution and a qualifying body, it is desirable that we should have the opportunity of putting forward opinions when major relevant matters are under consideration.

As a particular instance of this, reference may be made to Mineral Dressing on the status of which the Institution has recently expressed its views. In the field of Mining and Metallurgy, the ranking of this subject is very high because of the increasing attention which must be given to its application in the profitable working of low-grade and complex ores. In the teaching schools, however, its relative prominence has not hitherto been established. The

Council are of opinion that this deficiency should be remedied and consider that instruction in Mineral Dressing should be professorially directed in a special department or departments and should lead to a degree in the faculty in which it would be appropriately placed.

(2) It is being repeatedly stated that we lag behind other countries—some larger, some smaller—in the provision of well-equipped technological institutes. Attention has been drawn to the provisions made by the Massachussetts Institute of Technology and the California Institute of Technology in the United States and by the Delft Institute in Holland. Within these organizations the various applied sciences are dissected into their several parts and each part placed in charge of a qualified teacher, generally of professorial rank. Students are trained in the fundamentals and their applications to the demands of industry.

The view has been taken by many of the various Committees and also by individuals that technological education in this country needs the provision of at least one large comprehensive institute. The Imperial College, although admirably conceived and administered, does not appear fully to meet this need and it may be that the solution could be found by creating around it as the nucleus, the larger, grander conception that would compare with what some of our rivals abroad already have.

(3) There is an increasing need for technical knowledge within the management of our great industries both here and abroad and there is an even greater need for some instruction in the principles of scientific management at some stage in a University or technical college course. Possibly, however, the time available for such courses is already too short and the insertion of subjects of this character may have to be delayed to a postgraduate period. But an appreciation at some stage of scientific advances in management is increasingly called for by industry where new and more complex fields are constantly being explored.

It is the task of management to balance research and production effort without robbing essential administration of too many active and fertile minds. The Advisory Council has declared recently that the scientific man should be given the status which will enable him to take a part in the determination of policy where scientific factors are involved. Scientific attainments should lead to advancement to positions of the highest administrative responsibility when the requisite administrative talent becomes apparent. It has been wisely remarked that it does not follow that the transfer of a scientific man to administration is necessarily the loss of a scientific man to the organization.

Part-Time Students

We should not leave consideration of technical colleges without some word concerning the part-time students who for many years have formed the major part of their membership and whose numbers, including many with leanings towards Metallurgy, are increasing greatly under the provision for the daytime release of youths from industry in order to pursue their studies. In passing, it may be noted that the numbers enjoying part-time day release have increased from 43,000 in 1939 to over 200,000 in 1948.

A recent ‘ working party ’ set up by the Ministry of Education has recommended that a certain proportion of part-time students should be selected for entry to the Universities and, as the facilities for this increase, it is probable that such a scheme will expand. There will still be many, however, who, by force of circumstances will find it necessary to work during the day and study in the evening. The student's aims in such studies are of course a greater understanding of his job and the hope of bettering himself. One can but admire the tenacity with which some of these youths hold on to their opportunities and indeed how some—a good many in fact—have achieved great success. They represent what is perhaps the most desirable type of recruit, namely, those who by their own energy, initiative and ambition, make rapid progress.

It would seem that greater encouragement might be given both to the schools and to the colleges that undertake a full programme of this class of work and also to their worthier students. Such recognition would provide an incentive, reinforce the tuition and stimulate the outturn of the best type of product.

I. J. Pitman, in a letter to *The Times* earlier this year, had these youths in mind when he wrote : ‘ The great glory of a continuation education has been in the millions who, having an interest in applied rather than pure knowledge, have enjoyed a new educational deal and have concentrated and worked, rather than indifferently “ done time ”, and so have received an education (quite apart from any use it may have in their vocation) through studying a useful subject ’.

Part-time students come under review, so far as Metallurgy is concerned, by the Joint Committee on Education. As a rule they have not had an opportunity to matriculate and examinations for which they can sit, and thus attain some qualification, have, for many years past, been arranged by the City and Guilds of London Institute and more recently by the Joint Committee for National Certificates in Metallurgy and the Institution of Metallurgists. Representatives of the Institution of Mining and Metallurgy take part in the deliberations of these bodies. We have thereby a direct interest in the people for whom they cater and we make an annual grant to assist the National Certificate scheme.

The examinations of the City and Guilds of London Institute have long been recognized for part-time students and for such full-time students as have cared to take them. They are held on a national and uniform basis and may lead to the granting of a full technological certificate. In these two features they differ from National Certificates, the papers for which are set locally by teachers in the various institutes and are assessed by the controlling bodies. The existence, in the first case, of a common national standard of examination appears to have many advantages.

The National Certificate scheme in Metallurgy, although directed by the Joint Committee of the Technical Institutions, is administered by officials of the Ministry of Education. It has been in operation for only a few years and has been successful within its limitations. A large number of colleges and institutes are now arranging appropriate courses and soon it may be possible to add a Diploma grade, to be awarded after full-time work, to the present Ordinary and Higher Certificates, which are essentially related only to part-time studies.

Postgraduate Work

The Institution's interest in educational activities in regard to Metallurgy, however, does not cease at the Universities or at the technical colleges, for in recent years it has participated in arrangements for assisting postgraduate work and for stimulating vacational experience abroad. Four such schemes, financed by generous donors, are now in operation and the professions and the Institution are indeed grateful for the opportunities which have been provided to create a wider horizon for those to whom the awards have been made.

The Trustees of the Nuffield Foundation, who administer funds provided by the munificence of Lord Nuffield, sought the advice of the Institution on the possibility of using a sum of some £70,000 spread over five years in the interest of extraction metallurgy on the further development of which within the Empire Lord Nuffield himself had become deeply concerned. It was agreed that about half the sum should be devoted to the provision of a maximum of five Fellowships, five Postgraduate Scholarships and ten Vacation Scholarships, for selected applicants from the Empire, during each of the five years. The remainder of the sum is to be devoted to research and to cover the cost of the necessary apparatus and the salaries of a Fellow and a team of assistants.

In three years of operation eleven Fellows, mainly from University staffs, fifteen Postgraduate Scholars and twenty-eight Vacation Scholars have been appointed, and it is gratifying to everyone concerned that the awards have been fruitful in providing the recipients with experience overseas, in generally broadening their knowledge, and in promoting useful contacts.

The present suggestion of the Trustees is that at the end of the period during which the awards are to be granted, if the principle of the scheme has been found to be justified, consideration should be given to its continuance under other auspices, either Governmental, institutional or industrial, or partly all three. There can be no doubt of the value and usefulness of the arrangements as they have developed. The general tendency to increase the opportunities for vacational experience at all colleges and universities may reduce, but not eliminate, the necessity for the vacation scholarships, but there is a clear need to provide professors, heads of departments and their staffs with means of refreshing their knowledge of modern developments, and to enable graduates of a few years' standing to find out what their colleagues abroad are doing, thereby achieving a clearer and more correct perspective. These needs will be continuing ; they should be equally urgent in mining as in metallurgy and in the future policy of the Institution the requirements of mining must surely assume an equally prominent place.

Appointment to the Nuffield Research Fellowship has yet to be made. It may be that if success attends this venture into sponsored research in Extraction Metallurgy—and there appears to be a great need for such development work in both mining and metallurgy—a nucleus will have been established of something akin to the Research Associations now in being which are subsidized partly by industry and partly by the Government. It may be that there is scope in both Mining and Metallurgy for bodies of this character to which problems of a fundamental and a practical nature could be brought for investigation with benefit to industry and to the professions as a whole.

The Institution is also represented on a Committee to administer funds, amounting to £50,000, which have been provided by the Mond Nickel Co. Other members of this Committee have been nominated by the Institute of Metals, the Iron and Steel Institute, the Institute of British Foundrymen and the Institution of Metallurgists. These funds are intended to enable those selected during the next fifteen years to spend periods of up to one year either in this country or abroad in studying the practical applications of metallurgical research and the principles of technical control and administration. Attention has already been drawn to the increasing need for scientific management and for the training of technologists in such executive work. It is confidently hoped that the Mond Fellowships will assist in meeting this need. Five Fellowships have already been awarded.

A further benefaction to which generous recognition must be given is that of Messrs. Capper Pass & Son, who have placed £100 per annum at the disposal of the Institution for seven years, and a further £100 per annum at the disposal of the Institute of Metals for a similar period, to encourage the writing of papers dealing with plant and processes used in extraction metallurgy and in the fabrication industries respectively.

Research

In recent years stress has been increasingly laid on research. Both problems and equipment have become more complex. The venue has extended from the Universities into the laboratories of industrial firms, of research associations and of the larger Government departments. As this expansion has unfolded it has become evident that knowledge is increasing at a greater rate than its application, that the man-power necessary for this purpose must be sought specially and that it must rank as high as that required for pure research.

This has been stressed repeatedly. In the First Report of the Advisory Council on Scientific Policy, Sir Henry Tizard said: 'The Council does not disparage research but the first requirement at the present time is the application of existing knowledge and the central problem in this is the scientific man and his proper distribution'. He emphasized this again in his Presidential Address to the British Association last year. An increasing number of trained men with imaginative and inquisitive minds must be directed into industrial channels in order that new processes and new inventions may be applied with the same orderly thought as was used in their creation.

There are many advantages from the points of view of both teachers and staff in bringing University and technical college teachers into closer contact with industry but it is not always desirable policy for practical industrial problems to be pursued in academic or pure research laboratory surroundings, where the consideration of fundamentals is more appropriate. It is, generally speaking, better that in the works there should be the counterparts of the research scientists of the schools, and it may well be that the technological colleges in their new garb, or the super-technological institute of the future, will be the main source of supply of men for these purposes.

Research has become an attractive career mainly because of the encouragement and publicity it has received. As a consequence the rate of scientific advance is not necessarily related directly to the number of people engaged in research. Indeed, men with special gifts in this direction have tended to form a smaller proportion of the whole. But side by side with this has arisen the beneficial team spirit in which workers in many spheres have united in an effort to solve a central problem. It would be unwise, however, to assume that this is necessarily a desirable step towards planned or operational research in peace time as it was known in war. Too much centralization of control deadens initiative and under normal circumstances is to be deprecated. Men work best when the spirit of adventure grips them and they retain their personal freedom.

* * * *

I have passed in very cursory review some of the salient points in the present movements in educational development and in the increasing interest of the Institution towards them. I am conscious that I have touched but the fringe of a very big and complex subject. If, however, the professions which we represent are to make orderly progress, if they are to become filled with men of the right type, if we are to maintain our traditions at the highest possible level, we should make determined efforts to encourage a knowledge of mining and metallurgy in the schools and to stimulate a regard for these professions as satisfying and profitable careers. We should carefully foster the need for a systematic grounding in the humanities as well as in subjects of a more special character so that our future members may not only act, but also speak and write effectively. We should give all possible assistance and encouragement to the teaching of Mining and Metallurgy in the higher ranges and also to the pursuit of research both pure and applied.

The Place of the Professional Institution

Graduation or its equivalent qualification is the spring-board from which a man starts his active work ; but, as the years pass and experience gathers, its first significance is replaced by the need for current contacts, for a knowledge of advances in his profession, for a recognized place among his colleagues, and for acceptance into a body whose dignity and status are recognized in other fields.

I cannot do better than quote, in this connection, from a recent memorandum drawn up by the Council of this Institution for presentation to the Parliamentary and Scientific Committee. ' To a man employed in his profession, membership of an institution such as this is of continuous and increasing importance. After graduation and in the course of professional work it may be said to be of even greater value than a degree or diploma which is gained at the beginning of a career and is not supported until later by the breadth of experience gained in actual practice. The desirable end is that one should be complementary to the other. Membership of a professional institution is the natural step for a graduate who wishes to maintain contact with, and derive benefit from, advances in technological matters, for by it he meets, and comes to know, men of wider interests in his own profession and is able to share in the presentation and discussion of contributions dealing with his own and kindred fields of work. Admission to the leading professional institutions representative of technology, and the attainment of full membership in them, are more likely to achieve national and international recognition as criteria of technological qualifications than the diminishing significance of a degree with the passage of time '.

Some apology should be made, possibly, for the emphasis which has been laid on Metallurgy in this Address. This is due to my own interest in that subject, and also to the fact—of which I, and many others, are very conscious—that it is at present more on the metallurgical than on the mining side that educational standards and landmarks are being violently, but profitably, disturbed. It must be our hope that the balance may be restored, and that the education and training of mining engineers and of mining geologists may in the near future receive the closer scrutiny and the generous assistance and guidance which have been forthcoming for the benefit of the metallurgist.

In thus dwelling at some length on matters coming within the purview of the Institution and which have assumed greater prominence during recent years, I trust I have not strayed too far from the relevance which they may have for its well-being. But I have been prompted by the desire to ensure, first, that those who may have some influence in guiding our future interest shall be aware of the movements both in our own and allied fields which are on foot at the present time for the betterment of our educational facilities in relation to those in other countries, and secondly that they may continue to give consideration to the earlier and vitally important stages of training for our professions.

THE INSTITUTION OF MINING AND METALLURGY

FIFTY-EIGHTH ANNUAL GENERAL MEETING

Held in the Rooms of the Geological Society of London, Burlington House, Piccadilly, London, W.1, on Thursday, 19th May, 1949.

Mr. S. E. Taylor, *President*, in the Chair.

The Minutes of the previous Annual General Meeting, which had already been circulated, were taken as read and signed.

Mr. L. H. Bartlett moved, and **Mr. J. S. Whitworth** seconded, and it was agreed that Messrs. J. B. Simpson, F. Yeates, H. E. Wilson and H. R. Kerr be appointed to act as Scrutineers to examine the balloting papers for the election or re-election of Members of Council.

Benevolent Fund

Mr. S. H. Ford said: The Accounts and Report of the Committee of Management for 1948 were submitted to a meeting held an hour ago, and were adopted.

I wish to thank you, Mr. President, for the opportunity to comment on the Report to this Meeting. Comments made here, unlike those to the meeting we have just held, go to all members and not only to those who happen to be in the room.

The Balance Sheet shows investments on Capital Account of £15,800 odd, as practically the same as the year before. Other investments at £1,480 are unchanged; cash in hand, in the Bank and in the Post Office is down by £50. Income is up by £16; as grants are also up, but by £460, the net result is that expenditure exceeds income by £250.

Mining companies and finance houses gave generous help, as they always do, and the Committee are very grateful for it, as it is a considerable part of the income.

The Roll of the Institution carried the names of 1,777 Members and Associates: 271 contributed last year, and so 1,506 did not. These figures are shown on a chart which goes out with the Report, and we think that members who see the wide open space between the lines will agree that it is not a creditable showing. The Committee are meeting real needs; the Fund is worth taking an interest in, and it is your Fund.

A more satisfactory figure is that income-tax-payer-covenanters made direct contributions last year of £516, and because of the covenants the Committee got back an additional £458—that is over and above the £516—as income tax recovered. Tax recoveries are going up steadily, and we hope that more members will use this very effective way of giving help.

The Report refers to food parcels from overseas. The Australasian Institute sent parcels to this Institution; other donors sent to the Committee, and still others sent to individuals whose names were suggested by the Committee. All the parcels went to the recipients through the agency, in one form or another, of the Committee. None of us here have had to live on an Old Age Pension, and so we do not know what an 8- or 10-lb. food parcel can mean. Some of those who received them do know, and their letters of thanks show it.

During the year, the Committee reviewed the amounts of grants. Some of these amounts were fixed some time ago, when money had a real value, but looking at them now, and at present prices, they appeared to be—and were—pitifully small, and so we put them up.

Also, during the year, three beneficiaries died and there were three new cases, two of which brought children to the Fund.

At the moment there are some 18 children whose education, or maintenance, or a mixture of the two, are being helped by the Committee. The extra cost of more children coming on the Fund, and the extra cost of increased grants, all took effect during the last three or four months of the year, and they are continuing in effect from the beginning of this year, so that we may expect that calls on the Fund will be heavier this year than last; in fact we are already feeling it. A chart of income and expenditure, which goes out with the Report, shows that the line marking the trend of grants is going steadily up; it has crossed and is now above the income line and is still going up.

Further, we are under the shadow of possible appeals arising from war casualties; they will come, but so far it is only the shadow.

The Fund is in a sound position financially, but it would not be a happy state of affairs if the Committee, who had to draw heavily on accumulated resources last year, have to draw a further considerable amount this year. However, the Committee are in the hands of members. What are you going to do about it?'

To sum it all up: the Committee have overspent largely, and are unrepentant, but they expect to receive absolution. It rests with you, gentlemen, to decide whether or not you will give absolution. If you do, there is a practical way of showing it.

Report of Council and Statement of Accounts

The **President** moved that the Annual Report of the Council for the Session 1948–49 and the Statement of Accounts for the year ended 31st December, 1948, be adopted.

Before calling upon the Honorary Treasurer to second the motion he referred to the main activities of the Institution during the Session.

The revision of the Bye-laws had been the outstanding achievement. After long and careful consideration and painstaking drafting the Bye-laws had been revised. The new Bye-laws were adopted at the Special General Meeting held in December, 1948, and came into force in February, 1949, when they were allowed by the Privy Council.

The alterations which had been made to the Bye-laws were not an end in themselves, but a valuable means to an end. They would have the effect of raising the standard and status of the Institution and its members and provide opportunities for widening the membership. The Institution now looked to its members to approach all those they knew to be eligible and encourage them to apply for admission to the Institution.

At the risk of repetition he wished to remind members that they looked for candidates for the new class of Affiliates from amongst the members of other professions—such as doctors and mechanical and electrical engineers—whose work was intimately connected with mining and metallurgy. There might also be suitable candidates in other departments of the mining and metallurgical industries.

The need to increase the membership was an urgent one and every member should regard it as a duty to the Institution to help. In that connection he commended to members' most earnest attention what the Honorary Treasurer would have to say regarding the finances of the Institution.

A wide variety of papers had been published in the *Bulletin* during the Session and there had been some very valuable discussions at the Ordinary General Meetings. They had been fortunate in securing Dr. Desch to deliver the second Sir Julius Wernher Memorial Lecture on the 6th July. The forthcoming Symposium on the Refining of Non-Ferrous Metals had been very well organized, and some 19 excellent papers were being presented, and interesting and valuable discussions were expected at the sessions on the 7th and 8th July.

Immediately following the Symposium on metal refining, the Fourth Empire Mining and Metallurgical Congress would be held. The great task of organizing the Congress had been admirably carried out as would be clear to all those who had received the circulars of the Organizing Committee. The technical sessions at Oxford promised to be of great interest—in fact, the Congress would undoubtedly be an unforgettable experience of great importance for all who were able to take part.

The use of the Library had continued to increase and so had the inquiry service. That was a most encouraging sign.

It would be evident to members from the Report that the Institution was actively engaged in furthering the objects for which it had been founded. If that active policy was to be continued it would have to be backed by widening the membership to include all those who were engaged in the industries they represented who were eligible to join. He again reminded members that the Council always welcomed their views and suggestions regarding the activities of the Institution and the service it gave to its members.

The Honorary Treasurer (Mr. Robert Annan) said that in dealing with the Accounts he wished to say that the Balance Sheet showed very little change from that which was presented last year. The total of the Institution's investments was shown at their original cost of around £113,000. To-day's market value exceeded that figure by a little over £4,000.

The reserve for contingencies and post-war expenditure remained at the figure of £6,700. On receipts and expenditure, receipts showed practically no change in total, expenditure on the other hand continued to mount.

The cost of printing, of publications, and of stationery still continued to rise, and accounted for the expenditure of nearly £400 more in the past year than in the previous year.

The expense of meetings was reduced, largely owing to the fact that no Annual Dinner could be held last year and the main cost of the Conference on Silicosis was met in the previous year, a very small balance remaining to be paid in 1947–48.

The net result in the Accounts showed an excess of expenditure over receipts of £930; that compared with a figure of £1,130 the previous year, £800 of which had been covered by withdrawing that amount from the reserve for post-war expenditure.

The financial position of the Institution had naturally undergone some change as a result of the war. The membership showed only a small decrease in total, but the significant figure was that the number of full members had decreased from 734 to 622. He was comparing the figure with that of ten years ago. There had been a much smaller increase in the number of Associates and Students, but the net result was that the revenue from subscriptions was down by nearly £500 per annum. With the steadily mounting costs they had, as members were aware, for the last year or two been unable to cover expenditure by revenue.

The increases in salaries, pensions, superannuation, etc., amounted to 30 per cent over the pre-war figures. It was in accord with ordinary commercial experience that the general scale of salaries had risen by that amount.

The cost of publications and printing had risen by almost 60 per cent and they had managed to offset that only by economy in the rent of the premises which they occupied. That was down by 43 per cent. But he thought it would be agreed that was not situation they would care to have exist permanently. The Council felt they must look to the time when the Institution would be seeking premises more suitable for its purposes.

The Council would do everything in their power to avoid cutting down the services the Institution rendered to its members. That left them with only one practical method of getting resources and that was, as the President had said, by endeavouring by all means to increase the membership. They should particularly encourage

Associate Members who had reached the status when they could apply for full Membership to do so and not wait. It was only in that way that they could expect to continue the services which the Institution should give to its members and meet their expenses. He was sure they would all agree that any question of an increase in subscriptions would be highly undesirable.

The Institution was in quite a sound financial position. They still had £6,000 odd which had been put by during the war and they were in a position to carry on. But he did feel that they should bend their efforts to increase the Institution's revenue so that they need not continue to draw on their resources.

He seconded the proposal for the adoption of the Report and Accounts.

The President asked for questions, but none was forthcoming, and the Report and Accounts were adopted.

Re-appointment of Auditors

On the motion of **Mr. A. F. Radcliffe,** seconded by **Mr. L. M. Winn,** Messrs. Woodthorpe, Bevan & Co., Chartered Accountants, were re-appointed Auditors of the Institution for the current year.

Vote of Thanks to the Geological Society

The President moved that the thanks of the Institution be accorded to the Council of the Geological Society for the use of their rooms for the General Meetings of the Institution during the present Session.

He said that it was a very genuine expression of gratitude, for the use of the rooms was quite invaluable to them.

The Honorary Treasurer seconded the vote of thanks and it was carried with applause.

Vote of Thanks to Council, Officers and Staff

Mr. H. H. W. Boyes said that he felt very honoured in having the privilege of proposing a vote of thanks to the Council and Officers of the Institution. Members thanked them for their very hard work and for their care and guidance of the affairs of the Institution and for maintaining the status and reputation of the Institution at a very high level.

When one read that during the past year there had been 53 meetings of the Council, one realized the very large amount of very valuable time all these gentlemen had given to their interests. He felt it might be thought that sometimes some members, especially those who lived a long way from headquarters, did not appreciate the amount of work, and very good work, which was done by the Council and Officers, but he was sure that in the majority of cases that work was appreciated and that members were grateful.

As regards the permanent staff, he felt that their stalwart Secretary was really deserving of special mention.

Mr. V. O. Reid said that it gave him great pleasure to second the vote of thanks. He thought Mr. Boyes had ably put into words what they all felt, that they were grateful to the Council and Officers for the hard work they had done and for the great amount of time they had spent on the affairs of the Institution during the past year. Everyone would agree that the work they had put in had met with a very great measure of success.

That vote of thanks would be strongly supported by members scattered over the world, to whom the monthly *Bulletin* meant not only a breath of home but also an encouragement and an incentive to put their best endeavour into their particular job.

Mr. Boyes put the motion to the Meeting and it was carried by acclamation.

The President, in acknowledgement, thanked the proposer and seconder for the way in which they had spoken. It was particularly pleasing to the Council to have the expression of thanks from members from overseas, because the Council was very conscious of the needs of the overseas members, and to know that those members appreciated their efforts was very gratifying.

(The Meeting then adjourned for tea until 5.15 p.m.)

Presentation of Awards

(a) *Certificates of Honorary Membership*

The President said that the award of Honorary Membership of the Institution was given, as their Bye-laws told them, to persons of distinction in the mining or metallurgical industries, the public service, science or the arts. He had great pleasure in announcing three such awards. The first went to Viscount Nuffield.

Viscount Nuffield needed no description, for he was known the world over. He had extended his gigantic benefactions to many original and deserving causes. In 1943 he had established the Nuffield Foundation with £10,000,000-worth of Morris Motors Stock. This had been aptly described as ' his greatest benefaction, the culmination of discriminating philanthropy extending over more than 20 years '. The objects of the Foundation were the advancement of health, social well-being, and the care of the aged poor. The second object included scientific research and improvements in technical education and the provision of scholarships.

Lord Nuffield's personal and keen interest in extraction metallurgy led directly to the decision of the Trustees of the Nuffield Foundation in 1946 to devote £70,000, to be spent over a period of five years in some way which would be of benefit to metallurgical education and research in the British Empire.

Through the good offices of Dr. J. H. Watson, *Member*, the

Foundation had been put into touch with the Institution, and the Council had been invited to make suggestions for bringing into effect Lord Nuffield's intentions. As members knew, the result was the scheme for awarding Nuffield Foundation Travelling Fellowships and Scholarships.

The first awards were made in 1947 and up to the present 11 Fellowships, 15 Postgraduate Scholarships and 28 Vacation Scholarships had been awarded. The Council had been happy to co-operate with the Foundation in all those awards, and had thus been able to appreciate the great value of the scheme to metallurgical education.

In addition to those awards, the Foundation had decided to allocate funds for a Research Fellowship in Extraction Metallurgy, tenable for five years at the Imperial College of Science and Technology.

In recognition of the great benefit to the advancement of education and research in extraction metallurgy which resulted from his benefaction, the President said he had the privilege to confer on Lord Nuffield Honorary Membership of the Institution. Their only regret was that he was unable to be present to receive the award.

Mr. J. Allen Howe, O.B.E., who received the second award, had a long record of service with the Institution to his credit; he had served on the Council since 1924 and was President for two Sessions during the war, from 1942 to 1944. He had inspired the 'Memorandum on the Production of Non-Ferrous Metals and Minerals, other than coal, in Great Britain', which had been submitted by the Institution to the Government. He was responsible for the resuscitation of the Empire Council of Mining and Metallurgical Institutions at the end of the war, of which he had been acting President until ill-health caused his retirement. In consequence he was largely responsible for initiating the Fourth Empire Mining and Metallurgical Congress to be held in London this year. He had been Chairman of the Library Committee for a large number of years and a member of the Geological Survey for 30 years, becoming Assistant Director of the Survey for England and Wales.

The President said that it was with great pleasure that he conferred the award upon Mr. Allen Howe. They all regretted very much that ill health prevented him from being with them on this occasion but the following message had been received from him:

> Please convey to the President and Council my sincere thanks for the very generous award of Honorary Membership and my great regret that the state of my health prevents my attendance at the Annual Meeting.—ALLEN HOWE.

Dr. S. W. Smith who received the third award, had been a Member of the Institution since 1908 and could therefore safely be elevated to the position of an elder statesman; yet he retained his youthful manner and never-failing sense of proportion and wisdom which were freely available whenever they were needed, whether it be at the Council table or in private consultation.

He had been a Member of Council since 1926 and had been President in 1932. The Institution was particularly indebted to him for his work as Honorary Technical Editor for metallurgical papers for the last ten years. The standard of perfection which was attained in their *Transactions* was due to his painstaking work. He had had a long record of service at the Royal Mint and had held the office of Chief Assayer from 1926 to 1938.

The President took very great pleasure in conferring the award upon him.

Dr. S. W. Smith said that the very kind words of the President and the very kind reception given by those present were far more than he deserved. It had always been a pleasure as well as a privilege to be associated with those who, in successive years, had been entrusted with the affairs of the Institution. It was true that he had served on the Council for a number of years. For 21 of those years, as an *ex-officio* member, he had been immune from the uncertainties of a ballot! That unconscionable period of immunity from election had not been due entirely to defects in the Bye-laws but had arisen from circumstances which no one had regretted more than he had. It had been due to the fact that the names of six of his successors now appeared in italics in the list of Past-Presidents and their places on the Council had had to be filled by some of their predecessors in office.

In thus continuing as an *ex-officio* Member of Council he had always felt that he had been attempting in a measure to act as a trustee for those who had gone, because he had counted them all as close personal friends and had felt he could perhaps on occasion realize what their reactions would have been to problems or circumstances which had arisen. That had been his aim and object.

There were many things he would like to have said about the Institution and its activities, which were increasing in all directions. There had never been a time when the Institution had been so active in so many ways.

By no means the least part of his pleasure in receiving the award was that his name had been linked with that of his friend Allen Howe. They all regretted that Mr. Howe was not able to be with them but the speaker had seen him on the previous morning and it was to be hoped that he would be seen at Institution meetings again before very long.

Dr. Smith concluded by saying that another pleasure he would derive from the award was that he would find the President's name inscribed on the document which would always recall the happiest of memories.

(b) 'The Consolidated Gold Fields of South Africa, Limited' Premium of Forty Guineas

The President said that to Mr. F. H. Fitch, A.R.C.S., B.Sc., *Associate Member*, went a Premium of Forty Guineas for his paper on 'The Tin Mines of the Pahang Consolidated Company, Limited'. In that admirable paper the author dealt with the history of the mine and then discussed the geology with particular reference to the geological structure and mineralization. Many valuable opinions were given and the lines along which further work should be directed were set out.

That paper had evoked much interesting discussion not only at the Meeting when it was presented, but also in written contributions. It was particularly well written and most thoroughly deserved that coveted award.

As the author was in Malaya the President said he would read a letter that had been received from him a few days previously:

Geological Survey,
Batu Gajah,
Perak, Malaya.

Dear Sir, 2 May, 1949.

I wish to express, to you and the members of the Council of the Institution, my gratitude for the totally unexpected award of the 'Consolidated Gold Fields of South Africa, Limited', premium of forty guineas for my paper on 'The Tin Mines of Pahang Consolidated Co., Ltd.'. This award has given me great encouragement in the final stages of the preparation of a more detailed account forming part of a memoir, on the geology and mineral resources of an area near these mines, which should be published by the Geological Survey some time this year. The discussion on my paper was of great value to me in re-orientating and expanding my ideas on the geology of the mines, a fact which will be acknowledged in the new publication.

Yours faithfully,
F. H. Fitch.

The President was sure they were all very glad to be able to award Mr. Fitch the Premium.

Vote of Thanks to the Retiring President

Mr. L. C. Hill moved a hearty vote of thanks to the retiring President on the conclusion of his year in office.

He had heard several retiring Past-Presidents say how much they had enjoyed their term of office, but he did not think that should mislead the rest of them into assuming that a President's duties were one long round of pleasure. It was true that the President did have to represent the Institution on various occasions, but there was more to it than that.

He was sure that their President had, on their behalf, attended several dignified dinners, and had probably enjoyed them to the extent which Mr. Strachey's stringent regulations would allow, but that after all was only the lighter side of the President's duties and could be counted as a perquisite no one would deny him.

The more serious Presidential duties were carried out more or less behind the scenes, and Members of Council would realize to what extent exacting calls had been made on both the time and patience of their President, calls to which he had given such admirable attention. It was not a bad record to have presided over something like 16 General and Council meetings and to have attended up to 20 or 25 Committee Meetings, in addition to all the other duties that even they knew nothing about. That was particularly true when they remembered that the political situation in India, with its obvious repercussions on Messrs. John Taylor & Sons' responsibilities for the management of the Mysore gold mines, must have added considerably to the burden of his external duties during the year. It had entailed two trips to India for him and flying, though speedy, was a tiring way of getting about the world. On one occasion the President had stated at a Council Meeting that he had left Karachi only 48 hours previously.

Attention had been drawn in the Presidential Address a year ago to the necessity for increasing by all possible means the services rendered by the Institution to their members, especially those members who lived abroad, and the President had himself made several most valuable suggestions which were being studied and which would no doubt bear fruit in the near future.

Members of Council, over whose deliberations Mr. Taylor had presided so effectively, had drawn inspiration from his conduct of affairs, and on their behalf and on the behalf of members in general the speaker wished to congratulate him on an eventful and successful year, and to thank him most heartily for the excellent services he had rendered to the Institution during his term of office.

Mr. E. G. Lawford seconded the vote of thanks to Mr. Taylor and said that Mr. Hill had spoken so entirely to the point that there was very little left to say. In seconding the vote he would confine himself to stressing one particular point—namely, that the President's year of office had coincided with a particularly strenuous and exhausting year in his own business and he, the speaker, had watched with amazed admiration the way in which the President had shouldered this double burden.

Mr. Hill put the motion to the Meeting and it was carried with acclamation.

The President in acknowledgement thanked the proposer and seconder most sincerely for the vote of thanks. It was always rather embarrassing to have to reply to such flattering comments, and all he would say was that it was to him most gratifying to know that his efforts to discharge the duties of that responsible office had met with approval. There was no doubt that those last moments of the President's term of office were the most pleasant, and he for one felt rather like a runner in a relay race who a year ago, with trembling fingers and beating heart was handed the baton of office from the previous runner and who, having completed the full circuit of a year's course, was now within the last few strides of the base. With somewhat tired limbs but nevertheless a glow of satisfaction the baton was held out for the next runner to carry on the race.

So one President succeeded another, each inspired by the tradition of the past and the opportunities for achievement in the future.

The President, however, was not alone, for in the management of the affairs of the Institution the multitude of counsel prevailed, and each President in turn acknowledged that any success achieved during his term of office was the result of the combined efforts of his team, the Members of Council and the Secretary. Like his predecessors he, too, would generously acknowledge the splendid help and support he had received so willingly from the Council.

It was true that he had had many other duties to perform and but for the assistance and spade work which was done by Members of Council in committee and at the Council Meetings what they had seen achieved would have been impossible.

Finally, he wished to acknowledge the unfailing help and guidance he had received from Mr. Felton at all times. It should be realized that the growing activities of the Institution carried with them a corresponding increase in the volume and importance of the Secretary's work.

Induction of the New President

Mr. Taylor said he had now come to the last of his duties—to which he confessed he had been looking forward with the greatest pleasure for the past few weeks—namely, the induction of the new President.

Mr. Newman had served on the Council since 1937 and had represented the Institution on the Joint Committees on Metallurgical Education and National Certificates in Metallurgy. He was an acknowledged authority on technical education and he had served for many years at the Royal Mint where he had held the responsible office of Chemist and Assayer since 1945.

In handing over the Presidential duties to Mr. W. A. C. Newman the President knew they would be in safe and capable hands, and he wished him and the Institution every success during his term of office.

Mr. W. A. C. Newman then took the Chair and said he was very grateful and appreciative of the kind words in which Mr. Taylor had referred to him and to the few directions in which he had been able to help the Institution—as every true member would have done if he had had the opportunity.

It seemed but a few years since the late Professor Merrett signed his application form as a student. Actually it was 38 years ago, and in the intervening years the value of the Institution to the individual and the profession had been increasingly borne in upon him. Until a few months ago, however, he had had no thought that he would be occupying the position to which the Council had so generously elected him and that he would be able to range himself with the stalwarts of the past who had raised the traditions of the Institution to such a high level. He was reassured, however, by the knowledge that a good many Past-Presidents were still active in the Institution, that the Council was at all times most helpful and considerate, and that they had a Secretary and staff on whom they could confidently rely. For his part, he was all too well aware of the responsibilities which Presidency brought, but he accepted them and would meet them to the best of his ability. If he should falter—as might very well happen—he trusted they would be very indulgent.

He then delivered his Presidential Address.

(*This is reproduced on pages* 1 to 16 *of this* Bulletin.)

VOTE OF THANKS TO THE PRESIDENT

Mr. Stanley Robson proposed a hearty vote of thanks to the President for his stimulating Address.

He said that last year they had listened to a review of the Institution's activities in the past from Mr. Taylor and to his comments on the present and the future of the Institution, in which attention had been called, amongst other things, to the great importance of metallurgical education. The Address that evening had followed in natural sequence and had dealt in more detail in the same masterly way with the special problems of education.

Few people could have covered the ground in the same adequate manner as the President had done that evening. It had been the speaker's privilege to have been associated with him in a small way in some of the activities to which reference had been made, and he wished to add his tribute also to the Nuffield Foundation and the Mond Nickel Company which had given such real help to education in endowing Scholarships and Fellowships.

The President's work in representing the Institution in those matters was greatly valued and he had been told by one of the senior officials of the Nuffield Foundation that he always rang Mr. Newman up when any metallurgical scholarship problem arose and he always helped him out—such indeed was the reliance that those who worked with Mr. Newman learnt to put upon him.

Mr. Newman's services to education, his experience and the breadth of his knowledge of metallurgy were great. He was interested in both sides of metallurgy—in its physical as well as its chemical aspects. He was the Honorary Treasurer of the Institute of Metals, whose main interests were in the former field. His close touch with extractive metallurgy had been clearly set out in his

Address and, of course, he had been for many years not only an active Member of the Institution, but practised metallurgy at a very high level in his own professional work at the Royal Mint.

Mr. Robson said he did not propose to comment on the details of the Address but he would like to express his appreciation of its logic and comprehensiveness and to record his pleasure in the emphasis which had been put on the need for a broad basis of general education and his remarks about the very high standard of training necessary for the profession of metallurgy.

Professor C. W. Dannatt, in seconding the motion, said that they had listened with the greatest interest to an able and comprehensive review of a subject that had been under fierce scrutiny for the past five years or more. Reference had been made to the excellence of the President's Address, but he thought that was only to be expected. Knowing the President, his thoroughness and his extreme interest in his subject, it was inevitable that the Address would be of outstanding merit.

There had been mention of the interest that had always been taken by the Institution in the education of students. That interest was most warmly appreciated, particularly so at the Royal School of Mines, which had thereby gained the assistance and direction of Mr. McDermott. It seemed probable that that interest would increase under Mr. Newman's guidance and he hoped that it would be so.

Dr. Smith had drawn an analogy to golf. He, the speaker, would like to draw a second analogy to cricket. As a professional teacher he felt that he and his colleagues in the Institution might be considered as 'Players' whereas the President was one of the 'Gentlemen'. Should the President decide to lead an Institution team in this educational game, he could be assured of the willing support of all the Players as well as that of the other members, the Gentlemen. In such an event he trusted that the President would 'carry his bat' through a highly successful innings.

Mr. Robson put the motion to the Meeting and it was carried by acclamation.

The President, in acknowledgement, thanked the proposer and seconder of the motion for their kind expressions. He had had some qualms in thinking out and constructing something suitable to say in his Address but he hoped that what he had said would be of interest and value to members of the Institution. He had, however, only been able to touch on the fringes of certain aspects of the Institution's work.

With reference to Professor Dannatt's cricket analogy, as regards Gentlemen and Players he wished to add that in all circumstances the Professor and himself and those whom they represented all went in by the same door and came out by the same door!

Report of Scrutineers

Major H. R. Kerr, on behalf of the Scrutineers, reported the result of the ballot for the election of Members of Council for the Session 1949–1950, which was as follows :

John Caldwell Allan
George Keith Allen
Arthur Theodore Climas (*West Africa*)
Cecil William Dannatt
John Beck Dennison
Randolphe William Diamond (*Canada*)
Sir Lewis Leigh Fermor
Sir Paul Guenterbock
Henry Robinson Holmes
Norman Ross Junner
Evelyn Godfrey Lawford
Carl O. Lindberg (*U.S.A.*)
Edward Amos Loring
Robert Springett Mackilligin
John Denys Mead (*Malaya*)
Robert George Kerr Morrison (*India*)
Geoffrey Musgrave (*Rhodesia*)
John Bryning Richardson
Kenneth Richardson (*South Africa*)
John Anthony Sidney Ritson
Stanley Robson
Richard Hugh Skelton
Sir Edmund Oswald Teale
Joseph Herbert Watson
George Augustus Whitworth
Arthur Robert Owen Williams
David Williams
Oliver Holmes Woodward (*Australia*)

The President proposed a vote of thanks to the Scrutineers for their work, which was carried with applause, and the Meeting then concluded.

[*A Paper published on* 14*th July,* 1949, *for discussion by correspondence.*

Management in Industry*

By F. G. Hill, B.Sc., B.A., *Member*

SUMMARY

This paper deals with the functioning of and the training for management in industry. It makes a claim for a broader interpretation of the scope and duties of management, and suggest that South African industrialists should not neglect what is being done in the older industrialized centres to train potential and active executives for the exacting tasks of management.

Nearly every President who has preceded me in office has prefaced his address by remarking on the difficulty of choosing a subject. I was more fortunate. The subject of management is one over which I have brooded for many years. That so important a factor in industry should receive so little mention in papers read before our technical and engineering societies has always been to me a matter of concern. I long ago resolved, therefore, that if ever the opportunity arose I would prepare a paper or give an address on the subject. By electing me your President, and thus making me follow the traditional procedure of delivering a presidential address, you have given me this opportunity. In my own mind I had no choice of subject—it just had to be management. If it seems to some not wholly appropriate, I would plead that it may at least be of general interest, and has also some claim to importance.

I propose in this address to show how large a place industry occupies in the present-day world, to indicate why management is so important, and to discuss changes in organizational structure in relation to the growth and complexity of modern industry ; I want to talk also of the special problems that confront industrial managers to-day, of the tasks of management and of the methods that are being devised and used to cope with size of organization, increased technical knowledge and the growing democratic consciousness of individual employees and of groups of employees. In particular I want to make a plea for education for management. In the years that lie ahead South Africa will have to face fierce industrial competition from abroad. Are we to go to our Government and ask for high tariff walls ? for a protecting shield which increases the cost of living ? Or are we to meet competition by increasing our industrial efficiency to the uttermost ? The last course is surely preferable, but as I see it this goal will never be attained unless we have first-class leaders in industry, and we shall never have these in sufficient numbers unless the latent managerial

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abilities of our young men are, by training, brought out and developed to the full.

The Position of Industry in the Modern World

I remember as if it were yesterday the stress which our history master laid on the Industrial Revolution which shook England in the eighteenth century. As with most other children, I had come to regard history as the story of wars and kings, with a far too liberal interspersion of dates and periods. Our history master's emphasis on so mundane a thing as industry was therefore striking and made a deep impression upon us, for previously we had thought of history in terms mainly of military achievement. But it was only in later years that one came to realize what a profound effect the advent of the machine was to have on the story of mankind. In England, where the machine was first developed for large-scale industrial production, wealth rose by leaps and bounds. England's trade and power became the envy of the world. Where she had led, others were soon to follow and the world's industrial era had begun.

During the past one hundred and fifty years there have been prodigious advances, and this brief glance back at the beginnings of the era has been made only to show the rapidity of industrial growth, a growth which has not stopped, and which indeed, judging by the past twenty years, is accelerating. Industry now looms large in the life of the so-called civilized peoples of the world. It is therefore important—how important the following facts and figures will show.

In the United States of America, the most highly industrialised of all countries, there are to-day nearly 16,000,000 wage and salary workers in the manufacturing industries. The total population is some 140,000,000, so that about 11 per cent are engaged in the manufacture of goods. Another million are employed in mining. But it is not only in terms of numbers employed that industry in this great country is important—it is important, too, because of the high production per unit of labour.

Thus it has been estimated that the U.S.A., with 7 per cent of the world's population, produces 40 per cent of the world's wealth. Expressed in another way, the U.S.A., per head of population, produces nine times as much as the other peoples of the world. She has great natural resources, it is true, but it is mainly by her genius for industry that the standard of living of her people is to-day so incomparably high.

How do the South African figures compare? The proportions employed in mining and manufacturing are not dissimilar. Thus of the 11,000,000 people in this country, there were in 1945 just over 1,000,000 employees in the mining and manufacturing industries. This represents nearly 10 per cent of the population, the corresponding American figure being 11–12 per cent.

Of our present 1,100,000 employees in industry, about 460,000 are employed in mining. This latter number has changed comparatively little in recent years—from 405,000 in 1935 to 460,000 in 1948. In other industries—the so-called secondary industries—the story is very different. The war has given a great impetus to development, and the figure for employees has risen from 382,000 in 1935 to 654,000 in 1945, the last year for which official statistics are available. These figures are proof, if such were needed, that industry is coming to play a larger and larger part in our national life and economy. Any factors which may contribute to the efficiency of our industries should therefore receive thought and support from all who have the welfare of our country at heart. One of the factors most far-reaching in its effects on efficiency is management, and it is to this subject that I shall now turn my attention.

The Place of Management in Industry

The word industry brings to mind labour, machines and raw material. But these in themselves do not make up industry—labour must be directed and controlled and organizations must be created if the aims of industry are to be achieved. This function of organizing, of directing and co-ordinating the work of men and machines, goes by the general name of management. The word administration is often used in the same sense, but administration and management are not strictly synonyms. Administration is directive, management is executive. The machine which management uses for carrying out the aims of the enterprise is the organization. This paper is in some measure concerned with all three—with administration, with management and with organization—but the main emphasis will be on management as an executive function, and management will be taken to mean all those activities in industry and commerce involving responsibility for the work of others. This concept may seem general and vague, but I think responsibility for the work of others is the kernel of the matter. Management is not the prerogative of the top executives only; the management function is exercised by any person in an organization who has supervisory duties, from the general manager to the foreman on the lowest rung. An example from industry may help to make this clear. Being more familiar with the gold mining industry than with any other, I shall take as my example a small gold mine on the Witwatersrand, it being understood that my remarks about the management and organization in relation to this small mine would apply equally well to other industrial concerns.

The mine crushes about 4,000 tons of ore per month and has a labour force of approximately 40 Europeans and 300 natives. The organizational set-up is as follows:

The Manager of the mine is the chief executive officer, who is responsible to his Board for the running of the mine. To assist him in his duties he has under his direct control a mine-captain, an engineer, a reduction officer and a secretary. Each one of these men is a departmental head, that is, he is responsible for the operations in his department.

The manager's main task is to get his departmental heads to work together as a team, to co-ordinate their efforts and ensure that there are no bottlenecks which affect production. His duties are mainly to plan, to co-ordinate, to direct and to control, and in a small undertaking such as this it might be assessed that 50 per cent of his time is taken up with this type of managerial work. The remaining 50 per cent of his time may be devoted to technical problems. He cannot do the job for each of his senior staff—he must delegate authority, and the authority he delegates is mainly in respect of the technical function. The departmental head is appointed because of his special abilities for the running of that department, and he finds there are far greater calls on his technical knowledge than is the case with his manager. The manager rarely gives explicit technical details as to how a thing should be done; he indicates goals or targets and says *what* should be done; the more expert (because more specialized) departmental head gives instructions as to *how* it should be done. The departmental head's duties may be 60 per cent technical, but 40 per cent would still be managerial. As we move further down the organization the management function is still exercised, but becomes quantitatively and qualitatively of less importance. Thus the next in rank below the mine-captain is the shift-boss—who is responsible for the running of his section of the mine. He gives instructions on the drilling of holes, methods of lashing, erecting of supports and so on, and 70 per cent of his time is probably spent on this type of work. But he still has to supervise several miners and trammers and perhaps a few hundred natives. He has to organize them into an efficient production unit, and he is a manager to the extent that he is responsible for the work of others. Below him is the miner in charge of a gang of, say, 20 natives. The miner does manual technical work himself, e.g., he marks off holes and charges them up. He gives detailed instructions as to where supports should go, where lashing of rock should be done, and is responsible for his small portion of the mine. His technical work may be 80 per cent—but he still has a 20 per cent residue of management work. Under him would be native supervisors or boss-boys, each of whom would be in charge of five to ten natives. The boss-boy has to train his gang and make them do their jobs well—probably 90 per cent of his duties are technical, the remaining 10 per cent managerial. Finally the native lasher or timber boy does work which is purely technical.

The percentages quoted vary with men and size of organization. They are at best only rough approximations, and have been used merely to show the decreasing trend of managerial responsibility as one moves down the ladder.

From the top to all but the bottom grade of the organization, then, we find the management function exercised—from the manager with wide knowledge and responsibilities to the boss-boy with smaller knowledge and responsibilities, from complex problems and situations to simple problems and situations.

I have dealt with this example at some length in order to illustrate how management operates at all levels, but in this address I propose to deal more specifically with the problems facing the general manager and top executives, for it is they who feel most the impact of the changes that are affecting modern industry.

First then, let us consider management's machine—the organization—and see how its simple line or departmental structure has been changed by increased scale of operations and by increased scientific and technical knowledge.

The Organization and the Advent of Specialist Departments

The technique of mass production, evolved to satisfy the demand for cheap goods in quantity, has led to large organizations. A characteristic feature of a large organization is specialization of the work of individuals. Henry Ford's assembly lines are too well known to need description. They exemplify the extent to which specialization may proceed—a man may punch a hole in a plate day in and day out for months and years on end. The change from small to large-scale industry has meant the change from a few workers each doing several jobs to many workers each doing a specialist job. The average industrial worker in a large factory enjoys a higher standard of living than his forbears, but he pays the price of monotonous work. Increased size, then, has brought specialization, and specialization has thrown into sharp relief the need for co-ordination. There must be great competence on the part of executives if losses are not to be suffered from indifferent co-ordination. The larger a concern the greater the skill required, and indeed the growth of a business may ultimately be arrested by the limiting factor of inability to co-ordinate well.

Another feature of large organizations is the functional or specialist department as distinct from the specialist individual. By this system, 'specific functions common to all or several departments . . . are each placed in the hands of a man specifically qualified for his particular function, and instead of giving attention to all the factors in one department he gives his attention to one factor in all departments'. Examples come readily to mind. Thus the chief storekeeper of any large firm is a functional head—his department purchases stores for all departments; another example would be the officer in charge of transport, while on a gold mine we may cite as an example the compound department, which cares for native labour not only for the mining department but for the other departments as well.

This form of specialization in which departments are responsible for a specific function has given rise to the term 'functional organization'. No organizations, however, are purely functional. Thus on a large gold mine the set-up is both departmental and functional. Departmental heads are the engineer, the secretary, the reduction officer, the underground manager; functional heads are the chief surveyor and the compound manager, these last two giving specific services to the other departments. It should be noted, however, that though these specific services are functional, they are 'line' or executive services, i.e., they are basic functions for production. A mine's output would suffer considerably if the compound manager allowed the housing, feeding and general welfare of his native labour to go awry.

A functional set-up, however, tends to reverse the principle of decentralization and to throw more work on to the general manager. If the transport officer has too many calls on his transport he must refer to the general manager for rulings on priority. Other functional heads must also come to the general manager for decisions, and it is clear that too much functionalizing makes control and co-ordination remote and difficult.

Experience has proved, however, that for efficiency in a large concern there must be some functional departments, and thus the manager becomes more and more a co-ordinator, and less and less a technician giving instructions as to how jobs should be performed. He does not necessarily do more work because he has more department heads under his direct control—he merely changes the nature of his work. As his responsibilities for co-ordination increase, so he sheds some of his technical burdens and delegates authority for technical decisions to his departmental and functional experts.

Increased size of organization, then, along with vast increases in technical knowledge, have resulted in specialization and functionalization. There must then necessarily follow a high standard of co-ordination, and this emerges as the manager's main task. Appreciation of this fact is so important that no apology will be made for the continual stress it will receive in this paper.

A significant change in the structure of the organization, then, is the advent of specialist departments. The grafting of these departments on to the organizational stem was successful mainly because they were of a kind with the parent plant, i.e., they were operating or executive by nature. More recently specialist departments have arisen which are ancillary and advisory rather than operating, but their successful grafting on to the parent plant has not been so easy because of a deep-seated tendency in human nature to regard advice as criticism or interference. The best examples are perhaps the production control and personnel departments. Despite the fact that these tend to be organizationally disruptive, they are in large-scale industry fundamental to efficiency. I do not think the management of any concern with a thousand employees or more is abreast with the times unless it is making use of these two specialist departments, for in my eyes they indicate nothing more nor less than the acceptance and use of the principles of scientific management. I propose, therefore, to deal with these two departments in some detail, indicating the special managerial problems to which they give rise and suggesting the lines along which the problems may be solved.

The Production Control Department

The aim of an industrial concern is production, and control of the processes of production is of the first importance. Control is exercised by the operating heads, but experience has shown that unless they receive help from specialists the quality of control will suffer. What then is the nature of this help? Help which enables the flow of production to proceed with a minimum waste of time, of effort and of material—in other words, help that enables a factory to produce at maximum efficiency? The answer is: the supply of facts relevant to the particular problems with which operating heads are concerned. The collecting of detailed facts consumes time, however, time which an executive cannot give for the detailed study of all the operations in his department. If he is intelligent he is aware of his dilemma—he knows that he cannot make the best of plans for production because he has insufficient information, but he cannot acquire the information because he has insufficient time. The solution to his dilemma is simple and plain for all to see—in some way these facts must be made available to him so that he can plan for maximum production.

This need for studying relevant facts and presenting them to executives to help them in their planning and control has long been recognized, the great pioneer in this field being F. W. Taylor. Taylor's main doctrine was that efficient production depends on efficient control, and he enunciated several so-called principles of scientific management. I shall give a brief outline of these principles because they may equally well be termed principles of production control.

The four main principles which emerge from a study of Taylor's writings are: investigation, standardization, control and co-operation.

And first, then, *Investigation*. Taylor believed in the stop watch, in finding the time elements of operations, of becoming thoroughly acquainted with *all* the facts bearing on the processes being studied. Once this basic work had been done, and done by trained specialists, it became possible for the second principle to be applied, namely, *Standardization*. With detailed information, output standards can be determined for men and machines. Application of the third principle—*Control*—then follows, and the control is effective because it is based, not on an output which an executive thinks is attainable, but on an output which scientific study shows is in fact attainable. This last sentence may suggest that men and machines are driven to maximum effort. This is not so—standards are such that no more than a fair and reasonable day's work is expected, the rate of work being such that a man can do it for weeks, months and years on end without detriment to his health.

The final principle laid down by Taylor was *Co-operation*. He believed that everything possible should be done so that men would work with a will, for, unless this spirit obtained, the goal of maximum efficiency would ever remain elusive.

Some years ago I visited a factory in Philadelphia where textile machinery was being manufactured, and found there as neat an example of the application of these principles, and of the work of a production control department, as one could hope to find. The factory employed about 500 men and resembled a large machine shop, the control room being in the centre. In this room was a master board on which were numbers—one for each lathe, drill, milling machine, etc., in the shop. On hooks next to each number were cards showing what jobs were ready for each machine, and what materials and tools were required for each job. The operator of each machine, shortly before finishing the job in hand, pressed a button and a light went on next to his number in the control room. The controller then despatched a labourer to collect and carry the necessary material and tools to the machinist, who thus lost no time in enquiring for work, or looking for tools before commencing his next job ; he, a skilled man, did only skilled work, leaving the unskilled work of transport to be done by a labourer.

One had the impression that work in this factory went along remarkably smoothly ; the manager confirmed this impression, but stated that the present highly efficient control had been reached and was being maintained by continuous investigation on the capabilities of machines and by numerous time studies to determine standards. Special time-study men did this work, but they had long been 'accepted' by the operators, who worked co-operatively and well because the standards were fair and enabled the great majority to earn a substantial bonus.

Efficient control of production, then, must rest on facts, and the facts are best determined by specialists. Modern industry has accepted these specialists, as witness the study or planning departments on our mines, the production management departments of firms overseas, and the work done by various consulting production engineers. But it has not always been a willing acceptance, and even to-day it requires a great deal of skill on the part of management to integrate this type of specialist successfully into an organization. He is prone to be regarded by the operating staff as an intruder or as a policeman, because to them his study of facts seems too interwoven with their control. They feel that the presence of these specialists weakens their authority. Frictions tend to develop, and the manager is faced not only with a delicate problem of restoring harmony but also with the problem of demonstrating to his operating staff that their authority has not been impaired. In situations of this kind a manager cannot restore the position by a mere giving of orders ; he must define clearly to his executives and specialists the nature and extent of their duties and responsibilities ; he must stress the ancillary nature of the new departments, the undiminished importance of the production departments ; above all, if he would succeed in his task of integration and co-ordination, he must inculcate in others the scientific attitude of mind ; he should regard this as one of his major tasks, and his efforts in this direction should be persistently persistent. To the extent that he imbues his staff with the scientific approach, the new specialists will be welcomed ; they will be seen in their proper perspective as men giving service to the executives, for the facts they give enable the operating staff to exert better control : their service is the supply, not the use, of knowledge. Seen in this light there is no need for friction, and the strongest reason for co-operation. The combination of fact-finding specialists and able executives gives an organization flexibility and strength ; it is poised for progress, for its structure is such that it is receptive of new knowledge, whether this comes from study internal to the organization or from new knowledge in the outer world.

I now propose to pass on to another management tool that is difficult to use, but if mastered will yield immeasurable benefit to an organization. I refer to—

The Personnel Department

Of all those specialist departments which have emerged during the growth of industry, there seems to be none which has been so misunderstood and so suspiciously received as the personnel department. Perhaps this is because, unlike most specialist departments, its activities reach into all the branches of a business, and because all supervisors feel that the handling of employees is peculiarly personal and no part of it can be delegated to a specialist. To prevent discord, a manager must show why these suspicions and fears are groundless and why, if personnel management is properly understood and applied, it adds considerably to the well-being of employees and to the general efficiency and morale in an organization.

Let it first be said, then, that though a personnel department does certain line or operating jobs such as employment and welfare, its basic nature is that of a service department, ancillary and advisory in scope and existing primarily to help the executive staff in dealing more efficiently with their human problems. Much of its work is on all fours with that of the production, study or planning department, but in this case there is not an ascertaining of technical facts—the department collects and supplies the management with those facts which will throw a clearer light on to the employees in an organization. Examples of such facts are absentee and labour turnover figures. If these are high, there is something wrong with the human situation. But it is not the personnel department that will take remedial action : having shown the facts which indicate that all is not well, it is for the executives to take action, and in doing so they may or may not pay heed to the views of the personnel officer. An example from one of the gold mines will illustrate the point that I am trying to make.

Absentee figures showed that the sickness and accident rate on this particular mine had for years been consistently worse than the average of a large group of other mines. The absentee figures were then kept departmentally, and every month the personnel officer presented these for discussion at a meeting presided over by the manager. The responsibility for executive action lay with the departmental head, and the value of the figures was this—where previously he had been vague about absentee rates, he now had standards of comparison for his own department; if his own figures were high, it called for inquiry. The result of supplying these figures has been that for the past two years the mine's absentee figure has been consistently better than the average of the same previous group, and has in fact fallen from 6 per cent to 4·8 per cent.

This example has been quoted merely to show that one of the aims of specialized personnel work is to give more emphasis and a more systematic and scientific approach to human problems. The keeping of detailed records and statistics makes this more systematic approach possible, and enables decisions to be made on a more factual and equitable basis.

Time, unfortunately, does not permit of my enlarging on the functions of a personnel department to the extent that its workings can be made perfectly clear. This, however, would be one of the manager's duties in explaining the new set-up to his staff. He would make them see the benefits of concentrating, in a full-time, specialist department of experts, such activities as employment, training, records, welfare and employee services. The nature of the advisory services would be more difficult to explain, but he would have to go out of his way to clarify uncertainties, so that all could see the place occupied by the personnel advisory function in the jig-saw puzzle which is the organization. In brief, the personnel department's advisory work is to help executives towards a more uniform interpretation of a Company's personnel policy. If no definite policy exists, the personnel department should see to it that a policy is defined and declared. The value to a company of a declared policy is that it imposes a measure of self-discipline on executives. They have publicly pledged themselves to certain approaches and standards in dealing with human problems, and deviations from these standards are readily apparent. How much better this than chop and change and vacillation, which invariably have a most debilitating effect on general morale.

A final point to be made by a manager in explaining the work of personnel specialists is that they should never be allowed to weaken the authority of the line staff in personnel matters; on the contrary, their aim should be to help each and every supervisor to be a better personnel manager, more expert and more sympathetic in that most difficult and important task in industry, the creation of satisfying human relationships. Their ultimate concern is the well-being of employees and their very existence keeps the employer primed regarding his responsibility towards employees—a social responsibility which is an efflorescence of enlightened modern management.

The organization and its parts, then, and the relationship of the parts to the whole, have been profoundly changed by increasing size and increased knowledge, and we may now pass on to consider in greater detail some of the tasks that to-day fall to managers of large industrial concerns.

The Manager's Tasks

(1) *Co-ordination.*—First and most important, he must be a co-ordinator. We have seen that increased size of organization and increased knowledge have bred specialists ; specialists have led to functional departments, and this development in turn has led to greater centralization of authority, i.e., the manager now has under his direct control not only operating departments, but also specialist departments. His master task thus becomes co-ordination, and he has to exercise the rare art of getting executives and specialists to work as a team for the common good. This task was reasonably straightforward with operating staff, but is most difficult with advisory or service specialists. To achieve success here he must draw precisely the demarcating lines of authority and responsibility, and be so lucid and persuasive in exposition that each head not only sees exactly where and how he fits into the scheme of things, but becomes a co-operative and active worker for the good of his own department and of the concern as a whole. To achieve this co-ordination calls for conferences, discussions and meetings. The manager of modern industry cannot be the type that leads by force. Top leadership must be leadership by persuasion, for this will produce the best results in our increasingly democratic industrial world. Men must be made to feel that they are partners in industry and not servants of an employer.

(2) *Stressing management as a science.*—Next in importance in a manager's tasks, perhaps, is that he should inculcate in his staff the scientific attitude of mind, for it may be categorically stated that without this no great progress is possible. This subject has already received some consideration in discussing scientific management and the production control department, and no further elaboration is proposed except to say that to make people scientific-minded is not easy. The scientific mind is prepared to accept change. Most people, however, do not want change, which means effort, a bestirring of oneself from a groove in which life is pleasant, comfortable and in keeping with our love of ease. Science disturbs this ease and comfort, and it is small wonder that it is resisted. Any executive of experience knows that it is far more difficult to introduce a new process or way of doing things than to maintain the old ; inherent in human beings is a reluctance to change : it needs enthusiasm and the scientific approach to over-

come this reluctance and to show that new knowledge is a powerful factor making for industrial progress, and in some cases indeed is necessary for survival. It is not enough, however, that a manager should have members of staff with a scientific approach to their work—the organization should be so constructed that it enables scientific method to be readily applied. The third management task to be considered then is:

(3) *Creating an organization which can make use of the scientific method.*—The most favoured way of doing this is to have specialist fact-finding departments, examples of which, as we have seen, are the production and personnel departments. Their functions bear repetition—they are service departments which collect the relevant facts on problems and situations, and present these to the operating chiefs for decision and action. Time-studies, motion-studies, assessment of standards, aptitude testing and so on, all aim at reducing waste by the appraisal and use of detailed knowledge, and to the extent that an organization uses these techniques it may be said to be scientific.

There are, however, other features which should grace an organization if it would claim to be scientific, namely, the use of consultants and the use of research workers.

The use of consultants is generally accepted and few industries would forgo the advantages to be gained by having outside consultants to advise them on their technical problems. The gold mining industry of the Witwatersrand, based on the 'group' system, provides an excellent example of the value of such consultants. Most of our mines belong to a so-called 'group' with a controlling head office staff. Attached to the head office are various consultants—medical, mining, mechanical, metallurgical, labour, geological—men whose services are available to individual mines. These consultants acquire wide experience in a narrow field, and because they have few executive responsibilities they can keep themselves up-to-date and informed on new developments in their special spheres: they are therefore admirably equipped to advise and help the mines in the solving of problems, a service which they most effectively perform. One has only to consider such diverse matters as the feeding of mine natives and the high percentage of extraction of gold from ore, to realize that the standards achieved would never have been possible but for the work of the responsible consultants.

In general, then, it may be said that the value of consultants i industry is appreciated, in much the same way that we appreci at the services of medical specialists for dealing with uncertain (serious diseases. But to what extent do we recognize the need fo research workers?

I do not think there is any direction in which we fall bel ow overseas' standards more than in our use of research. For our smaller industries this is understandable—they have not the staff, the funds or the facilities for adequate research. But why the gold mining industry should have lagged so far behind is somewhat baffling, unless the reasons are to be found in the impermanence of mines and the comparatively small number of scientifically trained men employed. Whatever the reasons may be, the point I am trying to stress is that an industrial concern, if it wishes to have that most desirable of attributes—the scientific approach—must encourage and use research, and must keep in touch with all research activities which may throw light on its problems. The establishment by Act of Parliament in October, 1945, of the Council for Scientific and Industrial Research should give a great stimulus to research in this country. The Council is there to help industry, and it is thus a responsibility of every industrial manager to maintain a liaison with that central research body.

How to make use of the knowledge supplied by outside research, and how to integrate research workers into an existing organization —these problems are not easy, but the manager of a mine or manufacturing concern must face up to them if he would discharge his duties well. The secret would seem to lie in having a suitable organizational structure, which should be so designed that it allows for growth, for specialization, and for the easy permeation of scientific knowledge.

The fourth and last of the manager's tasks to be considered is :

(4) *Creating the will to work.*—Industry is concerned with materials, machines and men, and of these three factors, what matters most is men. There is an old saying that you can take a horse to water but you cannot make it drink ; similarly, you may give a man a machine, but you cannot make him work unless you give him sufficiently strong motives. We are here verging on the very large question of incentives in industry, and I do not wish to enter this controversial arena—all I would like to say is that a manager must be alive to all the means, monetary and non-monetary, whereby a man may be imbued with the will to work well. Rates of pay tend nowadays to be determined by negotiation between trades unions and the leaders of particular industries, and for this reason a manager cannot use the promise of monetary reward as freely as before. He must therefore rely on other means—he must see to it that the conditions under which men work are as healthy, clean and cheerful as possible. He may worship economy, but this should never be to the detriment of the employees. Supremely, he must have the human approach, the desire and the capacity to see the other man's point of view. He must show sound judgement in selecting supervisory staff, the staff that can do so much to create congenial human relationships on an organization ; he must provide for transfers and promotions on a basis of merit, and his decisions regarding personnel must bear the obvious stamp of justice. Above all, he must make men feel that they are human beings and not units, individuals with desires and aspira-

tions, not just tools to be coldly used and lightly discarded. The root of the matter is that unless people receive some recognition, some sign that their endeavours count and are appreciated, they will not put their hearts into their work.

To create an atmosphere in which men willingly give of their best—that is the ideal, unattainable because of man's imperfections, but an ideal that should surely be a beacon light to those entrusted with the responsibilities of management. Those in authority in industry should be keenly aware of these things, for on them effective leadership depends.

I do not think that any of my listeners would disagree with the views I have expressed on the importance of people's feelings, and I wish, therefore, before passing from this section of my address to refer to a recent development in industry of great significance, namely, the increasing democratic consciousness of the worker. How is enlightened management meeting the change?

In the top levels of an organization we cater for the democratic feeling by holding regular meetings with supervisors on production results, production methods, accident prevention and so on. All grades of supervisors attend meetings of some kind or other, and they thus come to feel that their views count in the running of the business. This gives them a feeling of belonging to the enterprise—they can take a pride in its achievements. But what of the worker? For years and hundreds of years management has paid small heed to his views and his voice was seldom heard. Circumstances thus drove him to form bonds with his fellow workers, led to trades unions and to the idea that the interests of management and men are different. How many managers to-day see that one of their major tasks is to break down this myth? for in conflicting interests lie the seeds of endless suspicion, antagonism and friction.

The most successful attacks on this myth seem to have been made in those concerns where workers' committees have been formed or other machinery for joint consultation has been established. In these concerns, e.g., Vauxhall Motors in England, to an extent never envisaged before, management has taken men into its confidence. It has shown them the balance sheet, has explained some of its managerial problems, and has demonstrated so clearly a sense of social responsibility, that labour has been co-operative and responsive as never before.

I know of at least one concern in South Africa where joint consultation is practised—Lever Brothers. It has not yet come to the mining industry, but if we wish to achieve full co-operation from all ranks in this great industry, some day it must come. Labour can no longer be regarded as so many hands; it is becoming educated, and must be thought of in terms of hands and heads. Due to misunderstanding of the place and function of joint consultation, the successful use of this new technique of management may still be far off. A start, however, should be made, and it is up to the employer to show the initiative in building this new bridge veen management and men. The first meetings may resolve grievance discussions, but by wise and firm handling they can ome not disintegrating but vitalizing forces. In essence they ıld be forums for the discussions of problems, and should serve hannels of communication whereby each side is apprised of the bles and perplexities of the other. In this development, y care will have to be taken that authority is not impaired, on this still depends the vigour and drive in an organization. authority, however, must be accepted, not enforced.

cannot conclude my remarks on the manager's task of creating will to work without expressing again my belief that, if he rols a concern with more than, say, 500 people, he will receive luable support from a well-chosen specialist personnel staff. y will help him and the line supervisors, as no others can, in ing industrial man live as happily as is possible with the hines that his ingenuity has created.

ı the preceding pages I have suggested that amongst the ıager's main tasks are—first, the co-ordination of the work of ırtmental and specialist chiefs; secondly, the instilling into n of the scientific approach to problems: thirdly, the creation n organization which uses scientific method; and, fourthly, ting the will to work.

his is by no means a comprehensive list, but my address is an y and not a treatise on management, and the tasks listed are ones which to me seem most important and incidentally the t difficult. If there is agreement about the importance of these s, it follows that senior staff should be selected with a view to uate fulfilment of these tasks. Mere technical achievement, ı, and training, experience or age are not sufficient. More ortant than these would appear to be personality, natural lligence, a scientific trend of mind, an innate interest in and pathy with people, and what General Smuts terms the holistic , i.e., the ability to see not only the parts of a whole, but the tionship of the parts and the contribution which each makes he functioning of the whole.

Education for Management

his address on management, in which I have touched on its ortance, its nature, its techniques and its tasks, would be mplete without some consideration of the steps that are being n in such highly industrialized countries as the United States merica and Great Britain in preparing men and women for the onsibilities of management. I shall be the first to admit that ıagers are born, not made and that management is an art er than a science. This is true of all professions—the mere iiring of knowledge is one thing, successfully applying it is her. Theoretical study alone cannot make a manager, but man has the necessary disposition and basic abilities, his nate value as a manager will be far greater if in his early years he had steeped himself in studying the principles of management. Training in this sphere, as in any other, must yield results.

These statements seem truisms: there is a need for education in management, and evidence will be produced to show how completely this need has been recognized in America, and how, increasingly, it is being recognized in Great Britain.

In October, 1945, the British Government appointed a Committee on Education for Management. This Committee reported in 1946, and included the following facts in one of the appendixes to the report:

In the U.S.A., before 1908, some 20 business schools had been established, including the Harvard School of Business Administration. Between 1910 and 1930 a further 180 schools were established. As long ago as 1924, the Balfour Committee on Industry and Trade calculated that there were as many as 80,000 students attending 100 of the schools or departments of business administration—a figure out of all proportion to the few hundreds taking similar courses in Great Britain. In the intervening years the difference has increased, for while there have been some developments in Great Britain, the output has been a mere trickle compared with the thousands of young men passed out annually in the U.S.A. with some intellectual preparation for management.

The types of courses offered fall into two categories—graduate courses and undergraduate courses.

Graduate courses are perhaps best exemplified by that of the Harvard School of Business Administration, where a two-year post-graduate course is given. The subjects of study for the first year are: Elements of administration, including production, marketing, finance, control; administrative practices, and public relationships and responsibilities.

In the second year, a student has to take a course in administrative policy, and must select not less than five specialist courses from the fields of production, marketing, finance, accounting and a special field.

The staffing and equipment of these graduate schools are most generous. Thus at the Harvard School, the faculty for the session 1945–46 included 36 professors, 47 associate and assistant professors, and 21 lecturers, research fellows and instructors; 14 of the staff were specialists in personnel management.

Between September, 1937, and June, 1942, i.e., in 5 years, the school enrolled 4,075 students, and when post-war re-organization is complete, its output will be some 800 graduates per annum. This figure alone, i.e., excluding all other schools, is more than ten times the total annual output in Great Britain of men with a [illegible]rsity training in management. A single other school—[illegible]lt Hall at Northwestern Univeristy—has accommodation [illegible] students and library space for 80,000 volumes.

[illegible]*aduate courses* vary tremendously; a report on 52 [illegible] showed that they offered 3,234 courses in the manage-

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ment field, distributed as follows: Financial and accounting, 48 per cent; distribution and transportation, 25 per cent; office management, 8 per cent; personnel, 6 per cent; production, 3 per cent; real estate, public utilities, and purchasing, 6 per cent.

They also offered 825 courses on background subjects such as economics, psychology, law and geography.

The recitation of these facts may well give the impression that the teaching of management in Great Britain lags far behind that in the United States of America. This impression is probably correct, and yet there have in the last few years been such rapid developments in Great Britain that the gap should rapidly close. The forces propelling closure of the gap are the British Government and leaders of British industry. The first-named by appointing the Committee on Education for Management, and subsequently by assisting in the birth of the British Institute of Management, has shown its live interest, while British industry, by establishing the Administrative Staff College at Henley, has given a practical demonstration of its belief in the value of training young men for high managerial posts.

The Committee on Education for Management published its report in August, 1946. It is a valuable document of 32 pages, and includes the following recommendations.

That courses leading to qualifications in management should be limited to two stages, namely, ' Intermediate ' and ' Final.'

Intermediate Course.—That all Management Professional Institutions should adopt a common curriculum for the Intermediate Course and that Technical Professional Institutes should adapt any management requirements in their syllabuses to this common curriculum.

That the Intermediate Course should consist of three parts—an introduction to management, the ' background ' subjects and the ' tool ' subjects.

The introductory subjects should consist of:—

(1) the evolution of modern industrial organization;
(2) the nature of management.

The ' background ' subjects should be:

(1) the economic aspects of industry and commerce;
(2) the legal aspects of industry and commerce;
(3) the psychological aspects of industry and commerce;

The ' tool ' subjects should be:

(1) financial accounting and cost accounting;
(2) statistical method;
(3) work measurement and incentives;
(4) office organization and methods.

Final Course.—The reommendations further provide:

That there should be two types of Final Course, one for those who wish to qualify for management in some special field, the other for those who wish to qualify in general management. The suggested syllabus for general management comprises the following subjects: Factory management, distribution, development and design, purchasing, store-keeping, transportation, personnel management, higher business control and the practice and principles of management.

It was also recommended that both Intermediate and Final Courses should lead to qualifying examinations, and that the final examination should not be written before the age of 25.

The Minister of Education approved the Report and it is believed that, by its adoption, colleges and institutions will be able to provide higher standards of education than under existing arrangements.

The second interesting development sponsored by the British Government was the creation of the British Institute of Management, which held its inaugural meeting in April of this year. This Institute is an independent, non-political non-profit-making organization having as its object the improvement of the standards of management in Great Britain. It is to be the spear-head of the drive for better management and aims to achieve its object along three main lines:

First, the compilation of knowledge about management;

Second, the promotion of education and training facilities;

Third, the propagation of knowledge about management, and the stimulation of interest in the subject.

The Government has given £150,000 to the Institute to tide it over its first five years, it being anticipated that by the end of that period the Institute will be self-supporting.

The interest shown by British industrialists in promoting education for management is demonstrated by their founding the Administrative Staff College, which was formally constituted in October, 1945, and held its first course early this year. Its establishment was planned by a group of prominent industrialists and others interested in administration, and it receives financial support from a large number of industries.

When in England recently I had the good fortune to be shown over the College, which was at one time the home of Lord Hambleden and is situated on the banks of the Thames near Henley. The Principal is Mr. Noel Hall, one-time Director of the National Institute for Economic and Social Research: he is assisted by a staff of about eight experienced men, and at the time I visited the College there were 44 members in residence. Candidates for admission may be men or women, preferably over 30 years of age. The average age of the first group was 37. They must be persons nominated by their organizations and holding senior executive positions, or selected because they will soon be holding such positions. Each course lasts about three months. There are no regular lectures. The entrants are divided into carefully-selected groups or syndicates each composed of about eight or nine members, the aim being to put together people with different backgrounds. A member of the staff is attached to each syndicate, which for any particular session elects a chairman and secretary.

The course of studies is divided into six subjects, namely:

(1) comparative administrative structures;
(2) internal organization and administration;
(3) external relations;
(4) maintaining vitality;
(5) adaption to change;
(6) role of the directing authority.

Members of the syndicate meet every morning. They debate their subject at length, read widely on it, and may have an expert come to the College to address them. At the end of the period allotted for the subject the chairman has to present a report to the whole College—a report which is a summary of the syndicate's views. Generally, each member of the syndicate acts as chairman twice during the course.

It is as yet too early to pronounce on the success or otherwise of this experiment in education, but the whole set-up of the course and the tone at the College were most impressive. Individual members spoke of it with enthusiasm; without exception they had found their discussions informative, stimulating and refreshing, and felt that they would return to their respective spheres with a broader and more balanced view of their own tasks.

From what has been said of the awareness in Great Britain of the need for education in management, it may be expected that in a few years' time the trickle of trained men will become a stream. What is the position in South Africa? We are far behind not only America and England, but also Australia and Canada, and I cannot conceive that these countries are all wrong and that we are right. To do so would savour of conceit. We are not a rich country, and if we are to make the most of our resources, we should see to it that our potential managers are as well equipped as possible for their labours.

I believe the matter is one of urgency, and is one in which the largest industries should take the lead—the gold mining industry, for example, in collaboration with say Iscor and African Explosives & Chemical Industries, Limited, might well send a commission of enquiry abroad to investigate and report on facilities for education in management. Perhaps even the Government might be persuaded to send representatives on such a commission. State support for industrial training is not unknown. The part played by the British Government in endeavouring to improve standards of management has already been mentioned, and during the last war it actively assisted in developing a scheme for the training of supervisors in industry known as T.W.I.—training within industry. The results have been so promising that the British Government to-day, through its Department of Labour, runs courses for instructors to develop the three skills of instructing, leading, and improving methods. The courses are known as *Job Instruction*, *Job Relations* and *Job Methods* respectively. The Canadian Government has adopted the scheme wholeheartedly, and a friend recently returned from Canada tells me that in that country, under this scheme, over 100,000 foremen have already been trained. We should investigate these new fields, and the suggested commission, in addition to studying broader aspects of education for management, should examine the T.W.I. scheme in detail. We send men overseas to study mechanization, developments in agriculture, designs of railway stations and a host of other things. Why not then send representatives of industry to study education for management ?

This type of education will not come unless industry asks for it, for a University or Technical College will not cater for a seemingly non-existent need. Somehow, somewhere a start should be made. If the suggested commission were to go overseas and report favourably, it might well be that a small-scale replica of the Administrative Staff College could be formed here, or a post-graduate course in administration might be started at the local University, or a T.W.I. course be established. These developments will come as surely as night follows day ; it is only a question of time, but we should not let this time linger on too long, for our industries are in their formative years, and the pattern of administration and management should be framed on the best-known techniques and on accepted principles.

As I see it then, the call is for us to pause awhile and try to see the whole field of administration and management, from directors to foreman, in proper perspective. The lead should come from the top. It is there that we most need men with vision, men to inspire, men to give the organization muscle. Without such leadership, a concern tends to become flabby, and in due course succumbs to the rigours of competition. Sir Charles Renold, Chairman of the Council of the British Institute of Management, in his speech at the inaugural meeting, referred to the need for good generalship in industry. He said : ' Good management begins at the top. Such matters as the design and manning of Boards of Directors and the clarification of the respective roles and procedures of Boards and their Executive Managements may well be, in some cases, the first steps in raising standards of management '.

Industry to-day is large, complex and seething with problems, not the least being the stresses generated by the ferment of socialistic and other ideas which are unsettling the world. The future industrial leaders, the men who will hold the reins of production, have great responsibilities—to the employees, to the country and to the owners of the enterprise. To discharge them well they need not only natural abilities but preparation for their specific tasks. The sooner we in South Africa recognize this, the sooner we give more thought to the science and art of management, the sooner will the stage be set for a happier industrial population, increased production, reduced costs, and a higher standard of living for all our people.

*** *Extra copies of this paper may be obtained at a cost of 1s. 6d. each, at the office of the Institution, Salisbury House, Finsbury Circus, London, E.C. 2.*

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FURTHER CONTRIBUTED REMARKS ON

Ground Control—Theory and Practice*

By JACK SPALDING, *Member*

Mr. V. M. S. Rajan: The articles published recently by Mr. R. G. K. Morrison in *Engineering and Mining Journal* on a general theory of rockbursts and Mr. Spalding's present paper on ground control are extremely valuable contributions to the literature on this subject in that they bring this topic of exceptional importance before the mining and scientific world again.

The doming theory as originally conceived by Fayol has subsequently been developed on the Kolar Gold Field by Messrs. Jones, Crowle, Morrison, and Spalding. The writers associated with the Kolar field have enlarged upon but seldom departed far from the fundamental principles laid down by Jones. Unfortunately Jones's work has not received the publicity it deserves and, apart from his most useful contribution to the discussion of Crowle's paper,† his fundamental calculations based on certain simplified assumptions have never been published. The theory is now beginning to be increasingly accepted in Canada and America, particularly by those who have had any practical experience of rockbursts. In view of this the actual derivation of Jones's stress formulae for spherical and cylindrical excavations, as well as his fundamental assumptions, may be of interest to the scientific workers in this field, particularly because investigations into the physics of rockbursts will be one of the most important lines of mining research in the coming years. The notation used in what follows is that used by Love in his treatise on elasticity.

1. *Spherical excavation.*

Using polar co-ordinates (r, θ, ϕ), if u_r, u_θ, u_φ are the displacements along the three axes of co-ordinates, the principal components of strain are e_{rr}, $e_{\theta\theta}$, $e_{\varphi\varphi}$ and—

$$e_{rr} = \frac{\delta u_r}{\delta r} \qquad \ldots\ldots (1)$$

$$e_{\theta\theta} = \frac{1}{r}\frac{\delta u_\theta}{\delta \theta} + \frac{u_r}{r} \qquad \ldots\ldots (2)$$

$$e_{\varphi\varphi} = \frac{1}{r\sin\theta}\frac{\delta u_\varphi}{\delta\phi} + \frac{u_\theta}{r}\cot\theta + \frac{u_r}{r} \qquad \ldots\ldots (3)$$

* *Bull.* 507, February, 1949.

† *Kolar Gold Field Min. Metall. Soc. Bull.* Vol. 4, No. 22, July-Dec., 1927, pp. 66-70.

Then, the cubical dilatation— $\Delta = e_{rr} + e_{\theta\theta} + e_{\varphi\varphi}$

$$= \frac{\delta u_r}{\delta r} + \frac{1}{r}\ \frac{\delta u_\theta}{\delta \theta} + \frac{u_r}{r} + \frac{1}{r \sin\phi}\frac{\delta u_\varphi}{\delta \phi} + \frac{u_\theta}{r}\ \cot\theta + \frac{u_r}{r} \quad \text{.....(4)}$$

For an isotropic material, the principal stresses are :

$$rr = \lambda\Delta + 2\mu\ e_{rr} \quad \text{.....(5)}$$

$$\theta\theta = \lambda\Delta + 2\mu\ e_{\theta\theta} \quad \text{.....(6)}$$

$$\phi\phi = \lambda\Delta + 2\mu\ e_{\varphi\varphi} \quad \text{.....(7)}$$

where rr is radial stress, $\theta\theta + \phi\phi$ are tangential stresses and λ is Lame's constant ; μ is the coefficient of rigidity.

If a spherical cavity of radius a in an infinite mass of rock is subjected to a hydrostatic pressure Q and it is assumed that there is only radial displacement, that is $u_r = v$, $u_\theta = u\phi = 0$, then the above equations are simplified to—

$$e_{rr} = \frac{\delta v}{\delta r} \quad \text{.....(8)}$$

$$e_{\theta\theta} = e_{\varphi\varphi} = \frac{v}{r} \quad \text{.....(9)}$$

$$\Delta = \frac{\delta v}{\delta r} + \frac{2v}{r} \quad \text{.....(10)}$$

$$\therefore\ rr = \lambda\ \Delta + 2\mu\ e_{rr}$$

$$= \lambda\left(\frac{\delta v}{\delta r} + \frac{2v}{r}\right) + 2\mu\frac{\delta v}{\delta r}$$

$$= (\lambda + 2\mu)\ \frac{\delta v}{\delta r} + 2\lambda\ \frac{v}{r} \quad \text{.....(11)}$$

$$\theta\theta = \phi\phi = \lambda\ \Delta + 2\mu\frac{v}{r}$$

$$= \lambda\left(\frac{\delta v}{\delta r} + \frac{2v}{r}\right) + 2\mu\ \frac{v}{r}$$

$$= 2\ (\lambda + \mu)\ \frac{v}{r} + \lambda\ \frac{\delta v}{\delta r} \quad \text{.....(12)}$$

$$\theta\phi = r\theta = \phi r = 0 \quad \text{.....(13)}$$

If the excavation is in equilibrium, the equation of motion is—

$$\frac{\delta rr}{\delta r} + \frac{1}{r}\frac{\delta\ r\theta}{\delta\ \theta} + \frac{1}{r\sin\theta}\ \frac{\delta\ r\phi}{\delta\ \phi} + \frac{1}{r}\ (2rr - \theta\theta - \phi\phi + r\theta\cot\theta) = 0 \quad (14)$$

Substituting equations (11), (12), (13) in (14), we have—

$$\frac{\delta rr}{\delta r} + \frac{2\ (rr - \theta\theta)}{r} = 0 \quad \text{.....(15)}$$

$$\text{i.e.,} \frac{\delta}{\delta r}\left\{ (\lambda + 2\mu)\ \frac{\delta v}{\delta r} + 2\lambda\ \frac{v}{r}\right\} + \frac{2}{r}\left\{ (\lambda + 2\mu)\frac{\delta v}{\delta r} + 2\lambda\frac{v}{r} - 2\ (\lambda + \mu)\ \frac{v}{r} - \lambda\frac{\delta v}{\delta r}\right\} = 0$$

$$\text{i.e.,}\ (\lambda + 2\mu)\frac{\delta^2 v}{\delta r^2} + \frac{2\lambda}{r}\frac{\delta v}{\delta r} - \frac{2\lambda v}{r^2} + \frac{2}{r}\left(2\mu\frac{\delta v}{\delta r} - 2\mu\frac{v}{r}\right) = 0$$

$$\text{i.e.,}\ (\lambda + 2\mu)\frac{\delta^2 v}{\delta r^2} + 2(\lambda + 2\mu)\left(\frac{1}{r}\frac{\delta v}{\delta r} - \frac{v}{r^2}\right) = 0$$

$$\text{i.e.,}\ (\lambda + 2\mu)\frac{\delta^2 v}{\delta r^2} + 2(\lambda + 2\mu)\frac{\delta}{\delta r}\left(\frac{v}{r}\right) = 0$$

$$\text{i.e.,}\ (\lambda + 2\mu)\frac{\delta}{\delta r}\left(\frac{\delta v}{\delta r} + \frac{2v}{r}\right) = 0$$

$$\text{or}\ \frac{\delta v}{\delta r} + \frac{2v}{r} = \text{constant} \quad \text{...(16)}$$

The solution of this differential equation is—

$$v = A r + \frac{B}{r^2}$$

A and B being constants.

$$\therefore \frac{\delta v}{\delta r} = A - \frac{2B}{r^3}; \qquad \frac{v}{r} = A + \frac{B}{r^3}$$

$$\therefore \frac{\delta v}{\delta r} + \frac{2v}{r} = 3A$$

The radial stress at infinity is Q and since the excavation is in equilibrium, the radial stress at $r = a$ is zero.

$\therefore rr = -Q$ at $r = \infty$ and $rr = o$ at $r = a$

Substituting in equation (11), we have:

$$rr = (\lambda + 2\mu)\left(A - \frac{2B}{r^3}\right) + 2\lambda\left(A + \frac{B}{r^3}\right) \quad \text{...............(17)}$$

If $r = \infty$, we get $-Q = (\lambda + 2\mu)A + 2\lambda A$

$$\therefore A = \frac{-Q}{3\lambda + 2\mu}$$

If $r = a$, we get—

$$0 = (\lambda + 2\mu)\left(A - \frac{2B}{a^3}\right) + 2\lambda\left(A + \frac{B}{a^3}\right)$$

$$\therefore (3\lambda + 2\mu)A - \frac{4\mu B}{a^3} = 0$$

$$\therefore B = \frac{3\lambda + 2\mu}{4\mu} A a^3 = -\frac{Qa^3}{4\mu}$$

Substituting these in (17), radial stress =

$$rr = (3\lambda + 2\mu)A - 4\mu\frac{B}{r^3}$$

$$= -Q + \frac{Qa^3}{r^3}$$

$$= -Q\left(1 - \frac{a^3}{r^3}\right) \quad \ldots\ldots (18)$$

Similarly, tangential stress $= \theta\theta = \phi\phi$

$$= \lambda\left(A - \frac{2B}{r^3}\right) + 2(\lambda + \mu)\left(A + \frac{B}{r^3}\right)$$

$$= (3\lambda + 2\mu)A + 2\mu\frac{B}{r^3}$$

$$= -Q - 2\mu\frac{Qa^3}{4\mu r^3}$$

$$= -Q - \frac{Qa^3}{2r^3}$$

$$= -Q\left(1 + \frac{a^3}{2r^3}\right) \quad \ldots\ldots (19)$$

2. *Cylindrical excavation.*

Assuming that there is only radial displacement v, as before, the components of strain are:

$$e_{rr} = \frac{\delta v}{\delta r}$$

$$e_{\theta\theta} = \frac{v}{r}.$$

$$\therefore \Delta = \frac{\delta v}{\delta r} + \frac{v}{r}$$

$$rr = \lambda\Delta + 2\mu\frac{\delta v}{\delta r}$$

$$= \lambda\left(\frac{\delta v}{r\delta} + \frac{v}{r}\right) + 2\mu\frac{\delta v}{\delta r}$$

$$= (\lambda + 2\mu)\frac{\delta v}{\delta r} + \lambda\frac{v}{r} \quad \ldots\ldots (20)$$

$$\theta\theta = \lambda\Delta + 2\mu\frac{v}{r} = \lambda\frac{\delta v}{\delta r} + (\lambda + 2\mu)\frac{v}{r} \quad \ldots\ldots (21)$$

The equation of motion is:

$$\frac{\delta rr}{\delta r} + \frac{rr - \theta\theta}{r} = 0 \quad \ldots\ldots (22)$$

i.e., $(\lambda + 2\mu)\frac{\delta^2 v}{\delta r^2} + \frac{\lambda}{r}\frac{\delta v}{\delta r} - \frac{\lambda v}{r^2} + \frac{2\mu}{r}\frac{\delta v}{\delta r} - \frac{2\mu}{r^2}v = 0$

i.e., $(\lambda + 2\mu)\frac{\delta^2 v}{\delta r^2} + \frac{\lambda + 2\mu}{r}\frac{\delta v}{\delta r} - \frac{(\lambda + 2\mu)}{r^2}v = 0$

i.e., $(\lambda + 2\mu)\frac{\delta}{\delta r}\left(\frac{\delta v}{\delta r} + \frac{v}{r}\right) = 0 \quad \ldots\ldots (23)$

$$\therefore \frac{\delta v}{\delta r} + \frac{v}{r} = \text{constant} \quad \ldots\ldots\ldots\ldots(24)$$

$$\therefore v = Ar + \frac{B}{r}$$

$$\frac{\delta v}{\delta r} = A - \frac{B}{r^2}; \quad \frac{v}{r} = A + \frac{B}{r^2}$$

Now $rr = -Q$ at ∞

$$\therefore (\lambda + 2\mu) A + \lambda A = -Q$$

$$\therefore A = -\frac{Q}{2(\lambda + \mu)}$$

If $rr = 0$ at $r = a$, we have—

$$(\lambda + 2\mu)\left(A - \frac{B}{a^2}\right) + \lambda\left(A + \frac{B}{a^2}\right) = 0$$

$$\therefore B = \frac{2a^2}{2\mu}(\lambda + \mu) A = -\frac{Q a^2}{2\mu}$$

Substituting in (20) and (21), radial stress $= rr$

$$= (\lambda + 2\mu)\left(A - \frac{B}{r^2}\right) + \lambda\left(A + \frac{B}{r^2}\right)$$

$$= 2(\lambda + \mu) A - 2\mu \frac{B}{r^2}$$

$$= -Q + Q\frac{a^2}{r^2} = -Q\left(1 - \frac{a^2}{r^2}\right) \quad \ldots\ldots\ldots\ldots(25)$$

Tangential stress $= \theta\theta$

$$= \lambda\left(A - \frac{B}{r^2}\right) + (\lambda + 2\mu)\left(A + \frac{B}{r^2}\right)$$

$$= 2(\lambda + \mu) A + 2\mu\frac{B}{r^2}$$

$$= -Q - Q\frac{a^3}{r^2}$$

$$= -Q\left(1 + \frac{a^2}{r^2}\right) \quad \ldots\ldots\ldots\ldots(26)$$

From the above it will be seen that the formulae for tangential and radial stresses are obtained for spherical and cylindrical excavations with the following assumptions :

(1) The rock is isotropic.

(2) The excavation is subjected to hydrostatic pressure and is in equilibrium.

(3) Only radial displacement is assumed to be possible.

(4) Radial stress at the surface of the excavation is assumed to be zero.

In applying these formulae to determine the mutual effect of adjacent excavations, on p. 11 of his paper Mr. Spalding estimates that the stress in the rock at the sides of the tunnel in the direction of the ring stress of the chamber will be given by $\left(1+\frac{a^2}{r^2}\right) 1\frac{1}{4}\,Q$, with a maximum at $r = a$ of $2\frac{1}{2}\,Q$. In the above example, the tangential and radial stress in the rock surrounding the chamber at $r = 2a$ are $1\frac{1}{4}\,Q$ and $\frac{3}{4}Q$, respectively.

Thus the tunnel is not driven in rock subjected to hydrostatic pressure and consequently the formula $\left(1+\frac{a^2}{r^2}\right) Q$ cannot be applied to determine the tangential stress. Again, what is tangential stress with reference to the former excavation need not necessarily be tangential stress with reference to the latter.

It is to be hoped that the problem of the effect of adjacent excavations will be tackled in a more systematic and precise way.

Mr. H. W. Sawrey*: When considering roof/hanging-wall failure over an underground opening at shallow and moderate depths, the dominant controlling factor in rocks of high cohesion must be the shear strength of those rocks. The following method, based on the theory of doming, is an attempt to arrive at a rough estimate of the ' quantities ' in simple cases of roof failure.

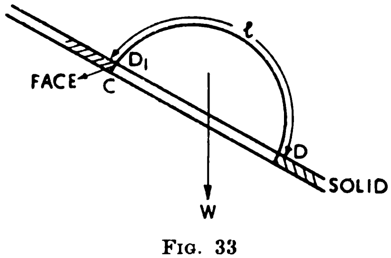

Fig. 33

Fig. 33 represents the condition of a face advancing to the rise from solid ground to the dip. The excavation is sufficiently deep to avoid surface caving, and the cohesion of the roof rock is such that a considerable area will stand unsupported.

D–D_1 represents a pressure dome about to be formed above the opening. For small variations in shear strength this should primarily approximate to a circle.

If the section is considered to have a thickness of 1 ft. then—

the force tending to closure $W = a\rho$, where a is the area beneath the dome and ρ is rock density in appropriate units; and the

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force tending to resist closure $= sl$, where s is the mean shear strength in the probable direction of movement, and $l =$ length of dome.

With bedded roofs of varying strata sl becomes $s_1l_1 + s_2l_2 + s_3l_3$......where $l_1 + l_2 + l_3 ...... = l$.

By trial and error *one* curve, and *one* only, can be drawn which satisfies the equation

$$fW = fap = sl.$$

(The factor of safety f is introduced to compensate for any horizontal pressures involved.)

If partial failure has not occurred before the point C is reached, violent failure should occur along the line D–D_1.

Laboratory tests to determine shear strength of roof rocks at various angles should be possible with a properly designed shear-box.

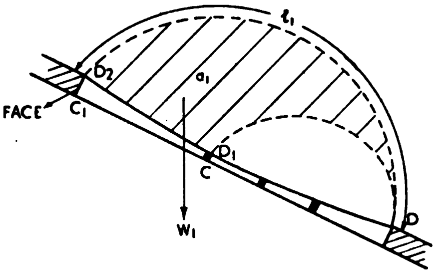

FIG. 34

Fig. 34 represents a further stage in working to the rise, at a time when the supports at C are fully compressed and closure has taken place without violence as a result of careful control.

This should result in the replacement of the critical curve D–D_1 by D–D_2 of length l_1. Curve D–D_2 can be drawn by trial and error to satisfy the expression

$$fW_1 = fa_1p = sl_1$$

(a_1 = crosshatched area).

This gives a critical point C_1.

In designing a stoping area under a roof of very high cohesion, a series of critical circles based on shear strength can be drawn by trial and error, giving points C, C_1, and C_2, etc., at which, on the foregoing premises, 'bumps' tend to occur *unless* advance is so regulated that the roof at the previous critical point is 'down' before the face has passed the next critical point. The inference is that under these conditions of face advance roof settlement would advance in surges as the face approached and passed each critical point.

The voluminous literature on this subject emanating from

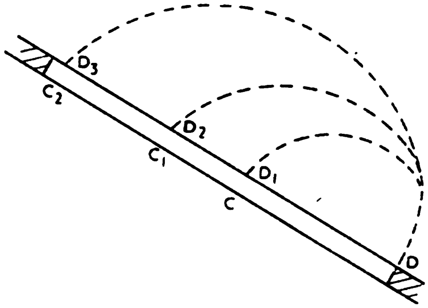

FIG. 35

colliery practice appears to deal with rocks of relatively low cohesion and shear strength. Critical curves would be, according to this theory, much steeper adjacent to the face and the pressure build-up before failure much smaller.

Adaptations of this method to cope with varying shear strength, surface subsidence, and steeply-dipping seams, etc., are possible.

AUTHOR'S WRITTEN REPLY TO DISCUSSION*

Mr. Jack Spalding: Mr. T. J. R. Sales asked if the radius of the effective excavation and the arbitrary limit beyond which stresses could be treated as trivial could not be more clearly defined. This is not possible because rocks are not consistent substances—considerable local variations occur in their physical characteristics, and there are external variations due to slips, cleavages, faults, etc., running through them, so that, even were accurate determination possible for a given rock, it could not be applied with certainty throughout even a single mine. The only application of the theory is one of comparison—one place or method is safer or less safe than another, one excavation stronger or weaker than another. This, together with local experiences as to what will stand and what will not, should be of assistance in designing layouts.

Mr. J. Norman Wynne enquired the ratio of excavated to solid ground on the Kolar Gold Field and on the Rand. At the former field over a strike length of 4½ miles and to a depth of 6,000 ft., under 40 per cent of the area has been stoped, whereas on the Rand there are areas of comparable size for which the figure is nearly 100 per cent.

* *Bull.* 509, April, 1949, and above.

As mentioned by Mr. Wynne it is possible that certain areas are more liable to rockbursts than others. This may be for local geological reasons—changes in the quality of the rock may easily bring this about—but it is more likely that the plans of stoping adopted and the changing shapes of the complex stoped areas are more instrumental in making certain areas dangerous.

The 'external influence touching off' rockbursts was a conception widely held in his day on the Kolar Gold Field, and considerable research has been done to connect the occurrence of earth tremors with such factors as temperature, humidity, barometric pressure, phases of the moon, etc. If mining operations have brought stresses at a certain place to the critical point, it is possible that, given time, creep may permit their partial dissipation and prevent a violent rock failure. On the other hand it is conceivable that an external influence, however small, might, if it occurred at the correct moment, upset the balance before the creep had had time to operate, thus bringing about the rockburst after all. Distinct correlation between earth tremors and an external factor has not, however, been found.

Tidal forces in the earth's crust are said to be extremely minute and are unlikely to have any effect: barometric change is probably the factor most likely to occasion a burst in rock in which pressure has reached the critical point. The walls of a stope are kept apart not only by the supports, but by the atmospheric pressure. A fall in this of 1 in. of mercury represents a drop of over 300 tons in the forces resisting closure of the walls of a stope measuring 100 ft. square, equivalent to the removal of one packed pigsty. Thus a sudden fall of the barometer before a tropical storm may result in a movement of the walls of the stope which may in turn possibly precipitate a rockburst. Quite apart from rockbursts this is an interesting point—with a rapidly fluctuating barometer closure can be conceived as occurring only on the falling barometer, with pauses when the pressure is rising.

This discussion should not be allowed to detract from the fact that rockbursts are caused by mining operations, not by external influences—the latter may occasionally touch them off, but the accumulation of forces which when relieved may cause a rockburst is built up entirely by the conduct of mining. None of the external influences are the *cause* of rockbursts—their energy is extremely small and they must be looked upon as 'last straws' added to a loading already in existence. This is borne out by Mr. Wynne's experience in mines in Japan during the earthquake—the loading on the rock in those mines did not approach the critical point of failure and so no tremor could cause that failure.

In the discussion, too much emphasis has perhaps been paid to rockbursts; these phenomena are merely incidental to the theme of the paper, and the publicity which they have been given should not be allowed to deceive the reader that the theories propounded and the conclusions reached are only applicable to mines where rockbursts commonly occur—they are applicable to all mines.

Mr. H. W. Sawrey's interesting contribution is welcome in that it opens up an aspect of the subject of rock pressure that is still widely misunderstood. The majority of mining men, including this contributor, still consider that what causes the closure of the walls of their stopes is the weight of the rock in the hanging-wall. Actually there are two distinct causes of closure—gravity and expansion.

When a stope is opened up, the weight of the rock within the hanging-wall fracture dome (if any) is taken chiefly by the supports.

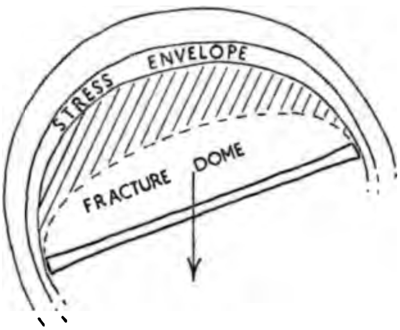

Fig. 36

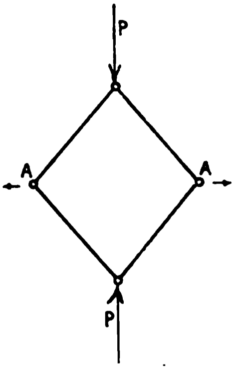

Fig. 37

This weight squeezes the supports and closure takes place (see Fig. 36); it acts vertically downwards, and so the steeper the dip the less the weight on the supports from this cause, and the less the closure. In a vertical stope with strong walls no closure due to gravity is to be expected.

The weight of the rock between the stress envelope and the fracture dome (shown shaded), or in cases where there is no fracture dome the weight of all the rock within the stress envelope, is carried by the rock itself as a beam, resting on the ends of ground and bridging the stope. This beam is loaded heavily in the centre and somewhat more lightly towards the sides. Part of the loading is transmitted to the rock in the ends of ground as the weight of a bridge is transmitted to the piers; but, unlike a bridge, part is transmitted to the supports placed under the span by the beam bending under the load.

Thus there are two forces causing closure by gravity—the dead weight of loose rock in the fracture dome, and the force exerted by the solid wall bending under the weight of rock in the expansion dome. Both these effects decrease as the dip becomes steeper.

It will be seen that there are severe limitations to the extent of closure by gravity. First, since the volume of rock within the hanging-wall portion of the stress envelope will not necessarily increase with depth, the forces which cause closure by gravity will be of the same order at all depths. Secondly, since gravity of necessity acts vertically, there can be no closure due to this cause in vertical stopes, except perhaps where the rocks are so fractured as to become mobile. Thirdly, there can be no movements due to gravity in the foot-wall.

In practice it is found that closure is not bound by these limitations—it takes place in vertical stopes and from the foot-wall, and increases with depth—and so there must obviously be some additional factor which causes it. This factor is expansion. When a substance is compressed, energy is put into it; when it is released from compression it gives back energy. All the energy put in is not, however, returned immediately, but only a portion of it depending on the elastic characteristics of the rock. For example, Sano found that for granite, subjected to a pressure equal to a depth of 6,500 ft., 40 per cent of the energy used in compression was returned on release.

In a solid rock mass at depth the rock is compressed under its own weight and is confined there, a store of energy, awaiting the chance to expand. Mining excavations afford it this chance, and it is the return of a portion of this energy that is responsible for all the phenomena of rock pressure seen in deep mining. When a stope is opened up the rock in both walls is given a free face to which it can expand. This it does until checked by its own resistance to bending or by the supports. Since the compression of the rock in the foot-wall must always be somewhat greater than that in the hanging-wall (greater depth), the expansive force, and therefore the closure taking place from this cause, must also be slightly greater in the foot-wall. In both walls the movement of expansive closure will always be at right angles to the plane of the lode.

Closure due to expansion therefore occurs in both walls and, given a hydrostatic state of stress, does not vary with the dip—it is the same in a vertical stope as in a horizontal one. On the other hand, since the original compression of the rock is proportional to the depth, the expansive force causing closure likewise increases with the depth. It will also vary with the elastic qualities of the rock—an inelastic rock, such as an open-textured sandstone or a shale, will only return a small percentage of the energy of compression put into it, whereas the more elastic the rock is the greater will the returning force be.

To sum up, the expansive forces causing closure are proportional to the depth and to the elastic properties of the surrounding rocks, but are independent of dip and wall; whereas those due to gravity vary with the dip, are independent of the depth, and occur in the hanging-wall only. In shallow flatly-dipping mines, therefore, and in mines at medium depth where the rock is inelastic the gravity effect is the most apparent; but in deep mines, and in mines of medium depth where the dip is steep or the elasticity of the rock is high, the gravity effect is masked by the greater effect of expansion. Mr. Sawrey's quantitative approach to the subject is therefore only applicable to the former class of mines, and in that only to mines where the fracture dome occupies the whole of the space within the stress envelope, which condition is common in British coal mines. It is these essential differences in the method of closure that account for the different approaches to the subject of rock pressure that have been made by coal and metalliferous miners.

Another misconception appears to be common among mining engineers. When the back of an excavation sags down, arches up, or otherwise causes trouble, it is often thought that this is the effect of a directly-acting vertical pressure. Actually this conception is too simple—a pressure cannot act vertically down through the rock and continue right down to the very skin-rock of the excavation and then stop. Unopposed by the open excavation below it, such a pressure must destroy the equilibrium, and collapse the roof. Actually the vertical pressure is diverted to pass each side of the excavation, and to bend the course of this vertical pressure a horizontal pressure is necessary.

Consider a system of links arranged in diamond form (Fig. 37): apply a vertical pressure, P, and the corners, AA, will be thrust outwards as shown by the small arrows. To preserve the equilibrium an inwardly-acting horizontal pressure must be applied to the corners, AA, to balance this tendency to thrust outwards. Where the vertical pressure, P, is greater than the horizontal pressure, Q, the linkwork must be elongated upwards as shown in Fig. 38; but where the horizontal pressure is the greater, horizontal extension is required (Fig. 39).

Substitute for the link system the rock surrounding a gallery. The links represent the stress envelope, though there should be an infinite number of them as indicated in Fig. 40, forming a circular shape (or elliptical, extended as above). Moreover, the stress should not be considered as merely horizontal or vertical, but as acting in every conceivable direction. This stress is represented in

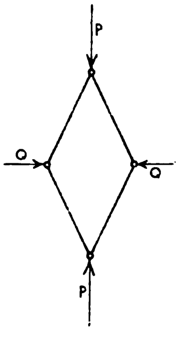

FIG. 38

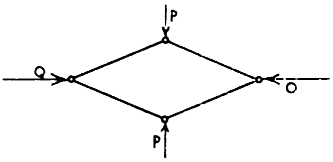

FIG. 39

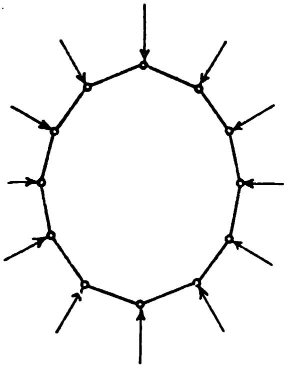

Fig. 40

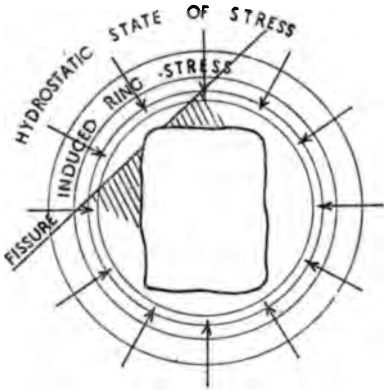

Fig. 42

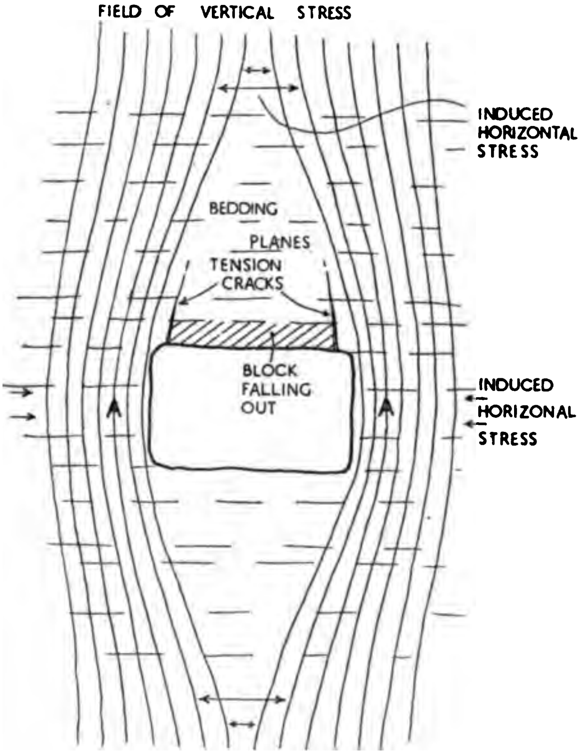

Fig. 41

the figure by the arrows. The links are thrown into compression and represent the stress ring. However, if the stresses are resolved into horizontal and vertical components, the system can be simply represented as above by four links only. The shape of the stress envelope depends then on the relative amounts of vertical and horizontal pressure. In this the writer disagrees with Mr. Sawrey's assumption of a circular dome in the case of low horizontal pressure.

In cases where there is little or no horizontal pressure it will be obvious that, in order to balance the vertical pressure transmitted through the rock to the sides of the gallery, those sides (the points AA in Fig. 37) will be caused to move outwards until they have set up a horizontal pressure sufficient to establish equilibrium, and the gallery will be slightly flattened. This flattening may sometimes cause longitudinal tension cracks to develop in the back of the excavation (see Fig. 41), causing one form of arching. Professor Sillick has already described mathematically how such a tension can arise. In flat-roofed excavations at shallow depths longitudinal tension cracks are frequently seen—they indicate the absence of a pre-existing horizontal component of pressure. Excellent examples can be seen throughout the Tura 'Caves'—adit mines into an escarpment near Cairo from which building stone was won for the earlier pyramids.

In these cases the highest stress in the rock is at AA in Fig. 41, in the sides of the gallery, and the rock may split or spall here under the pressure. Similar pressure-arching in the back, on the other hand, is more likely to be due to excessive horizontal stress than to a vertically-acting pressure.

In order to get a true conception of the stresses surrounding excavations at depth and their possible effects on the rock, the idea that the rock is an elastic substance under compression must be kept to the fore. Round an excavation such as a mine gallery it is easy to imagine the rock pressing inwards from all sides, jamming against itself and setting up the ring stress (Fig. 42). The relative amounts of horizontal and vertical pressure do not affect this conception—they merely affect the shape of the stress ring. Keeping the conception in mind the effect of a fault, or a fissure across which there is little cohesion, crossing the envelope of stress at an angle can easily be imagined, and the tendency of the shaded portions of rock to slip inwards can be anticipated.

In an elongated excavation such as a stoped area, at the ends of ground similar effects to those which occur at the sides of a gallery can be imagined, but in the expansion of the flat walls of a stoped area there can be no jamming action preventing further closure, and this closure is only prevented by the strength of the wall rock against bending and by the resistance opposed by the supports.

FURTHER CONTRIBUTED REMARKS ON

Geophysics and Economic Geology*

By J. McG. Bruckshaw, Ph.D., M.Sc., D.I.C.

Mr. Helmer Hedstrom†: When dealing with the electrical methods Dr. Bruckshaw states that of the two major divisions—surface potential observations and inductive methods—the former has been used most extensively. This statement, which is not supported by any figures (number of crew-months for each method, etc.), could hardly be correct if it applies to ore prospecting. From my 26 years of experience with electrical prospecting methods, from the crew-months statistics of electrical work of which I know, and from what I have read and heard, I would rather say that the inductive methods have had many times more extensive application than surface potential methods.

The author also discusses the well-known 'saturation effect' encountered in electrical prospecting and illustrates it with an example. He says:

> In a massive limestone of resistivity, say, 100,000 ohm.cm., there is little change in the magnitude of the anomaly if the body resistivity lies between 3,000 and 0 ohm.cm. In this range will lie the values of good conducting metal sulphides, but in addition bands of clay, graphitic schists, etc., may fall within it. They are detected with equal ease and no adaptation of this system will distinguish them.

While this is all very true for the first group of methods mentioned by the author—the surface potential methods— it does not apply to the inductive methods. For inductive methods the 'saturation effect' is closely linked to the *frequency* used. For instance, for a very large orebody of spherical shape, and with the very good electrical conductivity of 25 ohm.cm., the saturation value of the anomaly that the orebody may cause is reached at about 50,000 cycles. At 3,520 cycles the anomaly is still about 95 per cent of this saturation value, at 880 cycles 88 per cent, at 220 cycles 78 per cent, and even at the low frequency of 55 cycles the anomaly is still 60 per cent of the saturation value. Now, if this same orebody were instead to have a resistivity of 2,500 ohm.cm. ('medium conductivity') the anomaly at 3,520 cycles would be 58 per cent of the saturation value, at 880 cycles 20 per cent, at 220 cycles 5·5 per cent and at 55 cycles only 1·5 per cent.

I hope that these figures will show, without further comment, that the use of two different frequencies—e.g. 880 and 220—for

* *Bull.* 508, March, 1949.

† Aktiebolaget Elektrisk Malmletning (The Electrical Prospecting Company), Stockholm.

the same survey will readily distinguish between a very good and a 'medium' conductor.

In a paper presented before the American Institute of Mining and Metallurgical Engineers in February, 1937, printed as the A.I.M.E. *Technical Publication* No. 827, I have further shown that the ratio between in-phase and out-of-phase components of the 'secondary field' will tell the difference between two good conductors when the ratio between the resistivities is only 1 : 13.

'The major defects of the electrical methods' as summarized by the author—viz. 'a lack of discrimination, a limited depth of penetration, and a susceptibility to near-surface features which give an irregular background'—are, likewise, inherent only to the surface potential methods, not to the inductive methods. As already shown the inductive methods can discriminate between good conductors, such as metal sulphides, and conductors of 'medium' conductivity, like bands of clay, etc. As for depth of penetration, inductive methods have located orebodies at depths of 600 ft. under non-conducting rock in Sweden and Norway, according to results previously published. The susceptibility of inductive methods to near-surface features (conductors of 'medium' conductivity) can be very much reduced by the use of low frequencies, and can be nearly eliminated by using double-frequency surveys.

In the discussion of Dr. Bruckshaw's paper Mr. McPherson said that in mining, in looking for orebodies, the geophysical methods were a doubtful weapon ; there had been hundreds of cases of geophysical work carried out with few successes. He considered that geophysical methods should only be employed under the control of an experienced geologist who thoroughly understood their limitations. The last sentence quoted would be heartily endorsed by all serious mining geophysicists, who have always looked upon the geophysical technique as a 'tool for the geologist'. In fact, every geophysicist worthy of the name would himself have sufficient knowledge and experience of mining geology to be able to advise, from reports by 'local' geologists, from tests on samples and from inspection of the grounds, whether geophysical methods should be resorted to or not in a specific case. There are many cases where geophysical methods should *not* be used, and it is the practice of geophysical contractors of good standing to investigate prospecting problems thoroughly before they undertake contract work, and to advise against the use of geophysical work in cases where it would probably not be economically useful.

If the usefulness of geophysical methods in mining is to be judged from Mr. McPherson's 'hundreds of cases' it would therefore only be fair to exclude all those cases where the above procedure has not been followed, where the work has been done by commercialized 'go-getting' new-comers who call themselves geophysicists, and, for instance, where the client has not been a mining company with a good geological department, but some obscure ' prospecting syndicate ', with directors who ' do not believe in geology ' but want to impress their shareholders by being progressive and using the latest and most modern divining methods. If Mr. McPherson's statistics were to be limited to those cases where *serious* geophysical work has been employed, there is no doubt that his percentage of ' successes ' would rise very appreciably.

Mr. McPherson did not define what he meant by the word ' successes '. I do hope that he did not mean that a geophysical survey is successful only when commercial ore is found! After all, it is the mining company employing the geophysicists that chooses the area to be surveyed. Because orebodies are very rare occurrences it is natural that most of the areas chosen for investigation do not contain an orebody. When the result of a geophysical survey and of the subsequent exploratory work is therefore correctly negative, this cannot very well be reported as an ' unsuccessful ' geophysical survey. If the word ' unsuccessful ' is to be used, it would in this case rather apply to the mining company's choice of area to be prospected.

To a serious geophysicist the measure of ' success ' of a geophysical survey is simply the money that the survey has saved in the total programme of a prospecting campaign. It cannot be emphasized too strongly that geophysical work should be regarded only as one step in a sequence of a complete prospecting programme and as an economic measure used to effect savings in the total cost of exploration. In such a prospecting programme it is left to the orthodox methods of trenching, digging, shaft-sinking, and drilling actually to find the ore—if ore is present—but, because these methods are time-wasting and costly, it is often economical and wise to *eliminate* as much as possible of the area involved from the application of these methods, by detailed geological investigations and thorough geophysical work; the real economic importance of the geophysical methods lies in this *elimination of barren ground.*

In the discussion Mr. McPherson emphasized his belief that in most instances ordinary methods of prospecting by geological examination combined with trenching and shallow underground work, followed by drilling and deeper underground work, were the cheapest. If this reasoning is to hold good, however, it is necessary that geological examination should be *possible*, either on an exposed bedrock surface or in shallow trenches. However, in typical ore-prospecting nowadays it is not possible for a geologist to examine the bedrock surface over a sufficiently large part of the area to be prospected. (If it were, the outcropping ores would certainly already have been found—most probably long ago; the ' Ancients ' were very clever prospectors.) It is precisely in these, the usual cases, where geophysical work—the new ' tool for the geologist '—should be applied, in order to give geological information where the bedrock is hidden to the geologist and where it is considered to be too costly to expose it through trenching, shallow core-drilling, and so on. A necessary condition is, of course, that the mineralization looked for is such that will cause geophysical anomalies, or, at least, that geophysical work can be useful for other reasons, as exemplified on the Rand. If this is not the case, a serious geophysicist will explain that geophysical work cannot be usefully employed to cut the costs of exploration, and the only way out is then the 'ordinary methods of prospecting' referred to by Mr. McPherson, which are generally not at all cheap, but costly and time-wasting. Because the decision whether or not geophysical work should be applied is made on geological, geophysical and *economic* grounds, it follows that the use of serious geophysical work in a prospecting programme will always make the exploration cheaper (and faster) than the use of the ordinary methods referred to by Mr. McPherson.

In a typical case of exploration, where geophysical work is used, we may assume that the geologist is able to eliminate as barren, say, four fifths of the area to be investigated. The geophysicist may be able to eliminate most of the remainder, leaving, say, only eight or ten 'suspicious' places, where exploratory work may be concentrated. Out of these remaining places, a trained and experienced geophysicist can usually (for instance, by the use of several geophysical methods) eliminate another part, leaving only a few places where exploratory work has to be resorted to. This, of course, does *not* mean that there must necessarily be commercial ore in these remaining 'suspicious places'—it means only that one has been able to save the cost of exploratory work in a great many other places on barren ground. When this entire exploration programme has been completed, a comparison of the total cost with the calculated cost of a complete investigation of the whole area *without* the use of geophysical work will show whether the geophysical survey has been 'successful' or not. The question whether ore was found or not, in one or more of the 'suspicious places' indicated by the geophysical work, does not enter into this consideration.

The great savings effected by the use of geophysical work in an exploration programme has made possible prospecting campaigns that would not have been economically feasible without the use of such methods. The ore discoveries made in the course of these campaigns—about 100 cases in Sweden only, from 1918 onwards —can therefore be credited to the use of geophysical methods, even though the actual 'discovery' has always been made in drilling or shaft-sinking on geophysical anomalies, which for the most part did not correspond to commercial orebodies. The people who decided on the use of geophysical work in these cases, and who have continued to use it all along, were exactly of the type referred to by Mr. McPherson—that is, their 'essential purpose in life was not geology or physics but to find and exploit profitable ores and to find them by the cheapest methods.' They also fully understood that 'one could not mine anomalies' and that it was necessary 'to start in by the usual methods to find out what those anomalies meant'. Still, they found that it was good business to use geophysical methods—just as the most hard-headed business men of the American oil industry have found out. In this connection I would like to point out that Mr. McPherson went to the other extreme when he said that there was no question whatsoever of the value of geophysical methods in oil exploration. There are certainly many cases in oil exploration, just as in ore prospecting, where geophysical methods cannot be profitably applied, and the question of the value of these methods has to be decided in each separate case from the geological conditions and from test surveys.

Mr. McPherson admitted that 'in certain special cases geophysics could be particularly useful' to locate a strongly-magnetic or good-conducting orebody, but he added the condition that the orebody in that case would have to be 'covered by a few tens of feet of gravel, moraine, or other similar shallow cover'. He also said that 'in Sweden certain valuable lenses of auriferous copper sulphides and certain iron bodies had been traced or located successfully under shallow cover'. This statement is misleading. In the first place it is not a question of 'certain valuable lenses', but of a large number of orebodies, some of which are very big. Further, there is no question of 'auriferous copper sulphides' only, but of a large variety of orebodies of varying composition. Again, the 'shallow cover' over some of the orebodies located has been between 50–100 ft., and in one instance 165 ft. The greatest depths at which actual orebodies have been located by geoelectrical work in Sweden and Norway is 600 ft.—not under covers, as described by Mr. McPherson, but under barren rock. Iron orebodies with no outcrop at the bedrock surface have been found in Sweden during the last three years by diamond-drilling on geophysical anomalies at depths to the upper boundary of the orebody of, for instance, 325 ft. (a workable ore), at 500 ft. (a large orebody) and at 700 ft. (a very big and valuable orebody).

Mr. McPherson also said that he 'did not know of any ordinary type of orebody containing only lead and zinc sulphide minerals occurring under normal conditions in which the orebody could be definitely located by geophysical methods'. It is true that pure zinc sulphides have no electrical or magnetic properties that differ from 'normal' country rock. Their density is comparatively high, however, and in one case in Sweden during the last war a zinc ore was located by a drill-hole on a gravimeter anomaly. Lead sulphides, on the other hand, are extremely good electrical conductors—and for that reason many lead-zinc ores are electrically conducting, in part or throughout their whole mass. Thus, in Sweden the zinc-lead ore of Ammeberg, Stollberg, and Gruvberget, for instance, give definite electrical anomalies (with the inductive method). In a paper published in 1938 I have shown an example of 'phase anomalies' which were caused by very thin lodes of poor lead-zinc mineralization under a cover of moraine. Among zinc-lead ores that have been located by geoelectrical work in Sweden are E. Högkulla (only lead and zinc), Gränsgruvan (lead-zinc with very little pyrite) and Rävlidmyran (14 per cent Zn, 4 per cent Pb and about 0·1 per cent Cu).

[redacted]

FURTHER CONTRIBUTED REMARKS ON

Recovery of Sulphur from Smelter Gases by the Orkla Process at Rio Tinto*

By H. R. Potts, *Member*, and E. G. Lawford, *Member*

Mr. A. V. Bradshaw :† Owing to the decline in the use of the blast furnace for smelting copper ores, there has been little published on this subject in recent years ; thus this paper, which supplies such informative data on both plant and practice, is most welcome. Much repetition would result were I to give, as the authors suggested, a complete description of the plant of Mason and Barry, Ltd., at S. Domingos, Portugal, since in many respects the two plants are very similar. I will, however, give a brief description of the principal differences. in the respective plants.

At S. Domingos there are two furnaces designed for the Orkla process. No. 1 furnace, which smelts 150 tons of pyrites per day, was blown in for the first time during October, 1934, and is described in greater detail in what follows. No. 2 furnace smelts 100 tons of pyrites per day and was first blown in as a sulphur producer in July, 1946 ; this is simply a smaller edition of No. 1 furnace.

The furnaces differ from the conventional type in that the ends are boshed and the outlet spout is in the centre of one side of the furnace crucible. Above the furnace crucible—which is not water-cooled—are suspended two tiers of water jackets bolted together. Prior to ' blowing in ', the upper jackets are lined with fire-brick and the lower jackets are lined with ordinary red brick. This brickwork is soon fluxed away, but time is allowed for the formation of a protective slag coating, reducing local overheating and abrasion from the falling charge during the initial stages of a fresh campaign. The lower jackets last at least three years and the upper six years. The matte and slag run into movable forehearths of about 90-cu.ft. capacity, which are lined with a mixture of silica clay and sand. The furnace crucible is similarly lined. Although the forehearth lining lasts only a couple of months it is a simple operation to change and reline a settler and this is found more economical than using magnesite bricks. Chrome bricks are used for the tap-holes of both the furnace and forehearth.

There are two hot Cottrell chambers for No. 1 furnace, each with a cross-sectional area of 3·8 sq.m. The volume of gases passing equal 274 cu.m./min. at 380°C. and the gas velocity is 0·6 m./sec. (2 ft./sec.). At times precipitation is impaired by the condensation of a layer of arsenical sulphur on the electrodes,

* *Bull.* 509, April, 1949.

† Smelting Department Superintendent, Mina de S. Domingos, Portugal.

TABLE X

No. 1 FURNACE DETAIL

No. of charging bells	3
Height from bottom of bells to floor	23 ft.
No. of top jackets	12
Width at top of jackets	5 ft. 9 in.
Length at top of jackets	17 ft. 7 in.
Area at top jackets	101.5 sq. ft.
No. of lower jackets	12
Width at tuyères	3 ft. 3¼ in.
Length at tuyères	16 ft.
Area at tuyères (hearth area)	52·5 sq. ft.
No. of tuyères	30
Tuyère diameter	3 in.
Tuyère area	1·47 sq. ft.
Height of upper jackets (perpendicular)	3 ft. 3¼ in.
Height of lower jackets (boshed)	7 ft.
Bosh angle of side jackets	77° 35 ft.
Bosh angle of end jackets	81° 43 ft.
Area at top of furnace/hearth area	1·94
Hearth area/tuyère area	35·6
Tons mineral/sq. ft. hearth area/day	2·81
Tons burden/sq. ft. hearth/day	4·14

which builds up and eventually causes a flash over. This is remedied by closing the gas valves and opening up the Cottrell. With the admission of air this arsenical sulphur is burnt out.

No. 1 furnace cooler consists of water-filled tubes, around which the gas flows. The cooler surface of 317 sq.m. represents 2·15 sq.m. per ton of mineral smelted per 24 hr. or, expressed in another way, 21 cu.m. gas measured at N.T.P. are cooled per sq.m. of cooler surface per 24 hr. The gases are cooled from 370°C. to 155°C. and steam is produced at 50 lb./sq.in. The Rio Tinto cooler appears slightly more efficient, but this is probably due to working at two pressures. The authors' statement that the S. Domingos type of cooler is better for precipitating sulphur mist, is not borne out by tests, since in our process only about 40 per cent of the total sulphur comes from the cooler. I should, however, imagine that there is less possibility of choking due to dust accumulation than in the fire-tube boiler.

There is one sulphur Cottrell on No. 1 furnace of 3·8 sq.m. cross-section. The gas volume is 178 cu.m./min. at 150°C. and gas velocity is 0·78 m./sec. or 2·5 ft./sec. The current flowing varies between 30 and 40 milli-amp.

Sulphur refining is carried out continuously. The crude sulphur is pumped to the top of a vertical tower and cascades over a series of trays, coming into contact with milk of lime (40 g.CaO/l.) which is injected with steam into the lower portion of the tower. The sulphur then passes to the horizontal washer, where it flows counter current to the milk of lime which has entered the other end of the washer from the top of the vertical tower; mixing is assisted by a rotating shaft, to which are attached concave paddles. The waste lime liquors are discharged and the sulphur passes to the filters. Washing is carried out at 28 lb./sq. in. pressure. The lime consumed is 88 kg. per ton of refined sulphur produced.

There are two sulphur filters working in parallel. Each filter consists of two concentric steam-heated cylinders 11 ft. 6 in. high. The inner cylinder is perforated and is covered with an asbestos cloth. Washed sulphur enters the 3-in. space between the cylinders, passes through the asbestos cloth and leaves from the centre, whence it is pumped to deposit. The filters are cleaned by lifting off the outer cylinder and scraping the cloth. This is usually done once a week after filtering about 125 tons of sulphur. We have also suffered similar trouble to that described when thin impermeable residues are formed which contain up to 14 per cent lead. Filter cloths usually last for about one year and filter 5,000 tons of sulphur. The washed and filtered sulphur contains 0·06 per cent arsenic and 0·04 per cent ash. A typical daily charge of No. 1 furnace is given in Table XI.

The ratio of concentration copper in matte to copper in mineral is 2·9 : 1. The overall sulphur recovery over the last four years averaged 53·5 per cent.

Experience here confirms the authors' remarks on the necessity of having an acid flux with a high available silica content as well as the advantage to be gained in using a sized charge. At S. Domingos gossan is used as a siliceous flux on account of its precious metal content but in several instances when pure silica only was used an improvement in the sulphur recovery was observed. In addition to the difficulty of obtaining a satisfactory focus with an impure silica, the ferric oxide present is reduced by carbon or carbon monoxide, leaving less carbon available for the reduction of sulphur dioxide. Our normal practice was to smelt ore direct from the mine, large pieces being broken by hand; recently, however, we have been smelting sized mineral *plus* 2 *minus* 6 in. and this has resulted in an improvement in recovery of from 2 to 3 per cent. There has also been a reduction in the blast pressure, probably due to absence of fines in the charge, which would segregate and cause an uneven distribution of gas in the furnace shaft and therefore reduce the contact time between the sulphur dioxide and carbon.

If furnace design is at all important one would have imagined that the wider, more heavily boshed, No. 6 furnace would have shown different results from the narrower ones. Are there any figures available showing the relative performances of these furnaces? Owing to the heavier boshing of No. 6 furnace, and hence the lower relative speed of the gases in the furnace shaft, it would have appeared desirable to drive it faster than Nos. 4 and 5, whereas the tonnage smelted per sq. ft. of hearth area is apparently lower.

The authors' statement that there is no dissociation of pyrites by heat beyond the stage Fe_nS_{n+1} is in contradiction to Peters and the older authorities. This assumption certainly helps to account for the large amount of carbon necessary in the Orkla

Table XI

	Tons	Cu	S	SiO_2	Fe	CaO	Zn	Pb	As	Ash	*g./ton* Ag	Au
Pyrites	150	1·50	46·0	2·1	41·1		3·0	0·7	0·5	—	12	—
Quartz	16·6	—	—	98·0	—	—	—	—	—	—	—	—
Gossan	30·2	—		72·0	12·3	—	—		1·4	—	68	4
Limestone	11·0	—		—	—	53·0		—	—	—	—	—
Matte	10·0	4.35	—		—	—		—	—	—	—	—
Coke	12·3	—	—		—	—	—	—	—	—	—	—
Production												
Matte	58·3	4·35	23·1		—	—	2·7	—		—	76	3
Slag	108	0·12	2.9	35·1	37·9		3·0	—	—	—	—	—
Dust	1·9	1·2	30·6		19·2	—	7·4	11·4	—	—	—	—
Crude Sulphur	40·1	—	—	—	—	—	—		2·5	0·7	—	—

process, since the formation of Fe, FeS would result in the production of much more volatile sulphur and much less sulphur dioxide. In the experiments referred to, what was the maximum temperature to which the pyrites was heated, and was the sulphur, distilled off in the early stages of heating, removed ? Since we know that the matte does not contain sufficient sulphur for all the iron to be as FeS and therefore even less if it is as Fe_7S_8, presumably the Fe_7S_8 combines with FeO—

$$3Fe_7S_8 + 2FeO = 23FeS + SO_2.$$

The concentration of copper obtained in the Orkla process is very low and is due to the relatively high amount of coke on the charge and low amount of blast per ton of pyrites. From Table I and the subsequent figures in the authors' paper, the principal sulphur-producing reactions in the Orkla process may very approximately be summarized as:

$$3F3_7S_8 + 30{\cdot}5O_2 + 19{\cdot}5C = \underset{\text{(Slag)}}{14FeO} + 4SO_2 + 19{\cdot}5CO_2 + 6S_2 + \underset{\text{(Matte)}}{Fe_7S_8} \quad \ldots\ldots\ldots\ldots(1)$$

(The sulphur produced by direct distillation, the formation of COS and CS_2 and the percentage of CO_2 escaping unreduced from the focus have not been considered.)

From this reaction it is obvious that any increase in carbon without an increase in the blast per ton of ore smelted will cause a decrease in the concentration and, furthermore, there is a definite limit to the amount that can be added. For example:

$$3Fe_7S_8 + 30{\cdot}5O_2 + 24{\cdot}5C = 12{\cdot}48FeO + 24{\cdot}02CO_2 + 0.48CO + 7{\cdot}13S_2 + 1{\cdot}22Fe_7S_8 \quad \ldots\ldots\ldots(2)$$

Since equilibrium conditions are not always attained in the furnace, some carbon sulphur compounds are formed prior to the complete reduction of SO_2. If the concentration is to be maintained then it can be calculated that approximately 50 cu.m. of extra air is required for each increase in the carbon content of 1 per cent on the mineral. Even so there is a limit to the amount of carbon that can be added. On the other hand an increase of 33 per cent in the volume of air and 50 per cent in the amount of carbon over reaction(1) will produce:

$$3Fe_7S_8 + 40{\cdot}25O_2 + 28{\cdot}5C = 17{\cdot}5FeO + 3SO_2 + 28{\cdot}5CO_2 + 8.5S_2 + 0{\cdot}5Fe_7S_8 \quad \ldots\ldots\ldots(3)$$

All these equations presuppose that there is sufficient free silica at the focus to combine with the FeO. Preliminary tests here indicate that a higher concentration can be obtained, but since our blowing capacity is limited this results in a lower throughput and lower production. Greatly increased blast pressures were observed, also the furnace ran very full with high gas temperatures, and it seems that to obtain a high degree of concentration a taller furnace is desirable.

As can be seen from the equations given gas analyses are only satisfactory as a check on recovery if the quantity of gas per ton of ore and the sulphur lost in the matte per ton of ore is taken into consideration. At S. Domingos flow measurements of the sulphur running from the cooler and Cottrell are taken hourly as a check on the recovery.

The most economical percentage of carbon used will obviously depend largely on local conditions. At S. Domingos current practice is to use 6·9–7·4 per cent C on the mineral (8–8·5 per cent coke). If all extra coke added was to be converted to carbon monoxide prior to reducing any SO_2, then allowing for 10 per cent of CO_2 formed at the focus to escape unreduced, 790 kg. C should be required per ton of sulphur reduced. However, when the volume of gases per ton of ore is taken into account, Table III, in the authors' paper, shows that, by increasing the carbon from 6·1 per cent to 7·33 per cent, between 1·6 and 2·1 tons of carbon are required for each ton of sulphur loss in the waste gases, the exact figure depending on whether the carbon in the limestone is included in the figure giving carbon charged per ton pyrites. On increasing the carbon to 10·4 per cent it is evident that the total sulphur lost in the waste gases is much greater. It would be interesting to know the sulphur lost in the mattes per ton pyrites for each period.

The calculation of the relative amounts of SO_2 reduced by carbon and carbon monoxide, the latter being formed by burning carbon at the focus, can at the best be but very approximate. The percentage CO_2 which escapes unreduced from the focus will greatly influence the results, depending on the contact time between the gases and the carbon at various temperatures. The exact quantity of carbon in and the temperatures of the various zones in the furnace can only be roughly estimated, but it is difficult to envisage a contact time greater than 0·2 sec. at a temperature above 1200°C. and 0·4 sec. between 1200° and 900°C. If these times are correct then it is probable that considerably more CO_2 than that calculated escapes unreduced.

It is well known that charcoal reduces CO_2 more rapidly at lower temperatures than coke. Might not charcoal also reduce SO_2 more efficiently than coke. During the war much charcoal was used at S. Domingos and the results were not unsatisfactory. In 1945, 10·65 per cent charcoal and 0·58 per cent coke were used, giving an overall sulphur recovery of 59·79 per cent. Subsequent tests using charcoal have not enabled high recoveries to be obtained, but I think this is due to the poor quality of later supplies, some containing up to 30 per cent ash. Have the authors any experience of using charcoal at Rio Tinto ?

The writer desires to acknowledge the courtesy of the directors of Messrs. Mason and Barry, Ltd. in allowing details of their practice to be published, as these have not previously appeared in print.

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CONTENTS

INSTITUTION NOTES

Radium and Radon for Industrial Radiography

The Ministry of Supply wishes to make known to industry that the Government-owned Radiochemical Centre at Amersham can now accept orders for radium and radon as sources of gamma rays for use in industrial radiography.

Radiography, which is the most versatile of all non-destructive testing techniques, makes use of the penetrating power of electromagnetic radiations of very short wavelength. The two main types of such radiation are X-rays and gamma rays. The penetrating power of gamma radiation is comparable with that of the most powerful X-ray apparatus at present available for industrial radiography. For some industrial purposes gamma radiography may be the more convenient or only practicable method.

As in the case of X-radiography the successful use of gamma radiography requires trained staff, competent to interpret radiographs, and familiar with the special precautions necessary against occupational hazards. So far as is known the Kodak School of Engineering Radiography, Wealdstone, Harrow, Middlesex, is the only institution offering training at present, but the Ministry of Supply is making enquiries from the Ministry of Education regarding the possibility of courses being started elsewhere.

In general, the associated equipment is simple and, if properly designed, is safe to handle. The source of radiation is compact, so that the equipment is extremely mobile. It can frequently be made by the user firm.

The Ministry of Supply Radiochemical Centre at Amersham supplies radium and radon for industrial use. Applications should be addressed to The Radiochemical Centre, White Lion Road, Amersham, Bucks. (Telephone: Little Chalfont 2278-9), with a detail specification of the capsule required. Consultation with the Centre will be necessary to ensure that technical requirements are met.

Radium (which is best suited to factories and foundries where there is a continuous flow of radiographic work) is supplied only on hire for periods of not less than six months. Capsules containing 50 mg. and 250 mg. of radium are in common use and the present rentals for these are £11 5s. 0d. and £56 5s. 0d. respectively per annum. The radium remains the property of the Ministry of Supply and is not transferable by the hirer. The hirer is also required to pay the cost of fabrication and testing of the radium and its container at present averaging £20 for a 50-mg. capsule and £100 for a 250-mg. capsule. The average time needed to fulfil an order for radium is two months.

Radon is sold outright. Typical present costs are: £15 for a 250-millicurie source, £20 for 500 millicuries, and £25 for 750 millicuries. One week's notice of an order is required.

Demonstrations of gamma radiography can be seen without charge by appointment with

(i) The Superintendent of Radiology Research, Ministry of Supply, Armament Research Establishment, Woolwich, London, S.E. 18 (Telephone: Woolwich 2044. Ext. 129),

or (ii) The Director, National Physical Laboratory, Teddington, Middlesex (Telephone: Molesey 1380).

Advice on the suitability of proposed applications of radiography in industry may be obtained without charge from the Armament Research Establishment and the National Physical Laboratory.

The Armament Research Establishment may also be consulted by industrialists for advice on the principles of radiograph interpretation.

Correction

In the report of the Annual General Meeting published in the July *Bulletin*, Mr. S. H. Ford in his statement on the Benevolent Fund is reported to have said 'cash in hand, in the Bank and in the Post Office is down by £50'. The figure he actually gave was £270, the figures for 1947 and 1948 being, respectively, £2,136 11s. 4d. and £1,866 7s. 4d. The error is much regretted as it lessened the force of Mr. Ford's appeal for funds by making the position appear much less serious than it is.

Meetings in the Session 1949-1950

General Meetings of the Institution during the Fifty-Ninth Session will be held on the third Thursday in each month from October, 1949, to June, 1950. The dates of the Meetings are as follows :

20th October, 1949
17th November, 1949
15th December, 1949
19th January, 1950
16th February, 1950
16th March, 1950
20th April, 1950
18th May, 1950
15th June, 1950

Members from Abroad

The Council are always anxious to meet members who come to England after a long absence abroad, and ask such members to make themselves known to the Secretary when attending General Meetings of the Institution at Burlington House.

Institution Awards

'The Consolidated Gold Fields of South Africa, Limited' Gold Medal and Premium of Forty Guineas are awarded jointly or separately by the Council of the Institution for the paper or papers of highest merit contributed to the *Transactions* during each Session, or for researches on the occurrence, mining, or treatment of minerals. The Council shall be satisfied that the papers or researches are of sufficient merit to justify the award.

Two prizes of Ten Guineas each are offered annually for papers contributed to the *Transactions* by Students of the Institution, provided that the papers are, in the opinion of the Council, of sufficient merit to justify an award.

Papers for the consideration of the Publications Committee should be sent to the Secretary, if possible in duplicate, and should be prefaced by a summary of contents. It is understood that all papers submitted are original communications unless distinctly stated to be otherwise, in which event exact reference should be made to any previous publication. Figures illustrating papers should be drawn in ink, suitable for direct reproduction in a reduced size, and lettering on drawings should be in ordinary pencil. If there are photographic illustrations, prints on glossy paper should be sent ; it is not necessary to send negatives.

Candidates for Admission

The Council welcome communications to assist them in deciding whether the qualifications of candidates for admission into the Institution fulfil the requirements of the Bye-laws. The application forms of candidates (other than those for Studentship) will be open for inspection at the office of the Institution for a period of at least two months from the date of the Bulletin *in which their applications are announced.*

The following have applied for transfer since 14th July, 1949 :

To MEMBERSHIP—

Thomas Haden (*Bulawayo, Southern Rhodesia*).

Gilbert Frederick Hatch (*Johannesburg, Transvaal*).

To ASSOCIATE MEMBERSHIP—

William John Alborno (*Konongo, Gold Coast*).

Peter Alexander Nicholls (*Tarkwa, Gold Coast Colony*).

Moda Nagesa Rao (*Champion Reefs, Southern India*).

Gerald Derek Sheridan (*Avoca, Eire*).

The following have applied for admission since 14th July, 1949 :

To MEMBERSHIP—

Stacey George Ward (*Edgbaston, Birmingham, Warwickshire*).

To ASSOCIATE MEMBERSHIP—

William Davies (*Sheffield, Yorkshire*).
John Hogg (*Airdrie, Lanarkshire*).
Thomas Arthur Waller (*Krugersdorp, Transvaal*).

To AFFILIATESHIP—

George Furnace Brown (*Huelva, Spain*).

To STUDENTSHIP—

William Edward Cliff (*Camborne, Cornwall*).

Alan By Nicholls (*Camborne, Cornwall*).
Brian Harry Coles Waters (*Cambridge*).

Transfers and Elections

The following were transferred (subject to confirmation in accordance with the Bye-laws) on 6th July, 1949 :

To MEMBERSHIP

Raoul Gustin Bergman (*Paagoumene, New Caledonia*).
John A'Court Bergne (*Great Missenden, Buckinghamshire*).
John George Berry (*Ghatsila, India*).
Robert Pitman Hooper (*Broken Hill, N.S.W., Australia*).
Gustav Anthony Schnellmann (*Egremont, Cumberland*).
Patrick Francis Whelan (*Bristol, Gloucestershire*).

To ASSOCIATE MEMBERSHIP—

Donald Campbell Hitchings (*Brighton, Sussex*).
John Francis Murray White (*London*).

The following were elected (subject to confirmation in accordance with the conditions of the Bye-laws) on 6th July, 1949 :

To MEMBERSHIP—

Gordon Colvin Lindesay Clark (*Melbourne, Australia*).
Louis Lionel Colin (*Vila de Manica, Portuguese East Africa*).
William Reid (*Crossgates, Fife*).

To ASSOCIATE MEMBERSHIP

Fernleigh Edmondson (*Johannesburg, Transvaal*).
John Gallaway (*Hayle, Cornwall*).
Ames Gresley Hellicar (*Chester, Cheshire*).
Harry Reymond Miles (*Tarkwa, Gold Coast Colony*).
Francis Stark (*Oorgaum, Southern India*).

To AFFILIATESHIP -

Harold David Blackburn (*Newcastle-upon-Tyne, Northumberland*).

To STUDENTSHIP—

Robert Rennie Bell (*Johannesburg, Transvaal*).
William Bernard Hall (*N'kana, Northern Rhodesia*).
Kenneth Wright Jackson (*Gateshead, Co. Durham*).
Alan Smee (*Barrowford, Lancashire*).
John Henry Warnock (*Kidderminster, Worcestershire*).

News of Members

Members, Associate Members, Affiliates, and Students are invited to supply the Secretary with personal news for publication under this heading.

Mr. H. J. ALEXANDER, *Student*, has accepted a post with Messrs. Balfour, Beattie & Co., Ltd., on tunnelling work in Scotland.

Mr. G. W. BELLMAN, *Student*, is returning to England from South India.

Mr. R. P. BRODIE, *Associate Member*, has left England to take up a position with Consolidated Tin Mines of Burma, Ltd.

Mr. A. T. CLIMAS, *Member*, is in England on leave from the Gold Coast until the end of September.

Mr. W. R. DEGENHARDT, *Member*, has retired from his position as mechanical engineer to New Consolidated Gold Fields, Ltd., but is acting for that company in a consultative capacity.

Mr. E. A. FOLLOWS, *Student*, will be returning to England in December from the Transvaal.

Mr. E. P. HARGRAVES, *Member*, has arrived in England from Australia.

Mr. H. L. H. HARRISON, *Member*, is returning to Malaya this month.

Mr. R. G. HEAD, *Associate Member*, has recently been appointed chief engineer in charge of mine planning and design, Mufulira Copper Mines, Ltd.

Mr. K. A. KAWAR, *Student*, is leaving England for Amman, Transjordan, this month.

Mr. P. G. LINZELL, *Student*, has left A.O. Nigeria, Ltd., to take up the position of surveyor for Ashanti Goldfields Corporation, Ltd.

Mr. R. C. MCADAM, *Student*, has left England for Southern Rhodesia.

Mr. I. H. MCLEAN, *Student*, has left England to take up an appointment with Central Provinces Manganese Ore Co., India.

Mr. W. E. SEVIER, *Member*, expects to arrive in England early in September from British Guiana.

Mr. K. J. ST. GEORGE, *Student*, has joined the staff of Roan Antelope Copper Mines, Ltd., Luanshya, Northern Rhodesia.

Mr. S. J. VENNING, *Associate Member*, has relinquished his post as reduction manager on the Grootvlei Proprietary mines to take up the position of assistant managing director with an engineering and foundry combine at Benoni, South Africa.

Addresses Wanted

D. S. Broadhurst. J. A. Cocking. E. Dickson. R. B. Hicks. G. C. Morgan. A. I. Scott. A. Sloss.

OBITUARY

George Carter died on 2nd July, 1948, at the age of 73. He received his training at the Mining Schools, Penzance, and the Camborne School of Mines from 1891 to 1893, and at the Royal School of Mines in the following year. He went to South Africa in 1895 on obtaining a position at Robinson Gold Mining Co., Ltd., and joined Roodepoort United Main Reef Gold Mining Co., Ltd., in 1898 as cyanide manager and assayer. He subsequently transferred to Nourse Deep, Ltd., where, from 1899 to 1905, he rose from assistant surveyor to head surveyor. Mr. Carter worked for Nourse Mines, Ltd., until 1910, becoming assistant manager in the last year of his service, and in 1911 was appointed head surveyor to Ferreira Deep, Ltd. Shortly afterwards he joined Transvaal Gold Mining Estates, Ltd., first in the position of resident manager to Vaalhoek mine, and later, from 1912 to 1914, as resident manager of Central mines. He was then made acting general manager to the Company for six months, and from 1915 to 1917 resumed his position with Central mines. Mr. Carter joined the Royal Engineers in 1917 and until 1919 served as a lieutenant with the 175 Tunnelling Coy. in France. On demobilization he took up the appointment of manager of the Llanharry iron ore mine, South Wales, for the Glamorgan Hematite Iron Ore Co., Ltd., Cardiff. He retired in 1945 after sixteen years in that position, and went to live in Penzance.

Mr. Carter was elected to Membership of the Institution in 1916.

Arthur Delmar Combe died suddenly on 23rd May, 1949, at the age of 56. He was born in Australia and received his training in economic geology at the Sydney Technical College from 1908 to 1913, at the same time attending lectures in mining and surveying at Broken Hill Technical College and instruction in assaying and analysis at the laboratories of Messrs. Pale and Cameron. He had some practical and general underground experience at Broken Hill Junction North mine, N.S.W., in 1912, and worked at the cyanidation and flotation plant at Chesney mine, Cobar, in 1913. In 1914 he was employed at the Mt. Morgan Gold Mining Co. and the South Clifton colliery, Scarborough, N.S.W., and in 1915 at the North Lyell mine of Mt. Lyell Mining and Railway Co., Tasmania. He returned to New South Wales in 1916, and during the next three years obtained practical experience at the Great Cobar, South Blocks, Junction, New Burragorang, Broken Hill South and Broken Hill Block 14 mines. In 1919 he joined Scotchmans Gold Mining Co. at Stawell, Vic., and subsequently worked for Bendigo Amalgamated Gold Mining Co., New Red White and Blue Consolidated Gold Mining Co. and Rose, Thistle and Shamrock Gold Mining Co. He was employed for two months by Commonwealth Oil Corporation at Newnes, N.S.W., before a similar short period at Seaham No. 2 colliery, West Wallsend, N.S.W.

In March, 1921, Mr. Combe was appointed field geologist to the Geological Survey of Uganda, and remained with the Survey until his death, when he held the position of assistant director.

Mr. Combe was elected to Associateship of the Institution in 1926.

Nathaniel Malcolm died in 1948 at the age of 53. He began his professional training at Otago School of Mines in 1914, but joined up on the outbreak of war and left New Zealand as a private, eventually gaining a commission. On his return in 1919 he worked for a few months at Alexandra Coal Co., Ltd., and Blackwater Mines, Ltd., New Zealand, before resuming his training at Otago, which he completed in 1921. For a brief period he worked as prospector at Red Hills, West Coast, and miner at Nightcaps Coal Co., Ltd., and in 1922 left New Zealand to take up the position of sampler to Randfontein Estates Gold Mining Co., Ltd. He was transferred to the post of surveyor in 1923, and to shift boss in 1924. Two years later he was appointed manager of Luipaardsvlei Farm No. 10 of the Coronation Syndicate, Ltd., Transvaal, leaving at the end of 1928 to become assistant general manager to Empress Base Metals, Ltd., Que Que, Southern Rhodesia. After twelve months Mr. Malcolm returned to the Transvaal as mine overseer to Government Gold Areas, Ltd. From 1930 to 1931 he was manager to Potgietersrust Platinum Mines, Ltd. (Rustenburg Section), and then returned to Government Gold Mining Areas, Ltd., as mine overseer and acting sectional manager. He held the position of manager of Rustenburg Platinum Mines, Ltd., from 1933 to 1941, and then took over the management of East Champ d'Or Gold Mining Co., Ltd., until 1945.

Mr. Malcolm was admitted to Studentship of the Institution in 1920, was elected to Associateship in 1926 and to Membership in 1937.

Amos Treloar, J.P., died on 25th May, 1949, in Cheltenham Nursing Home at the age of 71. From 1891 he was trained as a miner and ore-dresser at Wheal Metal and Flow & Fortune mines, Breague, Cornwall, and in 1903 became manager of the Breague Valley Mining Corporation mines. After three years he took up the appointment of chief tin and wolfram dresser for the East Pool and Agar mines. He went to the Transvaal in 1910 as manager of the dressing works of Rooiberg Minerals Development Co., returning two years later to the post of instructor in ore-dressing at the Bessemer Laboratory at the Royal School of Mines. In 1914 he took over the management of Wheal Kitty and Penhalls mines, Cornwall, subsequently becoming manager of South Polgooth Mining Corporation, and in 1919 took charge of Tyndrum lead and zinc mines in Perthshire for four years. His subsequent appointment in 1923, which he held for over 25 years until his death, was to the position of manager of the lead and zinc mines of the Vieille Montagne Zinc Co. at Alston, Cumberland.

Mr. Treloar, who was joint author with Gurth Johnson of a paper contributed to the *Transactions* of the Institution entitled ' The separation of tin oxide from wolfram ' (vol. 17, 1907–8), was elected an Associate of the Institution in 1905 and was transferred to Membership in 1925.

Henry Morley White died on 1st July, 1949, at the Royal Cornwall Infirmary, Truro, at the age of 71. He was educated at Homefield House School, Camborne, Cornwall, and in 1893 began work in the Carn Brea and Tincroft mines under his father, where he learned surveying, and also attended Redruth School of Mines. In May, 1898, he obtained a position with Messrs. John Taylor & Sons as under agent at the Fortuna mines in Spain, and two years later was transferred to the Linares Lead Mining Co., in 1903 being advanced to the position of chief agent. Mr. White was appointed mine agent and assistant manager at the Cordoba Copper Co.'s mines at Cerro Muriano, Spain, in 1909, where he remained until 1919. He then went to India on joining Champion Reef Gold Mining Co., Ltd. He was appointed chief mine agent in 1920 and in 1934 became superintendent of Champion Reef, continuing in this position until his retirement in 1945. He returned to live in Cornwall, and in 1948 was elected President of the Cornish Mining Development Association.

Mr. White, who served as Member of Council for India from 1939 to 1943, was elected an Associate of the Institution in 1905 and was transferred to Membership in 1913.

The Council regret to announce the death of **Robert Claude Gibbs,** *Associate Member*, on 10th June, 1949; and **Geoffrey Musgrave,** *Member*, Member of Council for Rhodesia, on 13th July, 1949.

BOOK REVIEWS

Oil Shales and Shale Oils. By H. S. BELL. New York: D. Van Nostrand Co., Inc. (London: Macmillan & Co., Ltd.), 1948. 157 p., illus. 22s.

This American book, by summarizing existing information on oil shale and shale oil and by drawing on present-day practice in the American coal-mining and oil-refining industries, presents a picture of probable methods of producing shale oil from the vast deposits known to exist in that continent.

The economics of the various processes involved are also reviewed from the rather inadequate information available.

The first chapter consists of an historical review of shale oil production, notably in France and Scotland, but the industries of Australia, Manchuria, Sweden and Estonia are also briefly described.

Oil shale mining methods employed in Scotland form the basis of another section, which also includes descriptions of modern American coal mining machinery and methods which might be used in oil shale mining. Reference is also made to opencast coal mining methods.

In view of its importance in any shale oil industry, the section on retorting receives inadequate treatment. It consists principally in a fairly detailed description of Scottish and Estonian retorting methods, but fails to mention the Westwood retort of the Scottish industry as indicative of most recent practice in that country. Retorts in use or projected in other countries are also illustrated.

In the oil refining section most of the space is taken up with brief descriptions of petroleum refining equipment.

Much of the information on shale oil production is derived from the papers presented at the 1938 Glasgow Conference on Oil Shale and Cannel Coal.

A. STEWART.

' AnalaR ' standards for laboratory chemicals. 4th edn. Formulated and issued jointly by The British Drug Houses, Ltd., and Hopkin & Williams, Ltd., London, 1949. xviii+302 pp. 10s. 6d.

The publication of a new edition of ' AnalaR ' standards will be welcomed by all who are engaged in the practice of assaying. The companies concerned are to be congratulated on the production of the book, providing, as it does, valuable information on the testing of the purity of a wide range of analytical reagents.

In this new edition, the opportunity has been taken to revise and enlarge the contents. The number of reagents listed has increased and many of the tests have been modified.

Fifty-eight new reagents have been added to the series and it is worth noting that amongst these additions are compounds of ten elements not previously represented. One compound—anhydrous ferric chloride—has now been eliminated from the series.

Improvements have been made in many of the tests employed and many of the specifications have been revised, both with regard to the progress of analytical chemistry and to the requirements of users of the reagents.

Polarographic analysis has been introduced in several cases for the determination of impurities. The polarographic method has been found to give rapid and accurate determinations and its more extensive use is forecast.

Electrolytic deposition, whilst not a new technique, has been introduced into the range of tests employed. It is used for the assay of certain metallic salts and also for the removal, in certain cases, of the principal metal in order that the traces of impurities present may be more easily determined.

The availability of a series of reagents containing known limits of impurities is of the utmost value to an assayer. The existence of an up-to-date volume concerning the precise details of the tests employed in the determination of the impurities is, consequently, of the greatest importance.

F. L. SELFE.

ADDITIONS TO JOINT LIBRARY OF THE INSTITUTION AND THE INSTITUTION OF MINING ENGINEERS

*Books (excluding works marked *) may be borrowed by members personally or by post from the Librarian, 424, Salisbury House, London, E.C. 2.*

Books and Pamphlets:

TIMBER DEVELOPMENT ASSOCIATION LTD. *The use of timber in mining*, by H. Henshaw. London: The Association, 1949. 84 p., illus., map, diagrs., tabs., biblio.

Government Publications:

SCHEUMANN, Karl Hermann, senior author. *Petrography: part 1, minerals; part 2, minerals and ores.* (F.I.A.T. review of German science, 1939-1946.) Germany: Military Govt., 1948. 234; 145 p., biblios. (German text.)

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550.8 Prospecting

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553.5 Deposits of building stones

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622.233.3

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Subject to revision.] [*A Paper issued on* 11*th August*, 1949, *for discussion by correspondence.*

Experiments on the Removal of Selenium and Tellurium from Blister and Fire-refined Copper*

By W. A. Baker,† B.Sc., F.I.M., and A. P. C. Hallowes,‡ B.Sc., A.I.M.

Introduction

Selenium and tellurium impair the properties of some types of copper when the amounts present exceed certain limits, and methods of removing these impurities are therefore required. Their removal by electrolytic refining is entirely satisfactory where power supplies and precious metal recovery, etc., are favourable, but in some cases there is a considerable economic incentive to develop alternative cheaper processes.

The present paper describes experiments on the following methods for the removal of selenium and tellurium:

(1) Removal during conversion of matte,
  (i) by volatilization, and
  (ii) by selective converting.
(2) Removal during fire-refining by the addition of calcium and other elements.

The paper is a record of laboratory work only and is put forward in the hope that engineers in charge of converting and fire-refining operations may possibly find ways of applying, on a plant scale, the more successful of the experimental methods described.

It is thought that some of the methods described could be applied under full-scale working conditions, but it must be emphasized that the paper does not attempt to do more than set out results obtained in experimental work on a laboratory scale.

I. REMOVAL DURING CONVERSION OF MATTE

(i) Removal by Volatilization

Like bismuth, selenium and tellurium are not removed by oxidation and transfer to the slag during conversion of copper mattes. The elements boil at 680°C. and 1087°C. respectively and if they were uncombined in molten smelter products a substantial removal by volatilization would be expected. Keller's [1] early work indicated that substantial amounts are eliminated in the

* A communication from the British Non-Ferrous Metals Research Association. A summary of experimental work by A. P. C. Hallowes.
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(1) See list of references given at the end of the paper.

converting process and the following small-scale experiments were carried out to confirm or disprove his conclusion.

Experimental

350 g. melts of matte, white metal, or synthetic 'oxygen-free' coppers, were held in 2½-in. dia. by 10-in. deep salamander crucibles and blown with air or hydrogen in three temperature ranges—viz. 1250–1300°C., 1350–1400°C., and 1450–1500°C.; 35 g. of sand was added to the matte charges blown with air. The blowing time was usually four hours and the gases were passed through the melts at 4 cu. ft. per hour, so that the total volume passed per unit weight of melt was of the same order as the volume of air blown in a typical converter operation [2], although the speed was much lower. At the end of the blow the melts were poured into a thick-walled copper mould.

The initial charges and the products after blowing were analysed* for selenium and tellurium contents and the extent to which these elements were removed was calculated as described in what follows.

Results

(*a*) *Air-blown mattes* yielded slag *plus* matte or slag *plus* white metal and copper. Mechanical losses of the order of 20 to 30 per cent occurred by splashing and by skulls adhering to the crucibles. The total amounts of selenium and tellurium which would have remained in the charges had no losses occurred were therefore calculated from: (i) the contents of these elements in the end products and (ii) the theoretical yields of the end products, these being indicated by the iron content of the residual matte or by the extent of copper formation. In these calculations the small amounts of the impurities in the slag were neglected. The extent of removal arrived at in this way is shown below.

Matte containing 0·040 per cent Se and 0·035 per cent Te blown 4 hours with air

Temp. °C.	Removal per cent	
	Se	Te
1250–1300	6	3
1350–1400	nil	3
1450–1500	2	20

In view of the possible errors in the calculation of yields, the only significant loss of tellurium occurred at the highest temperature and no selenium was removed under the conditions used.

(*b*) *Hydrogen-blown mattes* (with the same initial selenium and tellurium contents) formed a few per cent of free copper, but this was neglected in arriving at the conclusion that no significant losses of selenium and tellurium had occurred.

(*c*) *Hydrogen-blown white metal* (containing 0·061 per cent of selenium and 0·042 per cent of tellurium) formed a little free copper.

* The method used involved double precipitation with sodium hypophosphite and gave results reproducible to ± 0·001 per cent.

No significant losses of selenium occurred and although a 14 per cent loss of tellurium occurred at 1450–1500°C. no loss was indicated at lower temperatures.

(*d*) *Hydrogen-blown synthetic copper melts* gave the results shown below.

Synthetic coppers containing 0·040 *per cent Se and* 0·055 *per cent Te blown* 4 *hours with hydrogen*

Temp. °C.	Removal per cent Se	Removal per cent Te
1250–1300	2	5
1450–1500	16	41

The small losses indicated at the lower temperature are not significant but some removal of both selenium and tellurium occurred with a reducing blast at 1450–1500°C.

(*e*) *Air-blown copper melts* containing 0·043 per cent of Se and 0·056 per cent of Te formed very little slag and lost no selenium or tellurium after blowing for one hour at 1450–1500°C. After 4 hours blowing in the same temperature range, about 15 per cent of the copper was oxidized and formed a silicate slag which contained about 45 per cent of copper, 0·009 per cent selenium, and 0·007 per cent tellurium. The selenium and tellurium contents of the remaining copper were unchanged, so that about 10 per cent of the total amount of each impurity had been removed, presumably by volatilization.

(ii) *Removal by Selective Converting*

Selective converting—i.e., partial conversion of white metal to copper and separation of the two products before continuing the blow—is a familiar process, which has been applied, for example, to the conversion of white metal of high bismuth content. The bismuth concentrates preferentially in the copper, so that, by interrupting the blow soon after copper formation has begun and separating this copper before converting the remaining white metal, the bismuth content of the bulk of the copper can be drastically reduced. The literature [3, 4] indicates that selenium and tellurium distribute themselves in the opposite way, both tending to concentrate in the white metal. However, the magnitude of this effect is not clear from the literature and both small-scale and large-scale tests were carried out to determine the extent to which these impurities can be removed by selective converting.

Experimental

Two mattes were used, one having high selenium and tellurium contents and the other a low selenium content with negligible tellurium. In laboratory tests 350 g. charges of matte, together with 35 g. silica, were heated in deep (10-in. by 2½-in. dia.) salamander crucibles to 1450–1500°C. and blown with air at 4 cu. ft. per hour for times ranging from 3 to 7 hours. The resulting slag, white metal, and copper was poured into a heavy copper mould and the products, which separated into distinct layers in the mould, were analysed for their selenium and tellurium contents.

The distribution of these elements between white metal and copper was also examined by melting together 150 g. each of pure synthetic white metal and of copper containing the impurities. These melts were stirred with a silica rod for 10 min. at 1250–1300°C. and then cast into a copper mould, where the two components separated into two distinct layers from which samples were taken for analysis.

These two procedures were also followed on a larger scale. In the first experiment 2 cwt. of matte, with 10 per cent of its weight of silica, was air-blown in an oil-fired furnace until it was judged, after 6 hours blowing, that 80–90 per cent copper formation had occurred ; this was judged from the appearance of an iron rod plunged into the bath at intervals. The dense and high-conductivity copper layer heated the lower end of the rod to a distinctly higher temperature than did the white-metal layer and the depths of both layers were easily measured in this way. It proved impracticable to decant the white metal from the copper bath, after skimming off the slag, but the white metal formed a thin layer on the top surface of the copper billet poured from the melt.

In the second experiment about 1500 lb. of white metal was decanted from a converter into an oil-fired furnace and about 6000 lb. of molten clear copper was slowly added. The bath was mixed by poling for ten minutes. An attempt to skim off the white metal was unsuccessful and the contents of the furnace were poured into a ladle lined with reverberatory slag. Good separation of white metal and copper was obtained.

Results

The results of these experiments are given in Tables I and II. The proportions of copper and of co-existing white metal are expressed as percentage copper formation. The last two columns of the Tables show that the selenium and tellurium were concentrated in the white metal, so that with suitable proportions of white metal and copper most of the selenium and tellurium was retained in the white metal.

II. REMOVAL DURING FIRE REFINING OF COPPER

Attempts were made to remove selenium and tellurium from copper at some stage in the fire-refining process, in which certain impurities in the blister copper are removed by oxidation and transfer to the slag.

The alkali selenides and selenites (and the corresponding tellurium compounds) have high heats of formation and it has been proposed [5, 6] to remove selenium and tellurium from molten copper by treating the latter with alkali salts under reducing conditions. The extent of removal by this method was examined

TABLE I

SELENIUM AND TELLURIUM CONTENTS OF COPPER AND WHITE METAL AFTER SELECTIVELY CONVERTING MATTE CHARGES

	Copper		White Metal		Copper Form-ation	Proportion of Total Se or Te present in Cu *per cent*	
	Se	Te	Se	Te			
	per cent	*per cent*	*per cent*	*per cent*	*per cent*	Se	Te
350 g. charges of matte containing Cu 58·5 per cent, Fe 14·2 per cent, S 24 per cent, Se 0·040 per cent, Te 0·035 per cent	0·0026	0·001	0·055	0·043	2	Nil	Nil
	0·009	0·007	0·061	0·050	14	2	2
	0·0025	0·0046	0·063	0·050	17	1	1
	0·0038	0·0046	0·072	0·054	28	2	3
	0·009	0·009	0·074	0·035	30	4	9
	0·004	0·006	0·078	0·050	32	2	4
	0·007	0·008	0·082	0·052	36	4	6
	0·003	0·008	0·089	0·062	49	2	9
	0·003	0·012	0·095	0·051	59	4	21
	0·005	0·015	0·107	0·077	63	6	21
	0·0096	0·010	0·114	0·084	69	13	17
	0·013	0·014	0·137	0·103	77	20	27
	0·013	0·015	0·142	0·108	83	26	35
	0·014	0·019	0·087	0·080	84	40	49
	0·018	0·017	0·113	0·100	91	56	57
	0·036	0·026	0·29	0·15	99	89	92
	0·064	0·053	—	—	100	100	100
	0·055	0·037	—	—	100	100	100
350 g. charge of matte containing Cu 64·8 per cent, Fe 8·8 per cent, S 22·3 per cent, Se 0·0067 per cent, Te about 0·0005 per cent	0·0003	n.e.	0·013	n.e.	32	1	—
	0·0004	n.e.	0·016	n.e.	47	2	—
	0·0008	n.e.	0·023	n.e.	66	5	—
	0·0015	n.e.	0·044	n.e.	86	13	—
	0·0079	n.e.	—	—	100	100	—
2 cwt. charge of matte containing Cu 64·3 per cent, Fe 9·48 per cent, Se 0·0068 per cent	0·0041	—	n.e.	—	97	40	—

n.e. Not estimated.

Table III

The Effect of Alkali Salt Treatment on the Selenium and Tellurium Contents of Molten Copper

Conditions of Experiment	Se *per cent*	Te *per cent*
‘*Oxygen-Free*’ *Copper*		
(*a*) Charcoal cover and frequent poling. Four experiments, three at 1150–1200°C. and one at 1250–1300°C., using the soda mixture in amounts between 1·5 and 4·5 per cent by weight of the copper gave similar results averaging :		
Initial contents	0·055	0·059
1 hour after addition	0·015	0·028
2 „ „ „	0·014	0·035
3 „ „ „	0·014	0·036
(*b*) Charcoal cover and frequent poling. One experiment at 1250°C. using 2 per cent potassium carbonate :		
Initial contents	0·0060	(<0·005 per cent oxygen
5 min. after addition	<0·0002	do.
10 „ „ „	<0·0002	do.
Copper Containing About 0·01 *per cent Oxygen*		
No charcoal cover but frequent poling. One experiment at 1150–1200°C. using 1·5 per cent soda ash mixture :		
Initial contents	0·061	0·063
1 hour after addition	0·047	0·047
Copper Containing About 0·06 *per cent Oxygen*		
Oxygen content maintained by flame control. One experiment at 1250°C. using 2 per cent potassium carbonate :		
Initial contents	0·0076	(0·06 per cent oxygen)
5 min. after addition	0·0063	(0·03 per cent do.)
10 „ „ „	0·0072	(0·08 per cent do.)
Copper Containing About 0·1 *per cent Oxygen*		
Oxygen content maintained by flame control. One experiment at 1150–1200°C. using 4·5 per cent soda ash mixture :		
Initial contents	0·066	0·065
1 hour after addition	0·062	0·065
2 hours „ „	0·062	0·063
3 „ „ „	0·061	0·063
Copper Containing About 0·4 *per cent Oxygen*		
Oxygen content maintained by flame control. One experiment at 1250°C. using 2 per cent potassium carbonate :		
Initial contents	0·0041	(0·48 per cent oxygen)
5 min. after addition	0·0044	(0·35 per cent do.)
10 „ „ „	0·0042	(0·28 per cent do.)

The results of the preliminary test with additions of various elements to phosphorus deoxidized melts containing selenium are given in Table IV.

TABLE IV

MISCELLANEOUS ADDITIONS TO DEOXIDIZED SELENIUM-BEARING COPPER
(The additions listed below were made to 4-lb. melts under charcoal, containing 0·055 per cent Se and 0·012 per cent P)

Addition	Analysis			Reverse Bend Value (No. of 180° bends)	
	Se *per cent*	P *per cent*	Added Element	As Rolled	Annealed 30 min. at 650°C. and quenched
—	—	0·015	—	8	38
—	0·048	0·013	—	1	29
0·25 per cent Ca......	0·0030	0·011	0·07% Ca	4	26
0·5 per cent Ca	0·0012	0·013	0·18% Ca	2	20
1 per cent Ca-Si alloy*	0·0011	0·013	n.e.	2	22
2 per cent Ca-Si alloy*	0·0024	0·018	n.e.	½	7
0·05 per cent Li	0·030	0·013	0·011% Li	1	22
0·1 per cent Li	0·038	0·004	0·016% Li	1	15
0·6 per cent Cd	0·050	0·0095	n.e.	1	24
0·06 per cent Mg ...	0·055	0·011	0·003% Mg	½	27
0·01 per cent Be......	0·037	0·011	0·011% Be	1	12

* Containing nominally 15–20 per cent Ca, 14–18 per cent Mn, 55–60 per cent Si, 10–11 per cent Fe.

n.e. Not estimated.

Calcium, lithium, and beryllium lowered the selenium contents of the copper significantly and the effect of calcium was pursued in more detail, because it is more readily available than the other elements. The reverse bend test data shown in Table IV give no indication that these elements minimize the effect of the residual selenium on the mechanical properties of the copper and in some cases the poor mechanical properties must be attributed to the added elements.

Table V gives the results of a further experiment with calcium additions. A 20-lb. melt of cathode copper was held at 1100–1200°C. under charcoal, selenium was added, and sample ingots were poured: (*a*) before adding calcium, (*b*) immediately after the calcium addition, and (*c*) at intervals during a subsequent air blow designed

The scatter of the points plotted in Fig. 1 makes it impossible to draw any definite conclusions, but to a first approximation the relationship between the selenium contents of copper and white metal is given by :

$$\text{selenium in copper per cent} = K \times (\text{selenium in white metal per cent})^2$$

when the copper contains up to about 0·005 per cent selenium.

The attempts to remove selenium and tellurium at some stage in the fire-refining process show little promise from a practical point of view. The fact that potassium carbonate removed selenium from ' oxygen-free ' copper more efficiently than the sodium salt

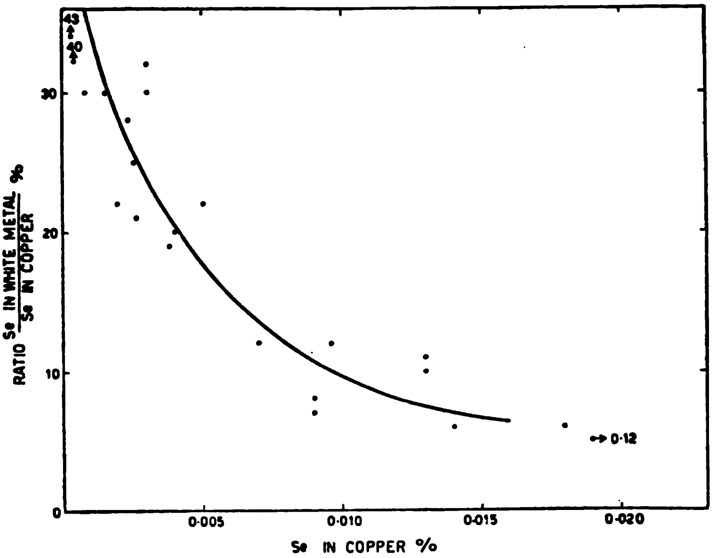

FIG. 1.—Distribution of selenium between white metal and copper varying with selenium content of copper.

is in accord with the higher heat of formation of potassium selenide. Free energy data have not been found, but it is reasonable to suppose that the entropies of the two alkali selenides would be similar and hence their free energies of formation would be in the same order as their heats of formation.

Calcium can remove selenium from copper during fire refining under certain conditions, but this use of calcium is also beset with difficulties. In the production of ' tough-pitch ' copper it would be necessary to pole the copper back to 0·01 per cent of oxygen or less, preferably in the initial stages of refining just after the blister copper had been melted. This would entail extra time and consumption of poles and, even if these considerations did not rule out the process, it is then necessary to remove the small volume of slag, rich in selenium, before proceeding with the oxidation cycle. The large-scale experiment described in Table VI shows that useful reductions in selenium content would require very careful skimming at this stage and in fact it seems unlikely that really satisfactory skimming could be done.

The use of calcium as a deoxidant and selenium remover in the production of a deoxidized copper deserves more serious consideration. The literature [7, 8] shows that calcium deoxidized coppers with controlled low-calcium contents have good working and electrical properties. The calcium and selenium contents of the third sample described in Table VI indicate that small residual calcium contents compatible with good working and electrical properties [7, 8] would probably maintain the selenium content of the melt at a sufficiently low level, but it must be admitted at once that serious practical difficulties would be encountered in attempting to maintain low calcium contents in the bath over the period required for casting. Moreover, the literature [8, discussion], shows that the calcium deoxidized bath is liable to pick up impurities from the refractory crucible or hearth which adversely affect the electrical conductivity of the copper. Thus the use of calcium as a deoxidant and selenium remover involves serious practical difficulties whose importance could only be assessed by trials on a production scale.

Of all the methods examined in this work the selective converting method offers greatest promise as a practicable method of removing selenium and tellurium. If practicable means could be found for continuously removing the copper from the converter during its formation from the white metal, the favourable distribution of these impurities between copper and white metal would be exploited to the maximum possible extent. This procedure should give substantially better separation of these impurities than the already good separation indicated by the data in Tables I and II and the amount of selenium- and tellurium-rich copper formed at the end of the blow would be lessened. The authors are not competent to express an opinion on the practicability of such a scheme, but, in view of the difficulty encountered in decanting a white metal layer from the copper, some means of tapping the copper from the lower part of the converter may well be necessary if the selective converting method is applied, in which case opportunity might be taken to tap the copper continuously as suggested above.

ACKNOWLEDGEMENTS

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THE SECOND SIR JULIUS WERNHER MEMORIAL LECTURE OF THE INSTITUTION OF MINING AND METALLURGY

on

The Effect of Impurities on the Properties of Metals

By C. H. Desch, F.R.S.

delivered at The Royal Institution, 21, Albemarle Street, W.1, on Wednesday, 6th July, 1949, at 5 p.m.

Mr. W. A. C. Newman, O.B.E., *President*, in the Chair

The President said that the lecture to be delivered that evening was the second of what the Council of the Institution of Mining and Metallurgy hoped might be a series of biennial lectures in memory of Sir Julius Wernher, that great pioneer in the South African mining industry. Two years ago the Chairman on that occasion, Mr. Taylor, gave a brief summary of the life of Sir Julius Wernher and of his influence on the industries with which he was most intimately connected, and it was therefore unnecessary to repeat those details on the present occasion. It was only fitting, however, to emphasize the great influence that Sir Julius Wernher wielded, with his associates, on the gold and diamond mining industries, had the consolidation which they effected in various ways, which and persisted even to the present day.

In addition, Julius Wernher had a great interest in education. It was in great measure due to his inspiration and to his help, both in time and in money, that the great Colonial University in South Africa was founded, and it was also well known that he gave great assistance in the establishment of the Imperial College of Science and Technology in London. It was fitting, therefore, that his name should be honoured by the Institution which was connected with mining, and which had many educational activities, by associating him with this series of lectures.

In choosing a lecturer for such an occasion, there were two ways of approach. One was to select a subject which was important and pressing, and then invite a lecturer who was an authority on that subject to address the members of the Institution. Those were the circumstances two years ago, when problems of silicosis and pneumokoniosis were uppermost in the minds of members of the Institution, and they invited Major-General Orenstein to give the Lecture. The other approach was one which the Institution could make to honour someone who had become an authority on, and done outstanding work in, one or other of the directions in which the Institution was interested. Those were the circumstances that evening, for there was no one practising metallurgy in England to-day whom the Council of the Institution felt was worthier of the honour than Dr. Desch. There was no one who was better able to discuss and review that fringe or boundary between pure metals in the refined state as they were put on the market by producers, which was a particular function of the Institution, and the use of those metals either as pure metals or in the form of alloys in their fabricated form ; and there was no one in Britain who was better able to discuss critically the influence of impurities on the products of the refineries and on the fabricated metals and alloys, and also the methods by which those impurities and their effects could be brought under control.

He thought that it was right to say that Dr. Desch graduated into metallurgy through the avenue of chemistry quite early. At the age of 28, nearly fifty years ago, he became a member of the staff of Professor Huntington, at King's College, London, at a time when the study of metals, the use of the microscope and the science of metallography, which had been associated with the names, to mention only three, of Sorby, Osmond and Roberts-Austen, were leading to the development of much new information which was soon in full flood. Dr. Desch had contributed in no small measure to the expansion of that science. Later he occupied the Chairs of Metallurgy at Glasgow and at Sheffield, and for many years he was Superintendent of the Department of Metallurgy of the National Physical Laboratory. He was now a Director of Richard Thomas & Company and Honorary Scientific Director of the Iron and Steel Research Council.

Dr. Desch had been, as was well known, President of the Faraday Society, President of the Institute of Metals, and President of the Iron and Steel Institute—a very notable record. To show his versatility, apart from his writings on metallography and metallurgy he had also written a book on the chemistry of cement and concrete, and he was, or had recently been, Chairman of the Council of Sociology.

The Council of the Institution were delighted that Dr. Desch had been able to accept their invitation to deliver the second Sir Julius Wernher Memorial Lecture, and he had in that way done great honour to the Institution.

Dr. C. H. Desch, before delivering his Lecture, thanked the President for his very kind words of introduction. It might perhaps be of interest to some of those present, he said, to know that he came into metallurgy through having been engaged in the brewing industry. It so happened that Professor Huntington wanted an assistant who was accustomed to the use of the microscope. He himself knew nothing whatever about metallurgy at the time, but had been engaged in the study of yeasts and bacteria in the brewing industry, so that he came into metallurgy by a side road. That might perhaps encourage some of the younger scientific men, who were a little doubtful about continuing along the line on which they had begun, to branch off on to some other line. Perhaps at the present time, when science had become so extremely specialized, that might be a little more difficult than it was in the days when he was young, but he thought that there was a great deal to be said for changing one's occupation if something else seemed more interesting.

Dr. Desch then delivered his Lecture :

The Effect of Impurities on the Properties of Metals

A PURE metal is one which is built up entirely of atoms of the same kind. We should perhaps qualify this by reference to the case of isotopes. A metal consisting of a mixture of isotopes of a single element, but entirely free from any other atoms, would be regarded as metallurgically pure. Important as are the differences between the several isotopes of an element for the physicist and sometimes for the chemist, the metallurgical differences are trivial and can for ordinary purposes be neglected.

Metals are, however, very sensitive to the influence of foreign atoms in their structure. If those atoms be scattered at random throughout the crystal lattice, as in a solid solution in the disordered state, their effect is to increase the hardness and to diminish the ductility, to lessen the electrical conductivity, and to produce other changes of physical properties which can be quantitatively predicted. If, however, the foreign atoms be rejected by the crystal lattice of the solvent metal, their influence on certain properties may be far greater and less predictable. A relatively small number of foreign atoms, segregating at the boundaries between crystal grains, may completely destroy the ductility of a metal. An instance of this, interesting as being perhaps the first effect of the kind to be examined by metallographic methods, is the observation in 1896 that gold, if contaminated by 0·1 per cent of bismuth, becomes so brittle that it is easily ground in a mortar. Much smaller quantities may profoundly alter physical properties.

The difficulty in removing impurities from metals varies within very wide limits. Gold and silver can be put through quite simple electrolytic processes which will leave impurities only appearing in the fourth or fifth decimal place. Mercury is very easily purified by distillation *in vacuo*. On the other hand, to prepare iron which could reasonably be regarded as pure is a task of extraordinary difficulty. Small quantities have been prepared for spectrographic purposes, in which the impurities have been reduced to an exceedingly low figure, but as powder, which can only be brought into a compact form at the cost of re-introducing appreciable quantities of impurity. The purest iron in the powdered condition has been produced in a laboratory where you would hardly expect it—the laboratory attached to the Vatican in Rome.

The best results so far obtained with the massive metal are those of Adcock at the National Physical Laboratory, elements other than iron being in such small proportion as only to be detectable spectrographically. This was obtained after elaborate precautions had been taken to prepare a pure salt of iron by chemical processes. This salt was then decomposed by heat and reduced with hydrogen, and the resulting metal had then to be melted without introducing foreign elements, the process involving the use of high vacua, of high-frequency current, and of crucibles of which the composition had to be arrived at by a preliminary research. The total quantity of iron of such purity that has been obtained is only to be reckoned in kilogrammes. That is perhaps rather an unexpected result, but very few people have ever handled such a pure iron. In view of the exceptional importance of iron for the study of metals, it is remarkable that only such small quantities have been prepared in a state comparable with that of so many other metals. Of industrially available varieties of iron, the purest is probably that prepared by the thermal decomposition of the vapour of iron carbonyl, $Fe(CO)_4$. This usually contains some carbon and oxygen, which can be eliminated in the form of carbon monoxide by heating *in vacuo*.

A volatile metal may be most easily purified by fractional distillation, and mercury is one of the most easily purifiable of all metals. After simple chemical treatment to remove baser metals, with suitable vacuum technique distilled mercury can be obtained in a condition which can be described as 'spectroscopically pure'; hence this metal is used as a standard of electrical conductivity. The standard ohm is defined in terms of the electrical resistance of a column of mercury of standard dimensions at a given temperature.

Certain other metals can be volatilized with comparative ease, such as zinc, cadmium, and, with more difficulty but with surprising success, manganese. This last metal was purified by Gayler at the National Physical Laboratory, the experimental difficulty caused by the readiness with which the metal reacts with refractory materials containing oxygen having been overcome by special vacuum technique. It is easier to prepare a volatile compound, such as a chloride or iodide, in a highly purified condition than the metal itself. The vapour of the halogen compound may then be decomposed by heat. This is the process used with so much success at Messrs. Phillips' Works at Eindhoven, and has the advantage that the metal does not come into contact during its preparation with any refractory material or other solid from which it could possibly pick up impurities. A thin wire of the metal from a previous preparation is stretched between conducting holders and heated by passing an electric current while the wire is surrounded by the vapour of the purified halogen compound. This method yields a metal quite free from oxide, but other metals which form volatile chlorides must be previously removed by chemical treatment. Niobium and tantalum, which are extremely difficult to purify by other methods, are obtained in the same way from their chlorides, at temperatures of 1800° and 2000° C. respectively.

Volatile impurities are, of course, removed in the process.

It occasionally happens that it is easier to obtain a metal in a highly purified state in the course of a commercial process than on a small scale in the laboratory. This is conspicuously true of aluminium. To refine the commercial metal—which is already purer than most industrial metals—by any of the ordinary laboratory methods is impracticable. In refining copper and several other metals, advantage is taken of the fact that the principal impurities have a greater heat of oxidation than the mother metal, so that they can be largely removed by a process of partial oxidation. This method cannot be applied to aluminium, which has an exceptionally high heat of oxidation, advantage of which is taken when aluminium is used for the thermal reduction of oxides of other metals; the so-called aluminothermic process. The preparation of highly-refined aluminium is therefore carried out by repeating the process by which the aluminium was first prepared—the electrolysis of a bath of fluorides. The aluminium to be refined is alloyed with copper to increase its specific gravity, and this molten alloy serves as an anode, lying below a bath of molten fluorides, so composed as to be lighter than the alloy but denser than the pure metal. As the metal is liberated at the cathode by electrolysis, it floats to the surface and is there collected. In practice, the salt bath is composed of a mixture of aluminium fluoride, sodium fluoride, and barium chloride. In this way, a metal is obtained by a commercial process, with less than 0·01 per cent of impurities.

The object of purifying the metal to such an extreme degree is essentially to obtain better chemical resistance. The increased softness is a property which has to be taken into account in considering its applications.

One must distinguish between metals needed in a specially pure condition for chemical purposes or for use as standards in spectrographic analysis from the same metals required for mechanical, magnetic or electrical investigations. For the former, a fine powder, prepared, let us say, by the gaseous decomposition of a highly-purified oxide or other compound, will suffice. The Vatican Laboratory has specialized in preparing metals with the minimum of impurities for use as standards in spectrographic work, but such preparations are not in the compact form required by the metallurgist.

It is in the conversion into a solid mass that the main difficulties occur. Fusion is the simplest and most obvious method, and by using vacuum fusion in a high-frequency induction furnace the entry of impurities from the furnace gases during the process is prevented. By employing crucibles of a highly refractory oxide another source of contamination is avoided, although it was found at the National Physical Laboratory that some metals will even reduce magnesia and thoria when melted in crucibles composed of them, and so become contaminated. This only occurs with metals of very high melting point. The difficulty is avoided when the metal is produced under conditions not involving contact with a refractory material; hence the great advantage of preparation from the gaseous phase, as in the decomposition of volatile halogen compounds already mentioned.

The melting or softening point of the available refractory materials sets a limit to the preparation of metals of very high melting point in a pure condition. Tungsten, for example, melting at 3382°C., cannot be melted in a crucible, but has to be prepared in the form of powder and formed into a rod by sintering at a temperature below the melting point. This may be done without special difficulty by passing a heavy current through a mass of powder shaped into the form of a rod by mechanical pressure sufficient to enable it to hold together. The melting point being so high, many of the impurities are removed from tungsten by merely heating, by passing an electric current, to a temperature near to the melting point, when most ordinary impurities are volatilized.

It would, of course, be absurd to determine the state of purity of a metal by a direct chemical estimation of the proportion of that metal in the sample, and the only practicable procedure is to determine by chemical, spectrographic, or microscopical methods the quantity of the several impurities which have been found to be present. It was proposed by Mylius in 1912 to express the degree of purity by a logarithmic quantity; this method has great advantages and should be generally adopted, grade 4, for instance, representing a metal with a total of one part of impurity in 10,000, that is 1×10^{-4}.

We may have a metal which has been very thoroughly purified from foreign metals but which has not the properties of a pure metal on account of the presence of its oxide, either dispersed through the crystal grains as fine particles or collected, as films or as chains of globules, in the boundaries of the grains. Small quantities of oxide in such a form may be difficult to detect by chemical analysis, but they can usually be seen under the microscope, provided that the technique of polishing has been of a sufficiently high standard. Electrolytic polishing—more properly described as electrolytic smoothing, since it depends on the removal of metal from high spots, and not on the flow of metal under a polishing material—provides a completely undistorted surface, and foreign particles, even of very small size, are readily detected.

So far it is not easy to apply the process of electrolytic smoothing to all metals, but work has been done with aluminium at the laboratory at Vitry, just outside Paris, where the most beautiful photographs are prepared entirely free from any distortion of the surface.

Even when the concentration of an impurity is below the limit of its solid solubility in the parent metal, some segregation may occur at the boundaries of grains, giving rise to brittleness. Leaving aside the case in which such segregation is due to imperfect equilibrium, removable by annealing, it may occur by a process comparable with that by which soap molecules, dispersed in solution throughout a mass of water, will collect in a superficial film whenever a free surface is produced, as in the blowing of a soap bubble. Such a concentration must occur if the dissolved substance lowers the surface tension of the solvent. That dissolved foreign atoms lower the surface tension of *molten* metals has been proved in a number of instances.

That a similar lowering of surface tension is produced in a solid metal is less easy to prove, as we have no really satisfactory method of determining the surface tension of a solid crystal, in contact only with similar crystals of different orientation, which is the case in a mass of solidified metal. Measurements have been made by indirect methods, and these furnish evidence that such a concentration of impurities in grain boundaries, although below the concentration corresponding with solubility in the mass, actually does occur.

In such a case, a very small concentration of foreign atoms in the intergranular boundaries may form a film which, by its higher resistance, brittleness, etc., may profoundly alter the properties of the metal in which it might be expected to have dissolved.

An interesting example may be cited from the alloys used for marine bronze propellers. These so-called bronzes are usually really complex brasses, composed of copper and zinc with smaller additions of other metals, and not true bronzes, which are alloys of copper with tin. Some of these technical alloys are solid solutions, appearing homogeneous under the microscope, and having the crystal structure known as body-centred cubic. (That is, the crystal lattice is made up of atoms so arranged that the unit of the lattice has a metal atom at each of the eight corners of a cube, and a ninth atom at the centre.) Marine propellers of such alloys, although giving satisfactory static mechanical tests, occasionally prove to be remarkably brittle under impact, so that a new ship's propeller has been known to break on striking a floating log. No brittle constituent could be detected in the grain boundaries, yet a mass of the alloy, immersed in a solution of mercurous nitrate, would in a few seconds disintegrate into a loose mass of separate crystals, like sand. The alloys which behaved in this way contained small proportions of aluminium. Examination of the ternary equilibrium diagram shows that this aluminium should have been in solid solution, but the chemical behaviour of the alloys showed that concentration in the grain boundaries had occurred.

The study of surface chemistry is thus important for an understanding of the properties of solid metals, although the surfaces considered are between two solid phases of similar composition. There is much experimental evidence to show that concentration of a dissolved element does actually occur in interfaces in a solid metal.

In speaking of interfaces within a mass of solid metal, we usually think of the surfaces separating crystals of considerably different orientation, such as are seen when a cast metal is sectioned and etched. The use of more refined methods of observation, however, shows that boundaries of another order must be taken into account. Workers with X-rays have concluded, from observations of the intensity of reflections, that most crystals—there are a few exceptions, of no importance for metallurgy—possess a 'mosaic' structure on a smaller scale than that of the crystal grains. This inference has received direct confirmation from the beautiful work of Jacquet and his colleagues in the laboratory of Vitry, near Paris. Using the method of electrolytic smoothing already mentioned and then etching under carefully controlled conditions, the sharp triangular etch-figures within a single grain of pure aluminium are found to form groups of slightly varying orientation, indicating the division of the crystal into regions much smaller than the grains normally considered. As these etch-figures are remarkably sharp, their orientation can be determined with great accuracy by means of a goniometer. The several regions into which a grain is thus found to be divided differ in orientation to the extent of 1° or so. This fine structure can occur in a pure metal, but it is no doubt affected by the presence of foreign atoms, and the etching characteristics suggest that segregation can occur at the mosaic boundaries, so that a reticulated pattern may be produced within what would be regarded, using ordinary methods of preparation, as a single grain. It will require very careful work to determine the influence of such a sub-structure on the properties of the metal, but such an influence must exist.

It is worth while to emphasize that this method of preparing a smooth and undistorted surface by electrolytic means, using a highly viscous electrolyte, makes possible the detailed study of deformation and of changes on heat-treatment, etc., when the effects of mechanical polishing are such as to destroy the evidence sought for. So far, the method has been applied with success to aluminium and its alloys and to copper, but there is no doubt that means will be found of applying it to other metals. The advantages of smoothing and etching a surface in such a way that both slip-lines and etch-figures have the appearance of having been drawn by using a drawing board and a ruler are such that valuable results may be expected from any extension of the method. If anyone is successful in applying that method to steel I shall be very glad to know of it, for I have tried for a long time without obtaining results which I regard as really first rate.

The mechanical, electrical, magnetic and chemical properties of a metal depend to an important extent on the conditions at the boundaries between crystal grains. In a pure metal there is no foreign material at those boundaries, but owing to the meeting of two lattices with different orientations the conditions must be different from those in the interior of a grain. It was at one time held by many metallographers that the region between grains was occupied by atoms in an irregular arrangement like that of an under-cooled liquid—the so-called amorphous phase, which might be as much as 100 atoms thick. Such a phase, with its resemblance to a liquid, might be expected to have a greater solvent power for foreign atoms than the adjoining crystal grains with a normal lattice. The assumption of a thick amorphous layer in a metal which has not been cold-worked is no longer accepted, and the transitional layer between two grains is now considered to be not more than three atoms thick at regions of misfit. Even so thin a layer, however, may be capable of absorbing foreign atoms more readily than a perfect lattice, and it will be convenient to assume that it does so. We would then expect to find a higher concentration of impurities, even of those which could enter into the lattice of the crystal grains, at the intergranular boundaries, and there is much evidence that that is so. We shall look for further evidence when the technique of using the electron microscope is more highly developed.

In the course of cold-working the crystal grains become distorted and, in later stages, largely broken up into fragments. Most workers on the subject to-day explain the changes produced in the metal by fragmentation, but I prefer the older hypothesis of Beilby, according to which the crystal structure is actually broken down locally, forming a disorderly ‘ vitreous ’ phase, which might be expected to have a greater power of taking foreign atoms into solution than the normal orderly space lattice. Such a phase, with its absence of planes of slip, would be expected to be harder than the crystalline phase made up of the same atoms and, on account of its looser packing, to have a lower density, as is in fact the case. The fragments into which the crystal is assumed to be broken are, however, so small according to most of the estimates, that the difference between the fragmented condition and that of a vitreous phase becomes largely artificial. Leaving aside the evidence of X-ray investigations, the hypothesis of a vitreous phase is adequate to explain the observed microscopic structures and the changes of density and hardness.

Since the plasticity and ductility of a metal depend on its capability for slipping on crystal planes, any distortion of the lattice tends to hinder slip and to reduce the plasticity. Alloys are harder than their component metals on account of the reduced liability to slip, due to the blocking of the planes of slip.

As a rule, any oxide which may have been retained in the course of the preparation of a metal is found as a separate phase, either in particles within the crystals, in which case its effect is comparatively slight, or segregated at the grain boundaries, causing mechanical weakness. Sulphides, if present, are frequently in the latter form, but sulphur is an obvious impurity which can be removed without great difficulty, and its proportion in any metal which is to be regarded as of high purity is negligible. Besides oxides and sulphides, carbides are liable to be found in metals that have been prepared by reduction with carbon or carbon monoxide, and carbide particles usually collect at the grain boundaries. Carbon in solid solution is unlikely to occur in a slowly cooled metal, as it tends to separate, either as graphite or as a metallic carbide, during the process of cooling. Phosphorus, however, if present in small quantity, is most likely to be in solid solution. These elements are not difficult to detect by chemical or spectrographic analysis, and can be eliminated by well-known methods.

Oxygen is not usually a cause of brittleness, except when present in such quantities that it can be easily detected by means of the microscope, the oxide particles sometimes collecting at grain boundaries. It is possible that oxygen is responsible for the observed brittleness of the most carefully prepared specimens of beryllium. This metal was purified by distillation in a high vacuum at the National Physical Laboratory. From its crystal structure beryllium might be expected to be malleable, but actually, although small pieces can be hammered flat at a red heat, the metal has never been found to be malleable at atmospheric temperatures. For this it is quite likely that thin intergranular films are responsible. It does not appear that the metal has been examined after electrolytic polishing, which would probably allow such films to be detected if present.

Inclusions of oxide may have only a trivial influence on the properties of a metal, as they are often isolated and present only local obstacles to deformation. Thoria is deliberately added to tungsten intended for use as lamp filaments on account of the resistance it offers to the excessive growth of crystal grains to which pure tungsten is liable if kept for long at the temperature of incandescence. Whilst, however, numerous large grains in tungsten are very undesirable—on account of the risk of producing grain boundaries extending through the whole thickness of the filament, on which displacements, involving a reduction of area and consequent local overheating, have occurred—it is remarkable that if grain growth be carried to the extreme limit, so that the whole of the wire consists of a single crystal, the mechanical properties are excellent. As the wire is drawn through a suitable temperature gradient, the large crystals, extending over the whole diameter of the wire, absorb the smaller ones, whilst the thoria, round which the large crystal is able to grow, prevents small grains from growing appreciably in size until the large crystal reaches and absorbs them. This is the basis of an important technical process.

Titanium and zirconium, which have certain metallic characteristics, can take their respective oxides into solid solution to a limited extent, lowering the electrical conductivity. Both these elements have been considered to have abnormal electrical properties, later explained as due to the presence of oxide and of nitride, which produces similar effects.

Aluminium and magnesium are metals which form exceedingly stable oxides, but both metals, when required in a highly purified condition, are refined by means of vacuum distillation, when the *non-volatile* oxides are left behind. The same is true of the alkaline earth metals. Although various methods have been used to obtain the metals of the rare earths, it does not appear that their mechanical properties are known with any degree of accuracy, research work having been directed mainly to the separation of the oxides of this difficult group rather than to that of the metals.

The changes in properties produced by the removal of the last traces of an impurity from a metal may be very striking. Reference has already been made to Adcock's work on highly purified iron. His product was softer than copper, hardness being measured by the Brinell indentation test. It needs only very small quantities of carbon or phosphorus to harden iron to an appreciable extent. An exact statement of the effects of purification on the magnetic properties is difficult, as these properties are affected by the grain size as well as by the quantity of impurities, and highly purified iron tends to form large grains. At the National Physical Laboratory we had very great difficulty indeed in preparing very pure iron in a fine-grained state. Any process to which we subjected the iron caused quite considerable grain growth.

The chemical as well as the mechanical properties of a metal also vary with the quantity of impurities present, and striking results are obtained when a metal in common use is subjected to special refining processes. Aluminium is an example. Commercial aluminium is readily attacked by many chemical agents, and its resistance to atmospheric corrosion is due to the rapid formation of a protective layer of oxide or hydroxide. Aluminium prepared by the Hoopes process, however, by which the impurities have been removed electrolytically, is chemically very inactive, being capable of remaining in dilute hydrochloric acid without being attacked. A speck of lead or iron is enough to start a local attack. This is, by the way, an excellent illustration of the view maintained by Faraday, that corrosion is an electrolytic process, needing the presence of anodic and cathodic areas on the surface, as well as that of an electrolyte. In an ordinary metal, the cathodic areas are provided by particles of foreign impurity, such as precipitations in grain boundaries. In the absence of such cathodes there is no electrolytic action and consequently no corrosion.

The same conditions apply to zinc. The zinc manufactured as pure on a commercial scale is not attacked by acids, and in attempting to dissolve it, say in hydrochloric acid, some foreign metal, such as lead, must be added in order to set up the necessary galvanic circuit. Hence, where resistance to corrosion is required, pure metals, or alloys consisting only of homogeneous solid solutions, such as certain types of stainless steel, have great advantages. Heterogeneity, to enable corrosion to occur, is not necessarily chemical. A metal which has been locally cold-worked, such as a steel ship plate with punched rivet holes, is very liable to local attack by sea water, because there is a difference of electrolytic potential between the plate in its normal condition and the region around the punched hole, which becomes the anode and is consequently attacked. A piece of iron wire, hammered at one end and immersed in the ferroxyl reagent, shows the hammered end to be the anode.

Copper is often required in a highly purified condition for electrical purposes, since all impurities diminish its conductivity to a greater or less extent. A comparative study of the effects of impurities on copper was undertaken at the National Physical Laboratory from 1920 onwards, and a monograph was published summarizing the results. Pure copper is an exceedingly soft metal, hardened by even small quantities of the usual impurities. Commercial copper, known as OFHC (oxygen-free high-conductivity) contains not more than 0·004 per cent of impurities, and rods of this metal, with a tensile strength of 13·8 tons/sq. in., have an elongation of 69 per cent on 2 in. and a reduction of area of 89 per cent. Great care is needed in the preparation of such a metal, as the elimination of oxygen and of the usual impurities are virtually incompatible processes, and special care is needed in obtaining a proper balance. In the pure condition annealed copper has little or no elastic range, and elastic properties are given to it by cold-working. However, when the electrical conductivity is not the chief determining factor, small additions (less than 1 per cent) of silver will confer elastic properties on the metal and also increase its resistance to creep at higher temperatures.

Copper from certain sources is liable to contain arsenic, antimony and bismuth, all impurities which can have a harmful effect, but that effect is greatly modified by the presence of other elements. Thus oxygen in suitable quantity renders bismuth comparatively harmless. In the study of such cases the microscope is an essential tool. As remarked earlier, the total quantity of an impurity is of small importance in comparison with its distribution, and oxygen exerts its effect largely by altering the form of the segregations. Precipitation, even of a brittle constituent, within a grain, is of trivial importance in its effect on mechanical properties in comparison with segregation at the boundaries of grains.

Aluminium, much of which is used industrially in the highest attainable state of purity, is very sensitive to the effects of foreign elements in small quantity. Careful studies by French workers have shown marked differences between two samples, estimated to contain 99·998 and 99·990 per cent of impurities respectively. (Frequently, little importance is to be attached to such figures, and it is better to enumerate the impurities actually detected, but in this instance both colorimetric and spectrographic analyses were used, and the figures given may be accepted as correct).

The temperature at which recrystallization sets in on annealing a metal which has been previously cold-worked is remarkably sensitive to the effects of impurities. For instance, silver containing as little as 0·1 per cent of copper has a recrystallization temperature 40° higher than that of pure silver. On the other hand, some additions actually lower the temperature at which recrystallization occurs, and I do not think that this effect has been adequately studied.

Aluminium of high purity (reported as 99·9986 per cent) after severe cold-working is completely softened by annealing for 6 to 10 minutes at the temperature of boiling water, and even at the freezing point of water annealing occurs spontaneously, although slowly. Even an addition of 0·01 per cent of further impurity causes a marked increase in the temperature needed to soften the metal appreciably.

Dissolved gases must be counted among the important and often troublesome impurities in metals. Oxygen, if present, is usually in the form of oxide and contributes to the non-metallic impurities, but it has an exceptional effect in the 'spitting' of silver. The oxide, Ag_2O, is soluble in the molten metal, but as the silver solidifies it is thrown out of solution and at once dissociates, liberating gaseous oxygen. With copper present in the metal, the oxygen is retained as the stable oxide Cu_2O, and no spitting occurs.

Hydrogen and carbon monoxide are both frequent causes of unsoundness in castings, being easily soluble in many molten metals but much less so in the solid state, except in a few special instances, such as that of hydrogen in palladium. Ordinarily, these gases are liberated during solidification, and as a metal usually freezes in the form of dendritic crystals growing from the sides and the base of the ingot mould, bubbles of gas are trapped between the branches of the dendrites, producing blowholes. In the laboratory, this difficulty is overcome by melting and casting *in vacuo*, and this device is sometimes used in industry, as in the production of alloys for electrical resistance wires, which must have exceptionally uniform properties throughout their length. At the Vacuum-Schmelz Gesellschaft at Hanover in Germany it was remarkably interesting to see two tons of metal being melted in a really high vacuum and the gases extracted. That was making alloys for electrical resistance wire. Where such special equipment is not available, it is best to arrange the casting conditions so that the metal begins to freeze at the bottom, the process extending upwards, so that the liberated gases can rise through the liquid without being trapped by the growing dendrites.

The rising gases, if the freezing be so controlled that they are not trapped, sweep the suspended particles of non-metallic impurities with them. With this object, the passage of a stream of chemically inert gas may be used as a means of cleaning a liquid metal before casting. Much use was made of this device in work on the light alloys at the National Physical Laboratory. Non-metallic inclusions, which have an unfavourable effect on the mechanical properties of a metal, rise only slowly when the metal is quiescent, the smallest particles taking longest to rise, as expressed in Stokes's Law. Gas bubbles rising through the metal tend to entangle the small particles in their envelope; a well-known surface tension effect; hence they rise more quickly than if the early times, and, he added, it was largely due to those facts that the early chemists had been sustained in their belief that a base metal could be enobled by transmutation. He went on to say that their prayers had been answered, but not at all in the way which they anticipated, because industrially speaking, though perhaps not scientifically speaking, metals were 'transmuted' by the presence of small amounts of impurities. Then, with prophetic vision, he added, 'Possibly we may be nearing an explanation of the causes which are at work'.

That evening Dr. Desch had given them the results of work during the intervening years which had done so much to clear up the mysteries of that time. In conclusion, he would like to say that he felt sure that, just as those present who were now in a somewhat advanced 'age-group' looked back to distant occasions when their teachers and lecturers 'knew in part and prophesied in part', but, at the very best, 'saw through a glass somewhat darkly', so the younger members present would look back on the present occasion as a memorable one, when more light had been thrown on those problems by the lecturer, who, with his unsurpassed gift of exposition, had shown them what had been accomplished during the intervening years.

It gave him great pleasure to second the vote of thanks which Mr. Stanley Robson had proposed.

The President said that the audience had already shown their enthusiasm, but he formally put the vote of thanks which had been so ably proposed and seconded.

The vote of thanks was carried by acclamation.

Dr. Desch thanked Mr. Robson and Dr. Smith for their very kind words, and the audience for the very kind reception of his address.

The proceedings then terminated.

*** *Extra copies of this Lecture may be obtained, at a cost of 1s. 6d. each, at the office of the Institution, Salisbury House, Finsbury Circus, London, E.C. 2.*

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THE INSTITUTION OF MINING AND METALLURGY

EIGHTH ORDINARY GENERAL MEETING of the 58TH SESSION held in the Rooms of the Geological Society of London, Burlington House, Piccadilly, London, W. 1, on Thursday, 16th June, 1949

Mr. W. A. C. NEWMAN, *President*, in the Chair

DISCUSSION ON

Notes on Mining Education and Postgraduate Training

By J. A. S. RITSON, O.B.E., D.S.O., M.C., *Member*

The President said that the discussion at the meeting would be slightly different from that usually resulting from the customary technical paper. Professor Ritson was to introduce a discussion on mining education and postgraduate training, a subject in which there were plenty of opportunities for varying views. There were at least four interested parties: the schools, the industry, the student, and perhaps the Institution. If a case were ultimately made out perhaps they should keep in mind what could be done to implement it, by action taken by one or more of these parties.

Professor Ritson said that he was very well aware that the notes which he had prepared were provocative. That was deliberate, because if by any chance it were possible to raise a genteel dog-fight the result might be most interesting. He had never minded a hard tackle, was quite prepared to take it, and, provided he was allowed, was willing to give it. So that, if his notes were received in the spirit in which a really good game of Rugby was played, he would have achieved his object, always bearing in mind that after a game the two sides generally foregathered and all differences were forgotten.

In his view there were three authorities concerned in the education of any engineer to-day: the school or education authority, the University, and the industry. He did not agree with the present school education because he thought a boy was made to specialize in technical or in scientific subjects far too soon, and he was quite sure that in the long run, although it added a year or two to his studies, the man who had had a broad training in the humanities as well as science would make a better engineer. He would like to see the school training altered to meet that demand. The same thing applied to the University, particularly a technological college of the type with which he was connected. He thought the course at the Royal School of Mines was lopsided. Students were, if he might say so, well trained in their particular subject, but that foundation was not broad enough. For instance, very few University graduates and not many mining engineers could write good English. Whose fault was that? It lay partly with the schools, partly with the University, but largely, perhaps, with the schools. He thought that was a great fault in general education, and if it were possible to introduce into the University training something which would overcome that one defect—inability to write clear, concise English—it would be greatly to the advantage of the future engineer.

He would have no objection to introducing into his School's curriculum a course on any of the liberal arts—not too much of it, but something to broaden the basis of education. He thought graduates would be better citizens and probably better engineers if they had more interests outside their industry. He said that because one of the principal duties or jobs of a mining engineer was human relations. Although it might be almost impossible to teach that particular subject attempts were being made to outline general principles. Had the training been broader in his younger days he was sure the engineer would take a wider view of his responsibilities and the human side would stand higher, or as high, in the qualifications of an engineer, as technical efficiency did to-day.

The University provided a foundation of basic science and applied science; beyond that it could not go. It could not teach anything like human relations except in the abstract. That had to be learned by experience and, unfortunately, was too often learned by trial and error, with disastrous results. One way of getting over that difficulty would be to allow a period of time, say two years of a young man's life, preferably immediately after leaving the University, to be spent in actual labour with manual workers. The young man learned how to perform manual tasks, which, in itself, gave him confidence when he had to control others who were carrying out similar work. There was no doubt that a mining graduate was not a mining engineer; he might think he was, but he was little more than a liability to the company who employed him for the first year or 18 months. After he had had some experience in the mines, say a couple of years, if he was any good he would become an asset. The industry must realize that when they took on a young man they had to complete his training: the University could not do it, and during that completion period, however long they chose to make it, the youngster was a liability, and an expensive liability. Young graduates expected high remuneration. Frankly, he did not think they were worth it, but they got it. In his day he paid to be taught, or at least his father paid, and he was very sorry in many respects that that apprenticeship system, in which boys were articled to a mining engineer and had to pay a considerable sum for the privilege, was no longer in existence because it meant that the young man got a really sound training in the practical side of mining. It was his view that a new graduate was not a mining engineer until his theory had been leavened by experience, and it was the duty of the industry, he submitted, to provide that experience.

Many mines had excellent training schemes on paper, but he regretted to say—from hearsay, not personal evidence, given to him by returning students—they were not always adhered to. Young men coming back on leave often told him that they had done nothing for two or three years except, for instance, measure air, collect dust-samples, or do routine sampling, and they had had little or no experience of the main job of a mining engineer, that of breaking ground. The young man to-day when he went out to his first job was usually keen and had an eye to the future when he would become a mine manager, and he thought, possibly rightly so, that if he was kept to routine jobs for two or three years he was getting nowhere: he got fed up, left, and went elsewhere. A young graduate going from England could do certain jobs adequately; he could measure air, he could take samples. He was willing to work and was put on to the staff of some senior mine official who, for example, was responsible for ventilation. The ventilation officer found the lad was becoming very useful and the tendency was to keep him there. That was quite understandable from the point of view of the official and possibly from the point of view of the company, because the young recruit was earning his keep. As a long-term policy from the company's point of view, however, he submitted that it was not good, and that was why he was pleading for a recognized scheme of training, extending over a period of, say, two years, during which the young man's jobs would be card-indexed. It should be the responsibility of some official to see that during the two years the young man carried out every one of those jobs. He suggested, therefore, that the Universities and the industry were partners in that work and that some model scheme of training should be prepared, possibly by the Institution.

As far as he was concerned, if anyone wished to criticize in any way the training which was given at the Royal School of Mines he would be extremely receptive of suggestions, and if they were within the Regulations of the College he would do his very best to adopt them. He knew he had no right to suggest to a company what training they should give but he had a fairly resilient skin and did not mind punches, and was, in fact, expecting them. He would be only too glad to hear what anyone had to say and would try to answer in due course.

Mr. Robert Annan said that the President had asked him to open the debate, and that was his sole excuse for intervening at that stage. He had already warned Professor Ritson that he did not agree entirely with all that he had said, but in his remarks he would not attempt to deal with the education of the young engineer before he entered the industry. He would confine himself to the matter contained in Professor Ritson's 'Notes', published in the *Bulletin*, dealing with training after entry to the industry.

In calling attention to the problem of postgraduate training for young mining engineers he thought Professor Ritson had done good service to the mining industry and to the profession, and there would be general agreement with much that he had had to say. As he said, it was the function of the mining school to provide a sound foundation in scientific principles. In the time available schools could not and should not attempt to do more, and training in the application of those principles to practical problems was something which should be done within the industry itself. No one with experience on the subject could deny that present conditions were unsatisfactory in many respects, and far too many young engineers were becoming discouraged in the early stages of their career. He spoke quite feelingly on that subject because it had been his experience that that did occur to a considerable extent, sometimes to the extent that the young engineer abandoned his profession entirely. The industry to-day was faced with an acute shortage of technically trained men and wastage of that kind was inexcusable from every point of view. It was, therefore, opportune to discuss the underlying causes and consider carefully the remedies which were proposed.

The principal complaint was stated quite simply, that the graduate often found himself in a routine or dead-end job from which there appeared to be very little prospect of escape. That happened far more often than it should. It could only arise from lack of proper interest on the part of the management in the well-being of the junior staff and by 'management' he included everybody from the directors downwards. The pre-occupations of the mine manager were many, and the struggle to keep down costs incessant, but it was the worst kind of economy to permit wastage of trained and valuable men, and all those concerned in the direction of mining enterprise should be alert to see that it did not happen, no matter whether it was in the small mine or the large organization. Professor Ritson was right in insisting that the functions of schools should be confined to giving the basic theoretical training. Experience, or the application of theory to practical problems, could only be found in actual operations and as this was an essential part of the equipment of a mining engineer it was up to the industry to provide the opportunities. So far the division of duties was clear, but he felt that Professor Ritson had not sufficiently emphasized the responsibilities of the student himself.

The great strength of the educational system in Great Britain had been the insistence that the student himself should take an active part in his own education and that he should be something more than a gramophone record of what he heard in the lecture room. That principle applied with equal force to the postgraduate student when he came to apply his scientific training to practical problems. Experience he must have, but before discussing the suggested means of securing it the objective should be defined a little more clearly and one should ask what was meant by experience. In the words which Tennyson put into the mouth of the aged Ulysses :

> Yet all experience is an arch wherethro'
> Gleams the untravell'd world, whose margin fades
> For ever and for ever as I move.

pointing out that experience never ended for any man whose mind was still alive. It was certainly not something which could be acquired in a short postgraduate course or in a limited term of years and then packed up in a mental toolbag for future use. Experience could be obtained in two ways, by observation and by experiment. That acquired by observation or by imitation was essentially backward looking, but nonetheless valuable on that account ; it was the skill of the craftsman, the ' know-how ' of the art or trade. Given too much emphasis it might even be an obstacle to progress, as the late Henry Ford and many others had pointed out. Traditional skill might avoid the errors of the past, but it could also kill ideas for the future and be the enemy of experience by experiment. It was obviously desirable for the mining engineer to get a large measure of traditional experience as early in his career as possible, but that was by no means the sole objective. The whole of the engineer's early training was designed so that he might bring scientific method to bear upon new problems ; it was not sufficient that he should acquire know-how from the past—his aim should always be the improvement of existing methods by all means in his power. It was that, together with the qualities of leadership, which raised him above the status of the artisan.

How was that desirable experience to be obtained ? No one would dispute that it was wrong that trained engineers should be kept indefinitely on work of a routine nature. The remedy obviously was to recruit men specially for this kind of work, men who did not need to have the same degree of technical education. On the Rand distinction was made between graduate learners and official learners. Professor Ritson suggested that the industry should provide a regular postgraduate course of instruction. Another suggestion frequently made was that the young engineer should, of his own initiative, change his job, and even the districts in which he worked, at frequent intervals in the early stages of his career. For his part he felt that a set course of instruction had great disadvantages. Really valuable experience was not to be had in work which was divorced from results. There was not a great deal to be learnt in any job when things were running smoothly, it was when things went wrong and must be put right urgently that real experience was gained.

When a graduate joined the staff of a mine he became one of a team responsible for getting results and to treat him as a student was to introduce an element of unreality and undermine his sense of responsibility. As a member of a team, in addition to his own job, he had the opportunity to observe and discuss the work of his team-mates, an opportunity which should not be missed. He particularly objected to the suggestion that the graduate should in every case be given the status of an official. Nothing could be better calculated to create a barrier between him and the men from whom he should learn. Mining was one of the few remaining occupations which gave scope for individual judgement and initiative and the skilled miner was conscious and proud of it. By tradition he was always ready to transmit his knowledge to those who were prepared to serve their apprenticeship, but he would not do so freely to anyone in the position of a detached temporary observer. While it was admittedly wrong to keep a trained man too long in a routine job, it was still more dangerous to go to the other extreme. There was a great deal of drudgery in any calling and the value of the experience was not only to be measured by the variety of the impressions received. Success depended rather on continuity of purpose and the ability to see a job through to the finish, and it was not a sound principle to suggest that as soon as the job became irksome it should be changed.

When a graduate left school he should become a member of his industry, and accept responsibility for his work to a greater degree as his capacity increased. Growth of experience was necessary, development of character still more so, and any device which retarded the acceptance of responsibility was a step in the wrong direction. The graduate should be given a responsible if small part in his industry; a set course of training was not advisable and frequent changes in employment might do more harm than good.

The solution of these problems, he believed, was entirely in their own hands. The operating staff of the mining industry was composed of mining engineers and it was for them to ensure that those who followed after should have a fair chance of development. If companies were to be persuaded to take on young men on a basis which was not strictly commercial, they should apply the necessary pressure. It had always been a tradition of the profession to hand on such skill and knowledge as they had and it only needed a reminder such as Professor Ritson had given to ensure that something was done.

In conclusion, might he say to those in control of industry that in the end they would be the chief sufferers if those opportunities for training young men were not available, and they should instruct their managers accordingly. To the profession he would say that they were in far the best position to do something and it was up to them to do it, adding a plea for a flexible method as all subjects did not respond alike. To the graduate he would say, experience was not given, it was gotten, and if at times the going seemed hard, he should remember that 'Tribulation worketh patience, patience experience, and experience hope'.

Mr. S. E. Taylor said that he spoke with some trepidation after the eloquent contribution by Mr. Annan. He thanked Professor Ritson most heartily for having brought the subject forward and particularly for the way in which he had done so. Although he had been purposely provocative he had been unable to avoid giving much sound advice.

Knowing that a number of members who would like to have expressed themselves at the meeting were unable to attend he had communicated with two who had sent him their notes. He would refer to them before giving his own observations. Apart from the general discussion on the subject he felt that actual experience and facts from various places would add materially to the discussion and he was able to give notes referring to the Kolar Gold Field and a recent experience in Canada.

On the Kolar Gold Field all new graduates started off as junior officials with full official status. There were no hard-and-fast rules and no timetables such as those outlined in Professor Ritson's 'Appendix'. A young man joining the underground department started off in the survey office, where he served for a period of six months or even twelve months, depending on circumstances and on the man himself. There he gained a knowledge of the mine plans and was soon able to find his way around—in a large and ramified mine geography was quite a lesson to be learnt. During that time he was employed on various jobs—such as sampling, air measurement, routine surveying, and drawing-office work. The main purpose of the period was to instruct him in the geography. On completion of that period he was transferred to the underground department and there he acted as assistant to an experienced official. Under the guidance of that official he gained a thorough knowledge of the practical side of mining and understood the problems connected with such important factors as ground control, stoping methods and breaking ground, development, ventilation, fire protection, and, indeed, all the sundry aspects of underground work. Most important of all, he acquired a knowledge of the handling of men.

After two or three years he became a competent underground official and was able to take over the supervision of a small section under the guidance of the chief underground agent. A young man who elected to remain in the survey department followed much the same initial training, but was brought on to carry out the more responsible survey work and was made acquainted with methods of ore reserve estimation and the general principles of running a modern survey office. The graduate entering the metallurgical department started in the assay office and then was transferred to the reduction plant where he became conversant with various methods of reduction and concentration, including gold smelting, and finally went to the cyanide plant.

In spite of those arrangements there were one or two ways in which the training was most certainly capable of improvement, both for the benefit of the industry and the individual, and it was there that the suggestions and provocative remarks of Professor Ritson were so helpful. The first point was that a new entrant into the industry should have the chance of gaining experience in branches of the industry other than his own—that is, a mining man should know something about the reduction department and the metallurgist something about the mining department. While he might not ultimately require to use that experience it certainly was of great assistance to him in understanding and appreciating the difficulties to be surmounted by his colleagues in other departments of the mine and unless the whole team worked together it was never satisfactory. At least two years of actual work in and about the mine was of the utmost importance in a young engineer's training. Some graduates had the necessary experience, but many of them had a very hazy idea of how to set about doing a job in a practical manner, and it was most noticeable that those who had had two years' experience in and around a mine, had grasped the essentials of the job and had a better understanding of the psychology of the workmen. In those days of ever-increasing importance of labour problems it was more important than ever to see that the young administrator had an understanding of them; it was so important that a course of labour management could, with advantage, be included as part of the training in the schools and Universities, even at the expense of other technical subjects.

Apart from those remarks which referred to the graduates coming from Great Britain there was also a scheme dealing with Indian graduates, and for a number of years there had been a scheme in operation which was much more definite, because there was a regular intake of so many graduates a year. They had all had much the same training and so it was possible to have a more regular system. In that system the graduates, who might or might not have taken a mining degree, were taken on as probationers for a period of two years. During the probation period they were paid a monthly salary and given quarters and accorded all the necessary privileges; for 18 months they were employed in the actual performance of manual work in all spheres of the operation. They spent time in each job, varying from one to three months; the scheme had been most successful and had resulted in training many excellent and capable officials. The actual carrying out of manual work was the most important part of the training, as most of them had never done any kind of manual work previously.

The other notes to which he would refer concerned Canada. The Canadian mining schools had the great advantage of easy access to mining areas of different types and mines of different sizes and that was partly the reason why the syllabuses tended to be more practical and less theoretical than in Britain. The present tendency was towards greater specialization in the last year of the course, which might tend to raise the standard of the theoretical as well as solving the problem of fitting an increasing number of subjects into the syllabus. In the syllabus there had to be concentration on the underlying fundamentals of the methods. The mining graduate could not have much detailed knowledge of all the operations which made up the routine of the mine, but he should have a sufficiently good theoretical background to enable him to study and understand any operation with which he came into contact. In Canada it was expected that every student should work in a mine during the summer vacation, which allowed about three months for that work. The attitude of the mines towards students had greatly changed and students were now well received and given the opportunity to become conversant with a number of jobs while at the same time earning a day's pay. He could live in the bunkhouse and save enough money to be of material assistance in meeting his University expenses. The three available summers could be spent on a base metal mine, a gold mine, a prospect or a mine in the development stage, thus providing valuable experience without any complications of official status or native labour.

His own views were that in approaching the subject it seemed there was a very wide variety of conditions and individuals to be considered and for that reason any hard-and-fast scheme was quite impossible; even a model scheme of training would have to have such a wide range of applications that it might be very difficult to draw one up which would be useful to all.

The desirability of getting practical experience both manually and in the junior supervisory positions was obvious, and the question was how it was to be done to the best advantage of all concerned. Once the general idea was more clearly and fully understood he thought the industry must work it out. It was very much the responsibility of the mine manager. However, it was not always realized how much a graduate with initiative could help himself. It depended a great deal on the individual; how he took advantage of opportunities which might come his way. There was an old saying that 'God helps those who help themselves' and if a student was keen enough both in observation and in endeavour he would find that opportunities could be found and taken advantage of. He felt sure that once it was realized that a young man was out to learn and gain experience he would find a very sympathetic attitude on the part of the management in nearly every case.

There were many cases where this period of training which was so desirable was impossible if a particular job had to be done. Sometimes a man arrived on a mine and somebody was taken ill and he just had to step right into the job and make the best of it and any question of training did not come up. That was one sort of variation of conditions which was so great that it was difficult to lay down rigid rules.

Then there was the other aspect of the question, which had been pursued with such success on the metallurgical side, and that was the institution of travelling scholarships, both for students and graduates. He did feel that that method of obtaining experience in different fields and in different types of work, given the right individual and the right keenness, could be most valuable. Personally, he would welcome a scheme for undergraduate travelling scholarships and for graduate travelling scholarships in mining, provided there was careful control by a selection committee to make sure that the men who were awarded such scholarships would take full advantage of them. The scheme could embrace not only the student and the new graduate, but also the man who had had a tour of experience of one to three years. He thought a great deal might be done along those lines to improve the opportunities given to suitable men to widen their experience.

Finally, he had felt that it was very easy to exaggerate the faults of the present system and schemes. He entirely agreed that criticisms could be made, mistakes occurred and faults could be put right, but there was in existence a very flexible and perhaps not too exact system, which worked passably well, and by and large some pretty good material was turned out. If improvements were to be made he would suggest that they should be considered with the greatest care, that sweeping modifications should not be made or wonderful schemes introduced unless they were quite sure that they really were improvements.

Dr. R. W. Revans* said that he had not prepared any remarks and all he could quote were some of the plans which the Coal Board were making to tackle the subject. He did not claim any originality for them ; he thought the bulk of the advice came in the first place from Professor Ritson, so perhaps what he was going to say had already been said by Professor Ritson in different words.

When the industry was nationalized it was clear that there was wanted at once a large number of first-class mining engineers—at least 200 a year for the next ten years. A problem of that size could not be dealt with casually ; there must be a plan. There were eleven Universities offering mining training and from them 200 men could be turned out every year fairly readily ; not all of those at present graduating remained in the country and the task was to persuade another 100 to enter the profession. The Institution of Mining Engineers drew up a scheme similar to the one put forward by Professor Ritson suggesting that the three years of experience which a graduate must spend below ground in any

* Director of Education, National Coal Board.

case should be spent in some kind of ordered way; what that ordered way should be had been taken from the collected wisdom of the industry. There had been cases where men had been working on the coal face on the Saturday shift and had gone back to manage the mine on the Monday. Although they had had experience as underground workers, what insight they had also acquired into the problems of management he did not know.

The Coal Board had taken up the scheme of the Institution of Mining Engineers and said that henceforth the 200 men whom it was desired to get into the industry should be given three years of directed practical training. Essentially, that meant that a man was going to be attached to an area general manager or a practising mining engineer and he was going to spend two of his three years in one area or even at one selected colliery, and in his third year, once he had seen the organic structure of the mine and knew the relation of one job to another, he would work in other mines and perhaps have a period abroad. He did not think there was any point in commenting on the details of the scheme; first, because he had not got them with him and could not remember them and, secondly, because not being a mining engineer he could not pretend to know the value of this, that or the other. The scheme was intended to be experimental and if it was discovered after a few years that more stress should be put on one feature of the practical experience than the other, the Coal Board would make the necessary adjustments. If there were 200 men a year in three years there would be 600 and a great deal would be learnt about the proper sequence of training; obviously if the same scheme were in force, without any modification, in ten years the Coal Board would not have deserved to learn anything.

On the question of University education he could speak perhaps with a little more authority and he thoroughly agreed with what Professor Ritson had said about liberalizing the theoretical course. He was a graduate of an American University and they had recognized that one of the crying needs of the technological course was to introduce into it some of the humanities. At California Institute of Technology, which had been responsible for one of the greatest pieces of precision engineering for all time—the 200-in. telescope—the engineering degree course lasted four years and included the equivalent of one year of humanistic or liberalistic studies. He was perfectly sure that an element of liberal study was essentially lacking from most of the mining degree courses at the moment and the professors themselves were considering what could be done to introduce it.

Mr. E. G. Lawford said that Mr. Annan had said most of what he had in his own mind, and had expressed it far better than he, the speaker, could ever have hoped to have done. Mr. Annan's contribution to the discussion was so full of wisdom and so entirely to the point that he hoped that it would be possible to print it as a separate pamphlet for circulation to boards of directors, managements and others in the mining industry.

Continuing, the speaker said that he felt that it was a mistake to prolong the period of education in the way suggested by Professor Ritson, because, in his view, it was psychologically bad for the graduate to be paid money without having to do some sort of responsible work for it. It was only when a graduate earned his first money for doing a definite job on a mine that he finally 'put away childish things' and became a man, and it was important, in his view, not to defer that stage of his development by two years of postgraduate training. At the same time, it could not be denied that the manual work outlined in Professor Ritson's schedule of postgraduate training was very necessary, and they were bound to consider how that experience of the various manual tasks in mine operation could be given to the young mining engineer. He would like to suggest that it should be possible to increase the amount of that kind of work done in vacation periods and that every minute of vacation time should be used to acquire that experience.

Even so, he felt that the available time was not sufficient, and he suggested that every young man entering a University school of mining should be required to spend one year underground immediately on leaving his public or secondary school and prior to entering the University. During that year he would be given a systematic course of manual work—i.e., timbering, track-laying, mucking, and the like. His work during subsequent vacations should be graded so that the later work embraced handling machines and, later still, air measurement, dust sampling and like matters calling for some scientific knowledge and manipulative skill.

The speaker was very much at one with Professor Ritson in desiring to see the humanities more prominent in the education of the mining engineer than they were at present. It ought not, however, to be necessary to include the humanities in the University mining courses if every young man entering mining could complete his sixth form work in classics before he touched chemistry and physics. The present-day tendency of the public and secondary schools to teach physics, chemistry, mathematics, and mechanics almost up to intermediate standard inevitably meant insufficient education in the classics and humanities. If a scheme were devised under which every young man entering mining first had to do one year underground, part of that year, say one day a week, could be spent in bringing his mathematics, physics, and chemistry up to matriculation standard. That would enable him, while at school, to remain on the classical side and would ensure a balanced education.

Exactly how that could be done would have to be worked out but he suggested that in most mining communities there would always be found either some outside teacher or a member of the staff who would be able to teach a sufficiency of chemistry, physics and mathematics to bridge the gulf between the knowledge of those subjects acquired during a predominantly classical education at school and that necessary to pass the entrance examination of University schools of mining.

To summarize, the ideal education for the mining engineer would be:

Up to age 17 *plus*: on the classical side at public or secondary school.

17 *plus* to 18 *plus*: four days a week underground on a definite training curriculum in manual tasks. One day per week elementary chemistry and physics, mathematics and mechanics up to matriculation standard.

18 *plus* to 22 *plus*: University course in mining engineering with as much vacation underground work as could be contrived.

Mr. A. R. O. Williams said that most engineers would agree that to be deemed qualified a graduate must have had some practical experience. Professor Ritson had suggested that that should be gained after graduation and amount to two years. In the speaker's view that would be, and in fact to-day was, unpopular with young graduates. On the Rand, to quote an example, graduates were required to undergo a course of practical instruction, along the lines now proposed by Professor Ritson, lasting from one to two years—the duration varied between the mining groups—before being appointed officials, and that was not proving attractive to those leaving the schools of mines in Britain at any rate. They preferred to accept appointments on other mining fields, appointments which had an official status and carried some definite measure of responsibility. After three or four years of hard work at a school of mines to gain a degree the offer of a post bearing for two years the unattractive title of 'learner' was something of an anti-climax and in his opinion understandably so.

He felt that the aim should be the production of engineers who at the time of graduation—that was the point—could claim with justification to be fully qualified by virtue of their theoretical and practical training and who therefore would be capable of filling immediately appointments of a junior, but nevertheless responsible, nature. The minimum amount of that pre-graduation practical training or experience could be assessed only arbitrarily, for, as Mr. Annan had observed, an engineer never ceased gaining experience. He believed that the aim could be achieved without altering the amount or character of the theoretical training given at the several schools of mines in Great Britain by spacing their terms, which corresponded more or less to that of the secondary schools, on the lines followed in Canada and, he believed, the U.S.A. This would lengthen the summer vacation to at least five months and allow of very much more practical training being done by students prior to graduation than now was possible. Such training could well be given in Britain and on the Continent; Canada too, only a short sea journey away, offered good opportunities.

Mr. G. Keith Allen said that it was apparent from the author's introduction of his paper that he hoped for a clash of opinion with the views he put forward, and that thereby more useful discussion would ensue. The speaker confessed at the outset that in him at least Professor Ritson would be disappointed, because he found himself in general agreement with his remarks. Nevertheless, he hoped he could contribute something useful to the subject.

Professor Ritson spoke mainly from the standpoint of the young engineer, whereas he spoke more from that of the employer. In practice there was not truly much difference, for the best conditions of service, in a wide sense, were advantageous to both—if they were not, something was wrong. No one would deny, he thought, that the industry was suffering from a shortage of technically-trained men, due largely to the war years, when the mining schools were turning out few students and metal mining, unless it happened to be a strategic metal, was in dire competition with more important wartime operations. Since the war there had been, in fact, a seller's market in respect of employment as well as of commodities, but he suggested that the time was approaching when conditions would be more alike to a buyer's market—that because the rapid increase in nationalism throughout the world was gradually limiting the opportunities abroad for mining engineers and mining schools were now running to capacity. In other words, supply might overtake demand.

He did not intend that to be a gloomy forecast of coming events, but merely suggested that it was a factor which the young graduate in applying for a position should bear in mind. He would, if he was wise, choose his job less—much less—for the commencing salary offered than for the gaining of experience and future prospects of promotion in his profession.

Promotion, however, could be too rapid, and although it might be gratifying to the individual at the time, he might find it a handicap if he left for another mine or field and found he had not had those few years of practical experience which were so necessary before he could fill satisfactorily even a mine captain's job. He fully endorsed the author's statement that the graduate should have at least five years' training before he could be considered a qualified engineer.

What was of concern to the young engineer was where he could best get the postgraduate training of the type suggested by Professor Ritson, and which was most desirable. As indicated in the paper, most large mines and mining groups had schemes of training laid down, but he was inclined to agree with the author that those schemes were not always honoured in practice, at least as fully as they should be. The reason was simple; the primary function of the manager of a mine was to operate the mine at a profit. A gold miner to-day often considered himself fortunate if he could even just make ends meet. It was only natural, therefore, that when he had an employee doing a good and profitable job he was inclined to leave him in it—it was one worry off his mind. It did not help the young engineer and that was a matter which the speaker had had in mind for some time, and he trusted that the publication of Professor's Ritson's paper in the *Bulletin* would do something towards reminding senior officials and managers of the duty the industry owed to the young engineer.

When interviewing candidates for work in the group of mines with which he was connected he frequently discussed that point with them, and his advice to them was that if they found that they were not gaining the general experience they were led to expect, they should have no hesitation in drawing the attention of the mine management to it. Generally, all that was needed was such a reminder at the right time and in the right quarter.

The author had set out in his paper an outline of postgraduate schemes covering South Africa and Northern Rhodesia and it would, perhaps, be helpful if he gave particulars of a similar scheme introduced some three years ago by the group of gold mines in the Gold Coast with which he was associated. Realizing the need, after the war, to ensure that the industry had available in the future for promotion to senior positions technically educated men of adequate practical training, two new categories of employment were introduced: (1) the technical mining assistant and (2) the technical reduction assistant. Technical mining assistants were selected from graduates who had specialized in mining. During their first three tours of service (for underground men a tour was 12 months' service on the mine) the intention was that apart from working normal shifts in the various underground sections they should serve at least six months in the survey department including ventilation and sampling, etc., and a short period perhaps in the assay office and the reduction plant. Experience in the stores and accounts department was also visualized. The commencing salary of the technical mining assistant was £45 per month, and at the end of the third tour he should find himself in receipt of a salary of £72 10s. per month and would be at the level of an assistant mine captain. From that point onwards he would be in competition for promotion with the assistant mine captain who had come up by the hard way.

The technical reduction assistant was chosen from those graduates who, during the third and fourth years of their University course, had elected to specialize in ore-dressing and metallurgy. Upon appointment those men would be required to work shifts for periods of at least six months in all sections of the reduction plant, in the assay office, on metallurgical records, and when it could be conveniently arranged would be given an opportunity of working for a period in the central metallurgical research laboratory on the coast. The commencing salary was £45 a month, and at the end of two tours (for surface employees a tour was 15 months' service on the mine) his salary should be £65 per month. He would be in line with the shift foreman for promotion to the administrative post of assistant reduction superintendent.

In neither of those categories had periods of service been specifically laid down, but the general principle was clear and every endeavour was made to carry it out in practice.

As a further encouragement to mining students (and in his remarks the word 'mining' included 'metallurgical') that year the mining industry of the Gold Coast had offered to give vacation experience to students from the Royal School of Mines and from the Camborne School of Mines. That had been made possible by the use of air travel, and twelve students would shortly be taking advantage of the offer. There had been an announcement in the Press of the scheme and he was confident it would prove a valuable and profitable venture both for the student and the mining companies.

He had welcomed the opportunity of discussing that important subject and offered his congratulations to Professor Ritson for bringing it before the Institution.

Mr. J. S. Sheppard supported Mr. Lawford's contention that the best alternative to a postgraduate scheme would be one year's training at a mine prior to entering the University. What were the pre-requisites for a successful career as a mining engineer? First, the necessary technical knowledge; secondly, the necessary personal qualities—he would not say personality! Those qualities were leadership, ability to mix with one's fellow men, and a physical make-up not just robust but with the necessary stamina which was so often required. There had been quite a change in the personal qualities of boys going from schools to the Universities. The boy going up to-day had his personal qualities less well developed than 20 years ago and the reason for that was that 20 years ago between 80 and 90 per cent of mining students were from public schools. They had lived a communal life away from home, and games had played an important part in their curriculum. At the present time a large number of boys were from grammar and secondary schools, they had lived at home and since the age of 12 had been obsessed with the examination complex; examinations had been the thing which counted and games had been very much at a discount. Their scholarship was possibly better but their physique was poorer at a time when the major mining fields were instituting very stiff physical examinations. When they got to the mining school, examinations again filled their whole horizon, and although they would be warned that they must develop their physique during the time of their entering the University and leaving if they hoped to work on some of the major fields, it was a long-term process and if they were not natural athletes they would play few games and concentrate on their examinations.

Again, the Universities were at fault because they tried to cast the mining engineer in the same mould as the research chemist, the tame scientist, and the scientific civil servant and it was not possible to do that. What was the solution ? One solution would be a much broader education similar to the education given by the older Universities to art students ; another would be an education such as was given at Sandhurst, for the attributes necessary for a good mining engineer and an engineer-officer in the Army were the same. Unfortunately mining courses had to be fitted in with other University courses and the University framework had to be adhered to. If it was decided to cut half the lectures and insist on students playing more communal games where they took hard knocks, and on a large amount of social activity, they would run into trouble, but if they said that before going to a mining school the student should first do one year of straightforward physical labour in a mine at the age of 17 or 18 it would be of considerable benefit to the boy. Unlike the arts and science students the mining graduate is exempt from military service and he could well afford to go into the mines for a year in lieu of service.

There was a large number of boys in the mining schools now who had had no contact with the mining industry. In days gone by their fathers or uncles were mining engineers, and they had probably lived in a mining camp during childhood ; there were students nowadays who had never seen a mining camp. They were not always keen on mining. He had heard that afternoon of a student who had wanted to take mathematics and although he passed the entrance examination there was not room for him and he took the mining course instead. If all students spent one year in a mine it would weed out those who did not know what they were letting themselves in for and it would improve their physique considerably. The number of failures would be reduced and the standard would be improved ; teachers would prefer to instruct men who had spent a year on a mine, and even if they were going to make metal mining their career they could work for the year in a coal mine.

Mr. E. J. Pryor said that during the past few decades the home life and social background of students coming to such institutions as the Royal School of Mines had changed. To this, for the next few years, would be added the unsettling influences of recent war. Evacuation and improvised school accommodation weakened the family discipline and high sense of moral value which characterized the British in administrative posts abroad.

In the paternalistic State schools were less concerned with developing character than with cramming for scholarships. High taxation forced parents to look toward the State for the higher education of their children. An examiner, confronted with a pile of scripts from complete strangers, adjusted the severity of his marking to fit the number of awards available. Hence, school education became a study in the art of pleasing an unknown body of people having arbitrary tastes. An entrant of high promise might be forced, against his natural trends, to scrape a pass in a subject he would then strive to forget.

That favoured the student with the photographic memory, but was hard on the type needed in mining—the young man who reasoned things out and developed a constructive attitude. Text-books and teachers could do a lot, but the valuable engineer was the man who could see the whole situation, select the vital facts, and from them decide the best line of action. Then came the test of character, which enabled him to see the plan through to success and to carry everybody with him.

At the Royal School of Mines the speaker trained a mixed class of third-year metallurgists and fourth-year miners, who at that stage were respectively technological and managerial in their outlook. Since written instructions sent from another country were much used professionally, the practical work of the course reproduced such conditions as far as possible. The instructors hung back, and left the students to puzzle out the instructions, make and correct their own mistakes, persist until reliable and controllable results appeared, and then report the whole thing succinctly as though it would be read by a busy director who had never seen the job being done.

As the student learned to handle the written word, he began to exhibit a new lucidity of thought in relation to the task. He communicated with increasing precision and self-confidence at all human levels with respect to the technical work. He knew, and could pass on that knowledge clearly, and that gave him mastery over personal frustrations and complexes of the sort that make young manhood difficult. By the end of the course the division between 'mining' and 'metallurgical' types was much less marked.

Modern mining called for a variety of specialists, held in the bond of common union—the King's English. That could be a magnificent tool, particularly with the working team scattered over (and into) a polyglot globe. Education for mining must, therefore, in his view, strengthen the common bond—the written word—until it was used with ease and lucidity. It was the only channel between head office and the distant mine and must carry its meaning precisely. The boy with a photographic memory won the scholarships, but the thinker who mastered the intention underlying what he studied was the type of student needed. Technology, like science, should foster accurate self-expression, with its objective and subjective elements held in just balance.

A graduate, however good, could be ruined by bad managerial handling. From the Board downward, the employers should remember that, and study to make the best of the raw material provided by the Universities. A graduate might lack local experience, but he had learned how to think. If he was not fairly treated, he used his powers of thought to find a more appreciative employer. A graduate had postponed independent earning for several years in order to have his mind trained for responsible supervision. If his employer put him on a par with manual workers who only knew how to do a few things by rote, he was misused. A properly trained man would rapidly master any specific local applications of fundamental principles and he was then ready for enlarged responsibilities. Unless his standing, terms of contract, and local opportunities recognized all that, a keen man felt frustrated. From that to the loss to the profession of the kind of engineer needed, was a short step. Mining employers should strive to keep promising youngsters, instead of letting them drift away.

Mining required an adventurous spirit. The engineer should be robust, sensible, cool in the presence of danger without being foolhardy, and a good leader. Managerially, he could never have all the facts before him in the same way that they were available in most other professions, and that put a premium on good judgement. At the same time, he usually kept two homes going, and should retire from his strenuous work while still young enough to enjoy a return to his own country. That justified good salaries and all the personal security it was possible for his employers to provide. Only thus could mining compete for the best talent with more sheltered professions.

Finally, he paid tribute to Professor Ritson, not only as the author of the paper, but also as a man. He judged his students as much for their manliness as for their academic ability, and his staff were not allowed to forget that both qualities were desirable in the graduating engineer.

Mr. J. A'C. Bergne said that he spoke as a rebel from the classics, who, from frustration in a classical school, had become a mining engineer. His difficulty had been not so much a dislike of the subjects at which he did not excel as an inability to overcome the inelasticity of a system which prevented him from advancing in subjects in which he could do well. Thus, he had not been particularly at one with Professor Ritson's ideas on the humanities until he had grown older, but he had now come to realize that a knowledge of the classics made a good background to an engineer's thoughts and writings.

He expressed his agreement with Messrs. Lawford and Williams on the subject of getting the essential practical work done during a student's course and not afterwards. Previous speakers had stressed the point that the programmes of engineering courses in Great Britain were modelled on existing University courses, but that, surely, was only because the dates of the terms were hallowed by centuries of tradition. Was it necessary to follow tradition blindly ? The North American system of a very long summer vacation appeared to have given North Americans a great advantage in engineering training, because it enabled a student to gain practical experience alongside his theoretical training and at the same time partly earn his keep.

From his own experience of summer vacation courses in mines, he could say that they were very often treated as a joke. One could learn quite a lot, but in his opinion an undergraduate would learn very much more if he did a paid job in a mine for six months. It would be worth while for a mine manager to engage a man for that time, but it was not worth his while to engage him for a month or two and then have him go back to his college just when he was beginning to be useful.

He was fully in agreement with the contention of earlier speakers that there were many lowly but essential jobs of which a competent mining engineer had to have practical knowledge, and they were of varying orders of intricacy. Thus, throughout a student's career through college there were always positions he could hold in mines to prepare him for the next six months' theory or to illustrate the previous six months'. But the education of a mining engineer did not stop when he had passed his finals. There came a time when the abler man reached the age of 25 to 30 and had been several years in the field. If changes were now being considered in mine courses, could not consideration be given to the possibility of organizing a refresher course at such a time as when the young engineer was applying for Associate Membership of the Institution ? In his opinion, such a course would be of great advantage to the profession. Changes in technique were very rapid to-day and after some years in the field, very often abroad and at a considerable distance from the centres of discussion and thought, the abler engineer on his return to the United Kingdom frequently found himself greatly in need of such a course.

Mr. J. B. Richardson said it would obviously be an excellent idea if the undergraduate could acquire sufficient practical training alongside his theoretical studies as was possible in such countries as Canada. Anyone trying to teach would appreciate the difficulties in imparting the principles of mining to young men who had no associations with mining and had never been in a mine.

Years ago most of the students were related to mining men and many had lived on mines ; now it was the exception for that to be so. To try and give the present-day students a true picture even with visual aid such as models, films, film strips, epidiascopes and such-like was a difficult job. Vacation experiences, as the last speaker had said, were sometimes not taken too seriously, although usually the reports produced were admirable.

In Britain, except for the Camborne School, all the mining schools, he thought, were attached to Universities and to extend the summer vacation to five months with correspondingly shorter vacations at Christmas and Easter as suggested by previous speakers was a pious hope. The Universities could not easily have different lengths of terms for one department only without seriously upsetting their domestic arrangements.

That the students should gain practical and manual experience for a year before entering a school of mines was an admirable alternative, but would it be possible to select applicants so far in advance ?

One thing that struck him about Professor Ritson's training scheme was that usually only large mines could afford such an arrangement and there were many more small mines than large ones.

Professor Ritson had pointed out that in their profession they were really ' mining money ' and had included in his schedule a short period for studying wages and accounts. Members with managerial experience would agree, he thought, that most of the young men who worked with them had an amazing lack of appreciation of the commercial and financial aspect of their job. If it were possible both in training and in curricula more time should be spent in learning how costs were built up. Many present had doubtless seen mine cost clerks sitting up all night working out elaborate monthly cost sheets built up from basic documents that were often inaccurate and incomplete and sometimes worthless.

It was a commonplace that, beyond all other kinds of engineers, the mining engineer must be always thinking of his job in terms of money and must acquire a commercial outlook before he could consider himself a successful mining executive. The bases of this knowledge should have been well inculcated before he left his University and emphasized in any training scheme carried out at a mine.

Mr. J. A'C. Bergne took exception to the first point raised by Mr. Richardson. He thought it was a matter of Mahomet going to the Mountain. On the other hand, his last remarks were very much to the point, the commercial side was most important, but it was the very part which the abler young man could best learn after he had absorbed the technical background, and when, after say five years in the field, he was ready for the postgraduate course he (the speaker) had advocated earlier.

Mr. C. I. Robinson said that he had been waiting for someone to get up who had been through the mill ; he might be wrong but he had not discerned any sign of such so far. He would like to draw attention to the fact that more than 40 years ago, at the instance of the Institution, he started on the Rand under conditions which were as nearly as possible those which had been outlined by Professor Ritson in his notes. He realized that very much depended upon the management of the mine on which the student started. He was very fortunate; he had had a manager to whom he always felt that he owed a great debt. The chief allowed him to go through practically every department of the work on the mine and he was not given any particular privilege. He started at the handsome salary of £10 a month, most of which went in the bunkhouse. His own experience was that among the departmental managers one received every encouragement, but among the underlings, of the foreman and shift-boss grade, one had a rather difficult row to hoe, because it was realized that the postgraduate student was a student. It was not so bad after he had done his drilling and got his blasting ticket. There were about seven who went to the Rand together. Some made good and became managers, others spent a great deal of time running rock drills, which was not the way to train a budding mining engineer.

He would emphasize that the Institution was doing good work more than 40 years ago, on the lines which Professor Ritson had advocated that evening.

Mr. Thomas Pryor said all members of the Institution would be grateful to Mr. Robert Annan for his outstanding contribution to the discussion.

He was unable to agree with Professor Ritson that metal mining engineers should undergo a standard course of postgraduate training in practical work. It would be a mistake for the Council of the Institution to require a standard course. Mining was very varied in its requirements, and individuals varied in their aptitudes. The detail of the work of the first two or three years after graduation should be left to individual employers and graduates to settle.

He agreed with those other speakers who had commented on the advantages of the Canadian system of one long University vacation per annum, which allowed undergraduates to employ their vacations in practical work with much greater effect than under the English system. It would not, however, be easy for Universities in Britain to alter their timetables to meet the needs of a few mining engineers. There were thus advantages in having in this country at least one school of mines like that at Camborne which was not tied to the timetable of any University. If thought fit, the course of a school like Camborne could be adapted to meet modern need in training mining engineers, without having to consider the requirements of men training for other professions.

A metal-mining engineer had to earn his living abroad, often in trying climates, and it was not easy for him to obtain a new post after the age of 55 or so. Only a small percentage of the mining engineers trained annually in England could look forward to being able to obtain remunerative professional employment in the United Kingdom in their later years. The relatively short period of *maximum* earning capacity, compared with some other professions, had to be borne in mind when considering how many years should be devoted to technical education and training.

A prime requisite for successful living was that a man should be in love with his work. During his first three years after graduation a young engineer should decide which branch of mining appealed to him most, and should then strive to devote himself to that branch. It was earnestly to be hoped that it would be found possible in future to have a few travelling scholarships available to mining men within three to five years after graduation, to assist them at that critical stage in their career.

It was the business of the schools of mines to train their students to think and to express themselves clearly; to train them how and what to observe and then to bring their reason to bear upon the observed facts, as well as to train them to know how and where to look up the information necessary to keep them abreast of current developments in technique.

The discussion had shown that it was common to-day for young men to enter schools of mines without previous knowledge of what the career of a metal-mining engineer might involve. Perusal of those two fascinating volumes of *Incidents in the life of a mining engineer*, by the late E. T. McCarthy, would show how greatly the work of a mining engineer had changed over the past 50 years. The next few decades would doubtless also show great changes. It was for the mining schools to turn out men capable of playing a constructive part in those changing conditions. The teaching staff of the schools of mines should watch each individual's particular aptitudes, so as to guide him towards that branch of the profession for which he was likely to be best fitted, while leaving it to the man himself to make his final choice in the first few years after graduation.

Given good health, a love for his chosen work, and a trained ability to observe accurately and to bring his reason to bear upon the observed facts of his experience, the young mining engineer need not fear for his future.

The President was sure it would be agreed that the discussion had been most stimulating and that Professor Ritson had introduced a subject upon which many interesting views had been expressed and from which probably some ideas for constructive work might arise in the future. No doubt Professor Ritson would like to summarize the discussion at a later date and reply at length to the main points which had been raised.

Professor Ritson said that he would prefer to reply later.

The President then proposed a vote of thanks to Professor Ritson for introducing such a comprehensive discussion. It was heartily accorded with applause.

CONTRIBUTED REMARKS

Mr. A. M. Bryan* : Professor Ritson's remarks in opening the discussion on this vital problem of education and training for the mining engineer are naturally directed towards training for the metal mining industry. But the principles of training are much the same for both coal and metal mining and, in my opinion, Professor Ritson has outlined an acceptable average course covering five years of academic and practical training. I would extend this course to six years.

Before the budding mining engineer enters the University, I would strongly advocate that he should spend not less than six months in and about a mine, under the supervision of a management which is interested in potential future officials. This period would help the young man to decide whether he is likely to find mining a congenial occupation and would enable the management to make a tentative assessment of the young man's possibilities by noting his personal qualities and behaviour, and the reaction of the workmen towards him, and vice versa. During this period he would also develop a little mining knowledge and background with early experience of working with youths of his own age and of men from many walks of life, getting to understand their different temperaments. This part of his education must be further developed, since it becomes vitally important to understand human and labour relations when one has to take a responsible post involving leadership. From his own experience he must know why and how to do any particular job in the mine and be able to judge rightly what is a fair day's work on any occupation. It will also enable him to understand the other man's point of view.

The academic study and practical work must be closely co-ordinated during the time the student is following his University course which, I suggest, should be on the 'sandwich' system and extend over four years. The student should spend his vacations doing practical work in the mine best suited to the type of work he is doing at college.

The University should aim not only at teaching the student to think and to express his thoughts in writing, but should do more to ensure that he can express his thoughts in speech. In these days of joint consultation it is essential that mining engineers should be able to speak clearly and convincingly. In this connection it should also be remembered that the responsibility of management does not stop at the mine. It overflows into the public life of the co[illegible]ity. A major problem confronting us to-day is that of d[illegible] system of education and training which will not only [illegible]e in making the individual a willing and proficient [illegible]of service to his profession, but also to the community [illegible]nd in convincing him that he is an entity whose life is [illegible]f making sense in a vastly larger cosmic scheme.

* H.M. Chief Inspector of Mines.

On leaving the University the student should follow an intensive course of not less than two years' duration of practical training for experience in both management and production problems. He should be under the close attention of an experienced person, with general supervision by a senior official. So far as practicable, this training should follow a carefully preconceived syllabus on a definite progressive course. It must be flexible enough to accommodate the young man's ability and it should ensure that he does not spend a superfluous amount of time on one particular occupation or operation. Towards the end of his course he should be allowed to show his ability and leadership by acting as a junior official, thus gradually getting the necessary training to assume greater responsibility.

If it can be arranged, the student should obtain part of this general training at more than one mine, so as to widen his experience of management, organization and administration. If he possesses the right personal qualities, by the end of six years the young man should have developed into a competent mining engineer and be qualified to take up his position as a manager. As the years pass he will gain further experience and broaden his outlook which will then be more readily related to the economics of the situation or, as Professor Ritson puts it, he will become a mining engineer who is really mining money.

In conclusion I would like to stress the point that the mining official of the future must have something else besides mere technical or academic attainments. Capacity to understand mathematical arguments does not necessarily imply the capacity to lead men. He must have in good measure those personal qualities so difficult to define but so well summed up in the words ' personality ' and ' character '. There is undoubtedly much scope for an inquiry in the mining industry into the problems of student selection and guidance.

Mr. A. Savile Davis : Only a small percentage of graduates finally attain the highest executive positions and many of them stop at the wayside in various executive positions requiring a detailed practical knowledge of some particular branch or branches of mining. Many others obviously from the outset are unsuited for any but junior executive positions and should as soon as possible be guided into specialist occupations.

A tendency exists for the practical training of graduates to be skimped, with the result that if they do attain executive rank they often fail to be first class owing to lack of practical knowledge.

Until such time as personnel management and the principles of Training Within Industry (T.W.I.) are embodied in the undergraduate training these subjects should form an important part of postgraduate training.

The length of postgraduate practical training required will vary with the ability of each person and the practical work he may have done prior to graduating, but on the Witwatersrand it should be about two years and three months, split approximately as follows :

Sample and survey	14 months
Ventilation and study	4 ,,
Native control	3 ,,
Mining (actually running stope and development end)	6 ,,

It is assumed that the graduate was trained in personnel management and T.W.I. methods, the rock-drill shop, reduction works and assay department during his University training period.

Like all other training schemes much of the benefit of training may be lost if correct control of the graduate is not exercised during his training period. It is unsound that he be kept in any department longer than his course demands, to suit the convenience of that department. Progress reports on each trainee should be submitted monthly to the officer in charge of the training, rating cards should be compiled and the trainee should be interviewed at approximately three-monthly intervals by the officer in charge—who should be one of the more senior executives.

There exists a tendency, engendered partly by the shortage of staff, to use trainees on stock jobs in a department and this may lead to other and more important items being missed.

The minimum scale, per month, on the Rand is as follows : On appointment, £30 ; after one year, £35 ; after two years, £40. It is questionable if these rates are sufficiently high.

Mr. A. Roberts* : The outline of a University course in mining given by Professor Ritson is that now generally accepted as desirable by almost all the centres of mining education. There may be slight differences in detail, but the general scheme is a sound one.

I would like to see available for all mining students more facilities of the type provided at the Mines Mechanization Centre at Sheffield for students of coal-mining. A great deal of time can be spent in attempting to describe machines, and in comparing the features of various designs, during lectures. This information could be obtained in a manner far more profitable to the student if he could handle the machines for himself, with proper instruction, under practical conditions. It is not enough to say that this information can be picked up in the mine. Unless a trainee is very fortunate, his knowledge of the very wide range of mine machinery will be limited to that used in his particular mine, which may or may not be highly mechanized. A student should enter the industry with a general knowledge of the tools used therein, and should have himself handled most of them. He should have run tests on pumps, fans, compressors, and electric motors, and have operated hoists, winches, loaders, conveyors and locomotives. He should also be familiar with air and electric drilling machines of various types,

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large and small. This knowledge could be obtained during vacations at a central mechanization training centre.

There is also an urgent need for greater opportunity to travel during vacations, not only for students, but also for lecturers. Indeed, it is more important that the lecturer should have first-hand knowledge of what is going on in the mining world than it is for the individual student. I wonder how many mining lecturers have personal knowledge of the techniques they are required to explain. Volume I of the international handbook, listing fellowships, scholarships, and educational exchange facilities, published by UNESCO, gives details of all available opportunities for trans-national study. The opportunities in the whole field of applied science are miserably few. There are none in the particular field of mining.

Mr. W. L. G. Muir : It will be generally accepted that the main objective in training a mining engineer is the proper fusion of theory and practice. Both the University and the industry can contribute to this end.

During the last two years of the University training careful consideration should be given to relating teaching to practice. It should be made clear at an early stage that mining differs from all other branches of engineering in that precise methods of calculation are inapplicable to a large sector of it. The student should be assisted to differentiate between those operations to which precise methods can be applied, those in which only statistical methods are useful, and those in which judgement, based on careful observation and experience, is the only true guide. A method used during the war to examine military problems and called, I believe, ' operational research ' is applicable to many mining operations. Failure on the part of the young graduate to appreciate the limitations of precise methods, and the field of application of other methods, has done much to accentuate the difference between theory and practice.

A further useful measure during the University training might be to carry out part of the teaching in the mine. The course on ventilation, for instance, might be rounded off by the students, under the direction of their lecturer, carrying out a complete pressure survey in a suitable mine. This is an operation which is not a regular routine feature of mining, and not likely to come within the experience of a graduate who spends only one month in the ventilation department during his practical training. It does, however, give a comprehensive exercise in the principles of ventilation and the use of instruments. The same procedure could be applied to a number of other subjects. A more extensive use of models in teaching might make a useful contribution to relating theory and practice.

When the student or graduate turns to his practical training the main problem is to obtain experience in a sufficient variety of operations to make his practical knowledge comprehensive, without either spending so much time in any one that it ceases to be profitable, or spending so little in each that his knowledge is too sketchy.

In such operations as pipe-fitting and track-laying a very short period is enough, but for stoping, development, and timbering a period of two months in each is too short for the trainee to experience more than a small proportion of all the eventualities that may arise. It appears desirable that in at least one of these operations the trainee should spend enough time to become properly experienced and confident in it. A period of six months would not be too long, the extra time being obtained rather by lengthening the training period than by reducing the time spent on other operations.

It is essential that part of the practical experience should be obtained before graduating. The value of lectures and drawing-office work is greatly enhanced if it can be related to practical knowledge already in the student's possession. It is difficult in Britain for students to obtain experience of metal mining. Might it not be useful if British metal mining schools had a third session of two terms instead of three to enable students at that stage to obtain six months' experience in Canada, where the metal mining industry is second to none in methods and equipment? Wages are sufficient for a young man to save enough in six months to pay the cost of his passage. This would also serve to overcome the difficulty mentioned by Professor Ritson of providing experience of manual work to young graduates who go at once to countries with native labour.

Mr. G. McPherson: Professor Ritson has rightly emphasized the tendency for a young mining engineer to specialize only in certain phases of modern mines organization and he has pointed out valuable lines for practical manual and technical training for after-graduation initial work. There is, however, from my experience, one outstanding phase of mining work in which the training of young men is sadly lacking. I refer to the practical knowledge of hygiene which should be possessed by any man responsible for the health and physical wellbeing of employees abroad. This seems to me to be vitally important to a young mining engineer or mining geologist: first for his own sake, for it is through unknowing errors in the first few years abroad that most men undermine their health; secondly, as regards the health of others, for usually from the beginning of his career in practice the young mining engineer abroad finds that he is in some way responsible for the sanitation provision for a group of employees and for their protection against insects and insect-borne disease; thirdly, from the companies' point of view, for it is clearly recognized nowadays that considerable financial losses are caused through unseen poor health conditions resulting in lowered efficiency of engineering staff, of embittered labour relations through irritation caused by ill-health, and also lower labour efficiency through the incidence of severe endemic disease.

For all these reasons, as a practical man, I consider it just as essential that a young mining engineer should be trained in the principles and practice of general hygiene as he should be trained for such subjects as set out in Table I.

It is too often overlooked by those responsible for the technical education of our young men that in this country they are largely drawn from sheltered circumstances in which problems of pure water supply, sanitation, and insect disease are either absent or never mentioned. Consequently when they go abroad their adjustment to a new and more barbaric environment can be a painful process, inflicting lamentable damage on their personal health and financial loss to their employers.

Mr. J. A'C. Bergne : During the debate on Professor Ritson's thought-provoking paper there was a lively interchange of views on how far college seasons could or should be changed to meet the needs of the profession. Professor Ritson also raised the question of the inclusion of some 'classical' or 'arts' subjects to leaven the engineer's background, while Mr. Sheppard dilated on the difference in character of the mining student of to-day *vis-à-vis* his counterpart of 20 years ago, who often had had a family interest in the business. Lastly, Mr. Pryor drew attention to the changing conditions in mining and the need for a freshman to make up his mind about a job suiting his temperament.

Are we not providing for a lifetime of study ? The plain fact of the matter is that there is just not the time both to educate and teach before necessity forces work upon a young man. This is very evident from the tendency to make schools take pupils as far as what was previously considered an intermediate University standard before ever entering a University. But one cannot finish off the education (including a knowledge of the humanities) at this early stage lest the adolescent has insufficient experience to grasp the finer points. Further, a loophole must be left for those who might wish to change their profession. Lastly, the teaching of a specialized subject (as distinct from a general education) should continue, in my submission, with finer points such as costing, accounting, economics, administration and the like, for selected men after some time in the field, in much the same way as the fighting services have conducted staff college courses.

Does not all this point to the desirability of spreading both the education and technical teaching of the best men over a much longer period of time than has been the practice heretofore ? I suggest that it is desirable for engineers to have at least three years' continuous training interspersed with two and a half years' field experience on a gradually increasing scale of responsibility and pay. A man should be able to leave after four and a half years with a certificate of competency, after five and a half years with first- or second-class associateships or degrees, and after a further half-year (which should be open to first-class associates from the normal course) with an honours degree, differing from the previous degree in that the man has spent the extra time in taking the course in the subjects mentioned above.

These ideas are very revolutionary to the set pattern of University life. They are put forward with diffidence, but in the belief that if changes are to take place now is the time, and with no half-measures at that.

FURTHER CONTRIBUTED REMARKS ON

Geophysics and Economic Geology*

By J. McG. Bruckshaw, Ph.D., M.Sc., D.I.C.

Mr. Oscar Weiss: Dr. Bruckshaw's paper is a useful and fair summing-up of the scope of geophysical methods in prospecting for metalliferous mineral deposits. The amount of discussion that followed the reading of the paper is a sure sign of the widespread interest taken in geophysical prospecting among members of the Institution. Those mining engineers who are up against the difficult problem of finding new ore reserves and new ore deposits fully realize the difficulties and are obviously eager to make use of any source of knowledge that could help them.

There is a shortage of ore reserves in the mining industries of the world, and as outcropping ore deposits were long ago detected and exploited we now have to search for mineralization which is hidden beneath the surface of the earth. Geology, geophysics, and geochemistry, together with drilling, are the tools of modern prospecting. Geophysics is only one of the tools and it obviously has its limitations and possibilities, which have to be fully realized and cautiously weighed up. There has been a tendency in the past to over-estimate the scope of geophysical methods and also to misuse them. There has also been a lack of understanding among mining engineers as to the proper application of geophysics. These difficulties, which were natural to a comparatively new technology, are gradually being overcome and the results derived from the collaboration between mining engineers, geologists, and geophysicists, such as we have seen recently in South Africa and to some extent in Canada, have fully justified expectations.

There are still mining engineers and mining companies who do not realize that the old times, when rich mineral deposits could easily be obtained, are over. It was not long ago that mining companies with sufficient capital simply had to sit in their headquarters and wait until some Yugoslav or other nations of little-developed countries knocked on the door with chunks of rich ore in their hands and rich mineralized areas behind them which they were only too anxious to offer for exploitation. Practically all the outcropping ore deposits are either being exploited or have already reached the declining stage. There is a shortage of mineral reserves all over the world and the far-seeing leaders of the mining industry are exploring every possibility to ensure that the world's demand for metals and minerals are met. In one way, new metallurgical and mining procedures are developed to a high pitch, which makes

* *Bull.* 508, March 1949.

possible the exploitation of low-grade ore deposits which have been neglected in the past. In another direction, systematic geological investigations of areas are coupled with geophysical exploration and drilling. There is no doubt that during the next few years all potential ore-bearing areas will be investigated in the latter fashion.

Now, neither geology nor geophysics nor drilling can produce miracles and we all know that exploration is not undertaken with the assumption that every attempt will produce a huge new mineral deposit. A single new important discovery would provide for past and future exploration expenses. It is necessary to realize that the geological, geophysical, and drilling programme in exploring a new area may take four or five years; if everything goes according to schedule, and is most successful inasmuch as a new ore deposit is discovered, even then another four or five years must elapse before the mine is ready for production. In other words, it requires ten years' planning under the ideal condition of assured success and we may say that 20 years are required in normal circumstances to produce some tangible important results. Thus the modern mining industry, at least those mining companies which wish to continue their life, are faced with the need for planning their prospecting programmes 20 years ahead. In many cases it is necessary to think 30 years ahead.

This is not an easy task, and apart from the difficulty of finding the high-grade technical staff and the highly imaginative and highly enterprising financier, a good deal of these prospecting programmes founder on the difficulties of personalities in charge of the technical operations of mining companies. This applies especially to those companies who have rich and large ore deposits and who, during their heyday, are fully occupied with their continuous expansion and with their day-to-day problems.

If one looks at the history of mining companies, one will notice that such rich and complacent companies were those which suddenly found themselves without mineral resources. It is quite understandable that the senior technicians find it difficult to attend, over and above their normal and numerous existing duties, to plan and consider 20–30 years of future exploration programmes. In most cases these personalities cannot expect to see the fulfilment of such long-range planning, and therefore the natural tendency is to let things drift in their day-to-day manner. In South Africa, on the other hand, we have proved that those mining companies which have the imagination and enterprise have been richly rewarded and there is no reason to think that in other suitable parts of the world the same could not be produced. Of course, to find such places is not an easy matter, but there are still a few corners of the earth left where large-scale mineral discoveries can reasonably be expected. In each such case it requires a long time and large sums of money to achieve success.

I would like personally to refer to Mr. McPherson's remarks concerning the results of geophysical prospecting in South Africa. Mr. McPherson is entirely wrong when he minimizes the importance of magnetic measurements in leading to the discovery of the West Witwatersrand goldfields. The success of magnetic measurements has always been admitted and praised by the Company which was responsible for the discovery—i.e., New Consolidated Gold Fields, Ltd. Mr. McPherson mentions that the particular area had its general structure revealed by ordinary geological methods and by certain drill-holes long before geophysical methods were used. He is referring, no doubt, to the efforts of Goerz and Co., who were the forerunners of Union Corporation, Ltd., and who put down several bore-holes in the West Witwatersrand area. Goerz and Co. did pioneer work and formulated some very pertinent ideas as to the possibility of the extension of the Witwatersrand System under the dolomite, from Johannesburg right up to Potchefstroom. They had imagination, they had enterprise, and they had the advice of very good geologists. Nevertheless, their effort failed, because, in the 3,000–4,000 ft. of Ventersdorp lavas and dolomites, they could not find any geological information to guide them to the sub-outcrop of the main reef horizon. The latter problem was solved by the simple magnetometer and the simplicity of the application of this method adds to the beauty and ingenuity of the exploration work which was carried out by New Consolidated Gold Fields, Ltd. The geophysical methods were, of course, used in conjunction with geology and drilling, but while several mining houses turned down the proposition for participation in the prospecting of the West Witwatersrand area, much of the confidence of the pioneers in this field was based on the striking information obtained by magnetometer surveys.

Dealing now with Mr. McPherson's other remark, concerning the Orange Free State goldfields, he admits that geophysical methods initially secured a bull, right in the centre of the target. He states that the negative side had little mention—namely, that large areas, later shown to be probably of high economic value, had been dropped on consideration of geophysical data.

Here again, I am afraid that Mr. McPherson does not appreciate the full story. At the time when Western Holdings, Ltd., and African and European Investment Company, Ltd., two pioneers in the Orange Free State, made their discovery, their prospecting was based entirely on geophysical results and on the basis of these results bore-holes were located which intersected the Basal and Leader reefs. History repeated itself—some of the rich and complacent mining companies again tried to minimize and ignore the importance of the discoveries. On the other hand, the two companies already mentioned joined by Union Corporation, Ltd., had enough vision to appreciate the possibilities disclosed by geophysical results and a few bore-holes and have reaped rich rewards.

At that stage, of course, nobody could have known the extent and continuity of the reef and its gold content, therefore the responsible managements of the mining companies very wisely acted with caution and, in fact, about 1948, when the first discovery was made, it was laid down that areas deeper than 3,000 ft. or thereabouts should be considered too deep for immediate exploration. Of course, it is quite easy to be wise after the event. We now have over 350 bore-holes which prove that the gold values can be relied upon to exist over large areas, but even now, if one compares the areas taken up by the original discoverers with those which have been turned down—not, as Mr. McPherson says, on geophysical consideration, but on consideration of depth—we can see full justification of the actions taken in 1938–39 when, in addition to the doubtful points of the area, a world war was imminent and, for some years after, the balance of the struggle appeared to be against us.

I would like now to give comparisons of depths between the areas originally taken up by African and European Investment Company, Ltd., Western Holdings, Ltd., and Union Corporation, Ltd., as against depth in the areas which in 1938–39 were considered too deep. The figures will prove that these areas are perhaps even too deep to-day; even in the light of recent events, when Orange Free State exploration passed from speculation to the stage where hard cash was needed for sinking shafts, opening up underground workings, and equipping reduction plants, the shallow areas appear very favourable as compared with the areas abandoned in 1938–39 because of their depth.

SHALLOWEST INTERSECTION OF LEADER OR BASAL REEFS

In Geophysically-Predicted Shallow Areas:

On St. Helena	1,086 ft.
On old African and European ground, now Welkom Gold Mining Company	1,788 ft.

In Geophysically-Predicted Deep Areas:

Freddies South	4,777 ft.
Freddies North	4,811 ft.
On Farm Leeuwbosch	5,826 ft. (no pay value)
On Farm Weltevreden	2,799 ft. (no pay value)

The remarkable results achieved by geophysical exploration in South Africa were not the outcome of random luck. Geophysical exploration was first carefully experimented with, then carefully planned, and, finally, carried out on a much larger scale than anywhere in the world as far as mining is concerned. The collaboration between mining engineers, geologists, and geophysicists has been most intimate and most persistent.

We continue to carry out geophysical prospecting in South Africa on a wide scale and with up-to-date methods. Recently some 40,000 square miles were investigated by the writer's aerial magnetometer unit and the same organization intends to test modern seismic techniques applied to prospecting in the Transvaal and Orange Free State. Quite recently, the discovery of the new gold-bearing areas near Stilfontein (New Pioneer Gold Mining Co.) was also the outcome of geophysical prospecting with the gravimeter.

'Time marches on!' Failures of geophysics 25 years ago are in contrast with some remarkable successes during recent years. Failures and successes will alternate. Of course, it is childish to expect startling results in every attempt. When we compare large-scale discoveries made during the past 15 years with the use of geophysics and without geophysics, the balance is much in favour of the combined use of geology and geophysics.

Dr. Bruckshaw was very modest indeed in his claims. The writer is now applying large-scale geophysical prospecting methods to searching for base-mineral deposits and the next five years will decide the balance of this effort.

AUTHORS' REPLY TO DISCUSSION ON

Diamond-Drill Blast-Hole Practice at the Roan Antelope Copper Mine*

By H. F.-C. Nevill and W. K. Burgess, *Associate Members*

Messrs. Nevill and Burgess: The authors wish to thank Mr. R. M. Peterson for introducing the paper in their absence; his introductory remarks definitely filled a gap in the paper and, we think, must have made the subject matter much clearer to those present.

The authors would also thank those who took part in the ensuing discussion which raised a number of interesting points.

Mr. A. R. O. Williams states that in the hard limestone at Gibraltar 1½-in. holes gave better fragmentation than 3-in. holes, and infers that it might be similar at Roan Antelope. The Roan ore shales do not, however, compare with the Gibraltar limestone owing to (*a*) the laminated nature of the shales; and (*b*) the presence of a large number of irregular slip planes. The laminations help the ore to break up in spite of the greater burdens used with the larger holes. The effect of movement along bedding or slip planes is diminished when using larger diameter holes, as 'semi-cut offs' can be overcome when loading. Also it has been found that when using Ex holes the first ring very often breaks back and causes the loss of a number of holes in the second ring. This does not occur when using a larger burden.

Regarding the use of Cordtex, all holes except the very short ones have detonating fuse along their full length. Nearly 3,000,000 ft. of Cordtex and Primacord have now been used without a single blasting accident attributable to its use. The only trouble encountered to date was when blasting with Primacord in a heavily-timbered reclamation block; in this case a small fire started and some time was lost in clearing the gases from this end of the mine. Mr. Williams's experience is well worth remembering, however, when charging fissured ground.

In Table IV we regret that an error was made; '80 per cent collars' should have read '20 per cent collars'. This means that on the average 20 per cent of the footage drilled remains uncharged.

The comparison between the diamond losses sustained for Ex and Ax holes (mentioned on p. 9) is worked out as a cost per foot drilled. At Roan Antelope there is no relation between the size of the hole and the cost per foot drilled.

Since writing the paper the Sullivan Sinta Set coring bit has been adopted as a standard throughout the mine for both Ex and Bx

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drilling, and it now costs less, as far as bits go, to drill a foot of Bx ($2\frac{25}{64}$-in.) than it does to drill a foot of Ex ($1\frac{1}{2}$-in.) hole.

Mr. G. F. Laycock may be interested to hear that further experiments are being carried out to do away with pulling core when drilling with a coring crown. It is still too early to say whether it is successful, but, by using a special bit and casing instead of rods, it has been possible to drill a 60-ft. up hole without pulling core. Rod grease for drilling has been inserted inside the casing when extending.

A considerable footage has been drilled with percussion machines, using extension steel equipment, at the Roan Antelope in past years.

The authors would agree with Mr. Laycock's remarks with regard to drilling through the gouge, but would add that 30–35 ft. of hole has been found to be a practical limit to extension steel drilling, while holes drilled from a set-up where the gouge is encountered generally average 50–90 ft. in length. Again, the size of the hole drilled has to be considered as pointed out in reply to Mr. Williams's query. Length of holes drilled varies considerably, of course, but a fair average would be from 30 to 50 ft. in stopes and from 60 to 100 ft. in pillars (refer to Figs. 6 and 7, Plates V and VI).

Apologies are due to Mr. J. A'C. Bergne and Dr. A. W. Groves for giving them the impression that 15s. is the present price per carat of handsetting boart. This was the price during the war, when most of our handsetting was done. During the last three years the price for this grade of boart has risen to 22s. 6d. per carat. Handsetting is now only used in special cases. Experiments have been carried out with ten different classes and sizes of stones and a grading of 8 to 15 stones to the carat of Congo drilling boart is preferred. However, the present-day supply position of boart is so poor in South Africa that the distributors are forcing the manufacturers to make up the stones as they are delivered without any sorting, so that the buyers now have no choice at all.

The percussion rock drill has not been discredited by the diamond drill in actual stoping operations. The choice of either type of machine depends upon the ground conditions.

Mr. D. H. Shute's remarks concerning the technique of manufacture of bits were very interesting. Actually we have already asked some manufacturers to make up non-coring bits with a soft centre and a hard periphery suitable for our conditions. Also, a few bits have been ordered with a cemented tungsten carbide matrix for use in particularly rough rock.

The authors regret that they have had no experience of 'reversed water flow drilling' but it would not be a practical method when drilling in stoping areas as the return water at present is frequently lost down cracks in the rock.

Professor Ritson's remark concerning our use of the term 'dynamic air pressure' is doubtless justified from the purely scientific point of view. From the literary point of view the words 'dynamic' and 'working' are synonymous according to the

Concise Oxford Dictionary. The paper is on practice at the Roan Antelope and is not intended to be a history of diamond drilling nomenclature. We feel sure that Professor Ritson must be familiar with the 'x' series of diamond drill equipment as listed in the U.S. Bureau of Standards Publication *C.S.* 17/42, since this has been an accepted standard for the last 20 years. The relevant bit sizes are all given in Table IV of our paper.

The term 'loading efficiency' is necessary to indicate that there is a direct comparison of the total footage drilled for each size rather than a mere list of the number of feet into which a case of powder may be tamped. For instance, on p. 24 it was stated that a case of powder can be tamped into 8 ft. of 3⅝-in. hole, whereas the comparative figure in Table IV is 10. The actual footages differ in the same way in the other sizes.

The authors feel that whether the theory of drilling was touched on in Part III of the paper may be left as a matter of opinion. As to Professor Ritson's comments on stemming, it is felt that they have been occasioned by an imperfect reading of the relevant paragraph on p. 24 of the paper.

Mr. J. B. Richardson states that 'Roan Antelope were the first people to apply the diamond-drill blast-hole system to a soft orebody with weak walls'. We would wish to point out that we lay no claim to being the first to do this.

With reference to his other remarks which we thought were very much to the point, the figures for the percentage of holes lost and the maximum time a hole could be left before charging were omitted purposely, as they could be very misleading. The percentage of holes lost varies according to the type of ground, the weight being experienced on the stope face (this varies according to the rate and type of stoping, and its position in the basin), and also on the location of the drilling chamber. For the whole mine, though, this figure may be taken as varying from 1 to 5 per cent. For the above reasons also the maximum time a hole can be safely left before charging varies from one week up to an indefinite period.

Further, we wish to point out that on Fig. 1—Geological sketch map of Roan Antelope and Muliashi districts—the legend reads: 'Granites, pegmatites, gneisses post- and pre-Roan—undifferentiated'. It should read: 'Granites, pegmatites, gneisses—pre-Roan intrusive into Basement Complex'. To date no evidence has been found to indicate the existence of a post-Roan granite. Between Structure Section 31 and Structure Section 41 on Fig. 1 where the 'younger' granite was assumed to have intruded the Roan sediments, underground development has shown that the absence of the R.L.7 horizon was due to the presence of a granite hill in the old Basement Complex land surface on which the Roan sediments were deposited. This absence of R.L.7 sediments was originaly interpreted as being due to magmatic stoping by a post-Roan granite.

AUTHORS' REPLY TO DISCUSSION ON

Notes on the Treatment of Pyrites Cinders at the Plant of the Pyrites Co., Inc., Wilmington, Delaware*

[By R. C. TRUMBULL, W. HARDIEK, and E. G. LAWFORD, *Member*

Messrs. Trumbull, Hardiek and Lawford: We are much indebted to Dr. S. I. Levy for his critical analysis of the reactions which we gave and we certainly accept his well-argued preference for our Equation (7) as representing the principal chloridizing reaction.

In 1937 a check on the sulphur distribution from the chloridizing furnaces was made and a nine-months' average showed that 72 per cent of the sulphur in the furnace feed reported in the discharge from the furnace, leaving 28 per cent reporting with the gases. Dr. Levy is probably right when he concludes that only 4 per cent of the total sulphur is retained in the scrubbing tower because, as will be appreciated, the purpose of these scrubbers is to remove hydrochloric acid gas.

The fact is that very little SO_2 shows up in the scrubber acid and the reason for a low absorption undoubtedly is that SO_2 is only very slightly soluble in weak hydrochloric acid at the temperatures at which these scrubbers work.

As some salt is eventually required to make a brine leach liquor to extract the silver there is, as Dr. Levy points out, always an excess in the furnace charge, and our belief is that where the sulphur is present as copper, zinc, and lead sulphide, or as mono-sulphide of iron, little SO_2 is formed. Where, however, pyrites is present, there can be little doubt that some sulphur is calcined directly to SO_2 without entering into combination with the salt.

Mr. Noakes draws attention to the use of swing hammer mills for cinder crushing. We would, however, point out that pyrites cinder is a fairly friable material and not to be compared with coke in its wearing effect upon the crusher parts. In fact, the wear on the swing-hammer mills is not at all severe for this type of operation.

Answering his other questions seriatim: Our estimate that 60–70 per cent of the non-ferrous metals were present as sulphates is based on the water-soluble copper present in the cinders as received. There is no other soluble compound present except the sulphate. However, in using the rather comprehensive term 'non-ferrous metals' we were in error and we would prefer to substitute for 'non-ferrous' the words 'copper and zinc' and we are grateful to Mr. Noakes for drawing our attention to this.

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With the type of chloridizing roast described, in which no special measures to chloridize the cobalt were taken, cobalt probably reports as to some 50 per cent in the leached cinders and 50 per cent in tail liquors leaving the system. Copper is not precipitated from its acid solutions by scrap iron.

The salt required to furnish a brine solution for silver extraction is frequently added directly to the tanks with a corresponding deduction from that added to the furnace charge. This practice was followed for a brief period at Wilmington, but was abandoned mainly on grounds of convenience.

The Wilmington specification for scrap iron has been fairly strict in two respects, in that it is customary to require that the scrap iron shall be supplied in loose bundles and not in compressed bales and that tinplate scrap shall not be present. The reason for this specification is that tin remains with the precipitated copper and some smelters impose a penalty if this element is present.

Rotary filters were used at one time, but it was found that these were costly in operation and very difficult to maintain in service. It will be realized that an acid liquor containing some copper in solution is extremely corrosive to almost all metals, while pieces of scrap iron in the slurry helped to complicate the already difficult problem of filter-cloth expense. Silver is fully precipitated from its solution by zinc, iron or copper in the conditions.

Dr. Groves raises a question which is really outside the scope of our paper and we must beg leave to be excused from offering a reply.

In answer to Mr. Gordon Duncan's question, it must be stated that there is practically nothing which the cinder-treatment plant can do regarding the sulphur content of *lump* pyrites cinders. When using lump burners calcination is invariably poor and the resulting sulphur content, of normal cupreous pyrites, is never lower than the figure desired by the chloridizing operator. Pyrites fines are usually calcined to much lower sulphur content and it is usually necessary to add green pyrites to the chloridizing charge in order to make up the deficiency. There is a long record of experience at Wilmington on well-calcined cinders and on those burned to the sulphur content desirable for chloridizing. In the former case, pyrites has to be added and our view is that there is no practical difference at all between the two conditions.

We thank Mr. Stanley Robson for his remarks ; we are very interested in the suggestion which he threw out concerning the possible reaction between salt and sulphur dioxide in the furnace with the production of hydrochloric acid and sodium sulphate.

In reply to Mr. J. Jacobi we must make it plain that the Wilmington zinc chloride/sulphate solutions were sold as such and no attempt was made to precipitate the zinc. In Germany zinc is precipitated as the hydroxide and then calcined to oxide. Mr. Jacobi is right in saying that there are difficulties ; these are mainly concerned with loss of zinc during the precipitation of iron and cobalt, both of which elements have to be removed before the zinc is thrown down.

We would remind Mr. Harry S. Lancaster that Wilmington were selling this lead in U.S.A. and that, at the time to which he refers, the American price of the metal was 7 cents per lb., or £39 per ton.

Recovery of lead by a special tapping of the iron blast furnace is, or was, occasionally practised in Europe, but this is not feasible with the modern blast furnaces. We have seen lead tapped from the iron furnaces in Europe, but only in cases where the furnaces were small.

The large number of these holes drilled renders possible a dependable decision as to the correct stoping width.

Self-potential methods of geophysical prospecting have been tried at Murchison, but the mineralization was too inert to give any positive results. The line of mineralization is not always apparent on the surface : indeed, it appears that there are probably large gaps where mineralization is extremely sparse or non-existent. The inclusion in the paper of a longitudinal section along the entire strike of the Company's claims was contemplated, but the idea was rejected, because it was felt that it would have been of little value. If Mr. Richardson will imagine a sheet of paper 13 ft. long with six patches each about 1½ in. long and varying in depth from 1 to 3 in. he will understand that most of the paper would be blank and I think he would agree with the decision ; also, this, I think, will explain why an underground haulage system is out of the question.

When drilling a hole from surface the position of the dolomitic zone is known with fair certainty from previous holes. Consequently, time is saved by drilling down to this zone with a non-coring bit. Afterwards, cores are taken to the end of the zone. No sludge samples are taken and on account of the relative shortness of the holes (average depth about 600 ft.) no deflections are made.

With regard to the method of mining, I entirely agree with everything that Dr. Keep said. All the drawbacks of the shrinkage method mentioned by him are experienced at Murchison and there is no doubt that where weak side-walls exist the method can be dangerous and wasteful. As the President pointed out, however, Murchison is really a group of small mines in which conditions vary considerably and it is probable that no one method would be suitable for the whole group. In the early days of the mine, when work was confined to shallow depths where the side-walls are normally strong and rock pressures low, the shrinkage system was probably the best, but it is evident to-day that with increasing depth some modification is called for in certain sections of the mine.

I think I was misquoted as saying that all our stopes are now worked on the cut-and-fill method. So far, only three stopes of this type have been laid out and all are giving satisfactory results with waste fill blasted from the side-walls. Pollution, the curse of shrinkage stoping, has been stopped and the danger of caving has been avoided. The most difficult problem, however—the wide stope—remains, since it is difficult to blast enough waste from the side-walls to fill it. We have not so far faced up to orthodox cut-and-fill with fill raises to surface. No cost figures are yet available to give a comparison of efficiencies between shrinkage and cut-and-fill stoping : I venture to predict that cut-and-fill will be more expensive per ton of rock, but cheaper per ton of antimony mined.

Mr. Richardson's enquiry regarding units is natural. 'Fathoms broken per case of explosives' means 'cubic fathoms broken per 50-lb. case of 50 per cent ammon dynamite'. I agree that the cubic foot and pound avoirdupois would be more logical units but the stated units are usual in South Africa for reasons which are unknown to me. It is also customary to use the short ton in mining and this is the weight denoted by the word ' ton ' in the paper, except when referring to weights of antimony concentrates or ' cobbs ', when the ' long ton ' is specifically stated in the text. The ' tonnage broken per shift per underground native employed ' means all natives employed underground, including survey, sampling, diamond-drilling, pumping, onsetting and timbering personnel, in addition to breaking, lashing and tramming gangs.

In reply to Dr. Keep, 40 per cent dynamite has been tried, but breaks too large, causing eventual trouble in the stope boxes; since the difference in price between 40 per cent and 50 per cent dynamite is very small, we have standardized on the latter for stoping. Owing to supplies of gelignite having become more plentiful, this type of explosive is now used exclusively in development.

Winzes are always sunk vertically, even if they pass outside the limits of the orebody, to facilitate the rapid hoisting of dirt. Winzing is preferred to raising for the reasons given in the text—viz. (*a*) owing to the nature of the rock, pumps are never required; (*b*) in raising, the blasting out of sprags wastes valuable European time, and (*c*) a good deal of pilot winzing is done to push development on a new level well ahead in advance of the haulage.

In reply to Mr. N. H. Monro, the tests on tungsten-carbide-tipped steel were conducted with 35 m.m. diameter bits inset direct into ⅞-in. dia. hexagon steel. The bits were discarded when they reached a gauge of 27 m.m., when binding in the hole began to occur. Gauge loss is extremely important in judging the correct time at which to regrind the bit.

All steels issued are returned to the shops for regrinding whether they have been used or not; this reduces fractures to a minimum since they are generally due to neglect in grinding. The bits are ground on a specially-designed grinder, which gives the correct shape. The average footage drilled per bit to date is 338 ft., but this is over too short a period to be reliable; it will, I believe, be considerably improved in course of time.

The President's questions regarding tungsten-carbide bits requires a lengthy answer. There is no doubt that the chisel tungsten-carbide bit is faster in any class of ground than the ordinary cruciform bit. Comparative figures at Murchison are 2·9 in./min. in hard ground, 7·9 in./min. in medium ground, and 16·75 in./min. in soft ground with tungsten-carbide, against 1·7, 6·0, and 12·95 in./min. respectively with ordinary steel. More important than penetration speed is loss of gauge per foot drilled, because this determines the life of the bit. I was engaged two years ago on rather extensive tests with tungsten-carbide bits on a large Witwatersrand mine and it became clear that a different conception of ' hardness of ground ' must be used when dealing with tungsten carbide. My conclusion then was that the success of tungsten carbide depends mainly upon the abrasiveness of the rock and that it was evident that steel and tungsten carbide have different characteristics in this respect and that a rock which causes rapid abrasion of steel will not necessarily show the same relative abrasion on tungsten carbide. Results seemed to show that this was in some way connected with the grain structure of the rock and that tungsten carbide showed its most marked superiority in very hard but fine-grained rocks. More recent work at Murchison appears to show that tungsten carbide has a wider field than was thought in softer rocks.

Regarding overall costs of tungsten carbide drilling, the cost of steel only when drilling with ordinary steel was 1·174d. per foot drilled, averaged over two years ; with tungsten-carbide jumpers, the cost of jumpers only was 2·053d. per foot drilled. The price of tungsten-carbide jumpers is 55s. 6d. per 4 ft. 6 in. jumper and 59s. 9d. per 6 ft. 9 in. jumper. The cost of the steel, however, is only part of the story. In the tests already referred to a complete study of several stopes and development ends was done for two months during which details of air pressures, drilling speeds, depth of holes, time delays, fathoms broken, explosives used, tonnage trammed, and labour employed were noted and costed out ; the cost of transporting steels and sharpening was also calculated. From all these data it was possible to determine at what price tungsten-carbide jumpers must sell in order that stoping or developing costs should be the same as with ordinary steel.

This is as far as underground observations can take us but it is by no means the end when a decision has to be reached whether to equip a mine with tungsten carbide or not. If by the introduction of tungsten carbide labour is saved, then even if mining costs are higher it may still pay to convert. As Mr. Dennison stated, by the introduction of tungsten-carbide jumpers at Murchison it has been possible to dispense with whole machine gangs, one machine in some cases doing the work which two did formerly ; in addition, each machine gang has been reduced by one boy and further labour has been saved in transport and in the sharpening shops. The labour thus saved has in actual fact been used for other work, for which no labour was available—viz. development and construction ; work which is non-productive now, but which will pay dividends in future years. But many mines to-day are milling well below their maximum capacity for the sole reason that insufficient labour is available. In such cases, labour saved by the introduction of tungsten carbide could be used for the production of additional tonnage. It is always the last few tons which yield the profit, so that the financial advantage to the company may be out of all proportion to the value of the labour saved.

In answer to Mr. Richardson and Dr. Keep regarding the reclamation of pillars, the systematic extraction of these pillars has only been started comparatively recently and so far only the upper levels which were driven in the reef have been tackled. The method is to start work at the travelling way at the extreme end of the orebody and retreat to the shaft ; downholes are drilled and the blasted rock falls down through the empty stope beneath and is recovered in the level below. The machine boys are secured by safety belts attached to the sidewalls by chains and hooks. So far only one level has been completed and the results have been surprisingly good, over 90 per cent being recovered. It is evident that in areas where the walls are very weak, nothing like this percentage extraction will be achieved if, indeed, it is possible with safety to extract anything. In some stopes, the 15-ft. pillar has collapsed before the stope is completely drawn. I agree that a cut-and-fill method would allow complete extraction if combined with foot-wall haulage layout, except in very weak ground because, unlike shrinkage stoping, it allows solid support of the hanging when approaching the top of the stope. We have every hope of achieving this in the cut-and-fill stopes now in progress.

I was very interested to receive a contribution from Mr. J. C. Mance. While doing some research into old files in connection with this paper I came across a report of his written in 1934, which gave from memory the results of his work at the Free State mine when it closed down in 1907. This report was of great value to me and I am sure Mr. Mance will be interested to know that the re-sampling recently of several old drives agreed very closely with his figures.

Regarding the Assay Plan Factor, the gold values are always erratic and the antimony values are usually so, but to a lesser degree. Gold values over 50 dwt. are reduced to the average of values on either side, an arbitrary method, which, as Dr. Keep suggests, might be too rigorous and lead to the high A.P.F. for gold. Erratic antimony values are not so reduced. I also agree with Mr. E. J. Pryor's opinion that the discrepancy between the A.P.F.s for gold and antimony is partly due to sampling technique, since high antimony values are invariably in friable ground and high gold values usually in hard ground.

There is undoubtedly considerable dilution in stoping, evidenced by the fact that a stope, when drawn, yields a higher tonnage in nearly every case than had been originally estimated. Therefore, Mr. Richardson's statement that the recovery of both metals is higher than 82·75 per cent is correct.

The effects of these three influences on the Assay Plan Factors for the two metals may be shown thus :

	Reduction of High Au Values	Faulty Sample Cutting	Dilution of Sidewalls	Net Result
Sb		—	—	—
Au	+	+	—	+

I do not think that the dip has much influence on Assay Plan Factor as suggested by Mr. Dennison. While I enter this controversial field with trepidation, I give my opinion for what it is worth that in my experience on the Witwatersrand and elsewhere the value of washings from stopes of all dips conforms in the main quite closely to the sampled value of the stope itself, except in those stopes where a rich band of reef lies contiguous to the foot-wall, a common phenomenon on the Reef. I came to the conclusion, rightly or wrongly, that the chief merit in washing a stope was to enable the senior officials to see whether the reef had been cleanly mined to the foot-wall, and I must say that in many cases the expense was justified.

In reply to Dr. S. W. Smith's queries regarding assaying procedure, the usual amount of sample taken is one assay-ton (short ton). The nitre and additional litharge are added, if required, to the stock flux and thoroughly mixed with the sample before fusion. The amount of nitre and litharge to be added depends upon visual examination of the sample by the assayer, long experience of continuous assaying of this type of ore having given him a considerable degree of skill in this respect. Even so, an error of judgement is sometimes made and then more fluxing material has to be added after fusion has commenced. No scorification is necessary.

When assaying antimony concentrates for gold, only ½ assay-ton of the sample is taken and mixed with 80 g. of the stock flux, with the addition of 5 g. silica, 20 g. nitre, and 20 g. red lead. Direct fusion follows with no scorification. No trouble is experienced with cupellation and results conform closely with checks done frequently on the Rand (where a roasting method is used) and in London. The total time required for a gold assay from the time of reception of sample to issue of result is 3 hours.

Replying to Mr. F. D. L. Noakes, the amount of fluorspar stated does not give a slag which is too thin. The price of fluorspar is approximately one quarter the price of borax.

In answer to Mr. D. G. Armstrong's questions on the ore reduction process the average millheads are 2·8 dwt. Au and 4·67 per cent Sb. He has put his finger on a weak spot in the crusher layout ; the size of the openings in the coarse ore-bin grizzly is 14 in. by 14 in. We do have trouble with occasional large rocks rolling down the incline belt and big rocks have to be taken off the picking belt and broken up by hammer because the jaw crusher is too small to take them. For the same reason, the picking belt frequently has to be stopped to allow the jaw crusher to free itself. These defects will be remedied in the new design for the enlarged mill, which will increase the capacity to 14,000 tons monthly. (The present mill was designed for a throughput of 8,000 tons and is actually doing over 11,000 tons monthly.) In the new layout, the run-of-mine ore will pass through a jaw-cracker before passing up the incline belt and a larger jaw-crusher will take the place of the present one of dimensions 16 in. by 9 in. The angle of the jaw-crusher discharge chute, which is lined with M.S. plate, is correct as stated at 32° and no trouble is experienced with choking.

No overgrinding is apparent in the mills, since they receive a heavy return from the classifiers which give an overflow of 70 per cent *minus* 200-mesh, of which 33 per cent is *minus* 350-mesh. The dimensions of the mills should have read 5 ft. by 16 ft. 6 in. and 5 ft. by 7 ft. 4 in., and of the strake tables, 3 ft. by 7 ft. 6 in. and 3 ft. by 4 ft. 6 in.

The lining of the mills is similar to that used on the Witwatersrand. Corrugated bricks are used, 20½-in. long by 8 in. wide, the thickness varying from 3 in. at the crest to 1¼ in. at the joint. Bricks are not bolted into position, but are keyed at joints with M.S. keys of various thicknesses depending on the gaps.

With regard to talcy gangue, which Mr. F. T. C. Doughty also queried, it is preferred that this material should float in the gold section, otherwise it comes up on the antimony section, reducing the grade of the concentrate. The antimony content of the talcy gangue concentrate is about 2 per cent Sb.

The Cambridge electrical pyrometers are bricked in above the flue of the roaster. It is agreed that it is the temperature of the ore on the hearth which matters, but, since the roaster is closed in, the temperature of the atmosphere inside the roaster forms a reliable guide to operating conditions. Calcine from the roaster causes very little dusting in its passage to the storage bin.

About 60 per cent of the gold in the calcine is recovered in the first treatment and 10 per cent in the second. No trouble is experienced with zinc provided it is thoroughly washed and dipped into lead nitrate solution.

With regard to the tabling of antimony concentrates, we have no difficulty with frothing on the table. Concentrate is watered down to 15 : 1 liquid/solid ratio in the cell launders before being pumped to the tables and frothing only occurs if the spray water on the tables is for any reason stopped.

In reply to Mr. Doughty, nickel sulphide does go through to the antimony section to some extent—the concentrates as shipped containing roughly 0·07 per cent Ni and 12 dwt. Au per long ton.

In reply to Mr. E. J. Pryor and Mr. Doughty, the actual gold recovery based on millheads is approximately 54 per cent, the losses being flotation tails 17 per cent, calcine 3 per cent, and gold in antimony concentrates 26 per cent. The smalters pay for part of the gold in the concentrates, so that this is not all a financial loss. This is not a good recovery, but the problem is to find the best compromise between the optimum gold recovery and the optimum antimony recovery. Before antimony was produced for the market, the actual recovery of gold was about 80 per cent. In the early days of antimony sales, the East mill was turned over to the treatment of antimony and the West mill was retained for the production of gold only. Later both mills were turned over to treat antimony. This explains why the flow-sheets of the two mills are different to-day, but there is practically no difference in the efficiency of the two units nor in the quality of concentrates produced. The enlargement to the mill will replace the present West mill and will be very similar to the present East mill. I hope this answers Professor C. W. Dannatt's question.

I thank Mr. J. A'C. Bergne for his remarks in connection with the power plant. He has certainly raised a subject which has been a source of anxiety to successive managers of this property for years past. I can only say that even with indifferent equipment, it is surprising what can be done with a devoted and competent engineering staff. Since the paper was written we have installed a new 1,000 kW generator driven by a 1,440-h.p. General Motors engine which is the last word in diesel design. We are now looking forward to a comparatively trouble-free period in regard to power.

It is true that engineering staff account for practically half our total executive, but the indifference of the plant is not the chief reason. We are at present engaged on a large programme of construction and the numbers given in the paper include masons, carpenters, plumbers, etc., engaged on this work, in addition to winding-engine drivers and a fair sprinkling of apprentices.

In reply to the President's queries regarding water supply, the total head from the pumps to the tanks on the top of Maid of Athens Kop is 481 ft. The head from the river to the pumps varies according to the height of the river, but is 15 ft. in normal weather. Recently in order to increase the water supply to the mine for the enlarged mill and also to supply water to the newly-developed Gravelotte and Jack West mines, a reservoir and 75 h.p. pump have been installed at a point on the pipe line 325 ft. below Athens Kop. The main pumps can now either pump direct to the mine as before or can deliver to the new reservoir, which is 80 ft. in diameter and 6 ft. deep, the water being pumped thence to the mine by the new transfer pump which can also deliver through a 3-in. main to Gravelotte.

The filter is placed on top of the mine ridge, near the central power station. The water from Athens Kop reservoir flows directly into it under natural head. Water for the mill may be tapped from the pipe line *en route*. As stated in the paper all domestic water is now filtered ; water for boilers and engine jackets may be taken filtered or unfiltered as desired.

BULLETIN OF THE INSTITUTION OF MINING AND METALLURGY

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No. 514 **SEPTEMBER, 1949**

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INSTITUTION NOTES

The Nuffield Foundation: Fellowships and Scholarships for the Advancement of Extraction Metallurgy

During the year 1950 the Nuffield Foundation is again offering a limited number of fellowships and scholarships with the object of advancing research and training in extraction metallurgy. Citizens of Great Britain, the Commonwealth, and Empire are eligible to apply.

Travelling Fellowships are open to men who are members of the teaching staff of Universities and approved Schools of Mines and Metallurgy.

Travelling Postgraduate Scholarships are offered to junior members of the profession who are graduates of Universities and approved Schools of Mines and Metallurgy.

Vacation Scholarships are offered to mining and metallurgical students at Universities and approved Schools of Mines and Metallurgy.

Applications for fellowships and scholarships to be taken up in 1950 must be received before 1st November, 1949, by the Secretary, Nuffield Foundation, 12 and 13, Mecklenburgh Square, London, W.C. 1, from whom full particulars and application forms can also be obtained.

Beilby Memorial Awards

Consideration is to be given to the making of an award or awards from the Sir George Beilby Memorial Fund early in 1950, and the administrators (representing The Royal Institute of Chemistry, The Society of Chemical Industry, and The Institute of Metals) will welcome information before 31st December, 1949, to assist them in their selection.

Awards are made to British investigators in science to mark appreciation of records of distinguished work. Preference is given to investigations relating to the special interests of Sir George Beilby, including problems connected with fuel economy, chemical engineering and metallurgy, and awards are made not on the result of any competition but in recognition of continuous work of exceptional merit, bearing evidence of distinct advancement in science and practice. In general, awards are not applicable to workers of established repute, but are granted as an encouragement to younger men who have done original independent work of exceptional merit over a period of years.

In recent years the amount of each award has commonly been 100 guineas.

Communications drawing the attention of the administrators to outstanding work of the nature indicated should be addressed to the Convenor, Sir George Beilby Memorial Fund, Royal Institute of Chemistry, 30, Russell Square, London, W.C. 1.

The Scientific Instrument Manufacturers Association: Electronics Symposium, 1949

The Scientific Instrument Manufacturers Association has decided to hold an Electronics Symposium each year, and the 1949 meetings will be held at the Examination Hall, Queen Square, London, W.C. 1, from Wednesday to Friday, 2nd to 4th November. A series of technical papers will be read and briefly discussed, and the latest types of British scientific and electronic instruments will be demonstrated. Full details of the exhibits and résumés of the papers to be read are obtainable from the Secretary of the Scientific Instrument Manufacturers Association of Great Britain, Ltd., 17, Princes Gate, London, S.W. 7. Admission to the Symposium will be by ticket.

Election of Members of Council for the Session 1950-51

As previously announced, the new Bye-laws governing the constitution of the Council and its mode of election will affect nominations for the election of Council for the session 1950-51. Nominations for the election of Ordinary Members of Council [*see* Bye-law 27 (iv)] and Overseas Members of Council [*see* Bye-law 28 (iii)] should be sent to the Secretary of the Institution to reach him not later than 1st November, 1949.

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Mr. S. HIGHAM, *Associate Member*, has returned to England from Bulawayo on leave.

Mr. A. W. HOOKE, *Member*, has returned to England from the Gold Coast.

Mr. R. C. HOWARD-GOLDSMITH, *Associate Member*, has joined the staff of the Cerro de Pasco Copper Corporation, Peru.

Mr. H. E. JEFFERY, *Associate Member*, has returned to England from the Kolar Gold Field to join the staff of Messrs. John Taylor & Sons.

Mr. R. B. JOHNSTON, *Student*, has taken up the position of mining engineer to Straits Consolidated Tin Mines, Ltd., Kroh, Upper Perak.

Mr. K. J. KENNEDY, *Student*, has left England for the Transvaal.

Mr. J. N. L. MONRO, *Student*, has left England for Nigeria.

Mr. G. E. OSBORNE, *Student*, is returning to England from Malaya on leave.

Mr. W. J. PALK, *Associate Member*, is returning to England from Sierra Leone.

Mr. V. A. PHILLIPS, *Student*, has left England and is now in the United States.

Mr. H. R. POTTS, *Member*, has left England on a professional visit of several months' duration to Queensland, Australia.

Mr. C. REYNOLDS, *Associate Member*, has left England to take up a post in Uganda.

Mr. A. J. ROBERTS, *Student*, has returned to England from the Gold Coast.

Mr. J. E. ROBSON, *Associate Member*, has returned to the Gold Coast after leave in England.

Dr. G. A. SCHNELLMANN, *Member*, has been appointed mines manager of the Millom & Askam Hematite Iron Co., Ltd.

Mr. R. P. SHEPPARD, *Student*, has returned to India after leave in England.

Mr. D. M. SHERIDAN, *Associate Member*, in partnership with Mr. TOM EDWARDS, *Associate Member*, is engaged in developing mines in Southern Rhodesia.

Mr. G. M. STOCKLEY, *Member*, has returned to Tanganyika after furlough in England.

Mr. R. TEALE, *Student*, is returning to England from Northern Rhodesia.

Mr. L. VAUGHAN, *Member*, is returning to England from Malaya, having finished his tour as Tin Adviser to the War Claims Commission.

Mr. P. L. VAUGHAN, *Student*, is leaving England for the Transvaal.

Mr. J. WEEKLEY, *Member*, has returned to Malaya after leave in England.

Mr. D. L. WHITE, *Associate Member*, has returned to England on leave from India.

Mr. J. C. WILSON, *Student*, is leaving to take up employment with Nanwa Gold Mines, Ltd., Gold Coast.

Addresses Wanted

D. S. Broadhurst.
J. A. Cocking.
E. Dickson.
R. B. Hicks.
G. C. Morgan.
A. I. Scott.
A. Sloss.

OBITUARY

Lieut.-Com. **Robert Claude Gibbs**, R.N., died on 10th June, 1949, at the age of 54. He received an engineering training at the Royal Naval Colleges of Osborne and Dartmouth, and served at sea from January, 1913, and throughout the war in the Royal Navy as midshipman, sub-lieutenant and lieutenant. In 1920 he left the Navy and was appointed engineer to Minerals Separation, Ltd., on testing and research work, and after three years went to Germany as engineer and later chief engineer to Central Europäische Schwimm-Aufbereitungs A.G., Berlin, a company formed to introduce the Minerals Separation flotation process into Central Europe. He was there for nearly seven years, and returned in 1929 as engineer to the Dorr Co., London. During the 1939–1945 war he rejoined the Royal Navy, retiring with the rank of lieutenant-commander to resume his work with the Dorr Co., which continued until his death. Lieut.-Com. Gibbs was elected to Associateship of the Institution in 1931.

Geoffrey Musgrave, *O.B.E.*, died in Selukwe, Southern Rhodesia, on 13th July, 1949, at the age of 67. He was born in Bradford, Yorkshire, and was trained as a mechanical engineer, serving his apprenticeship with Messrs. Smith Bros. and Eastwood Valley Iron Works. From 1902 to 1906 he obtained practical engineering experience in England and Wales. In 1906 he went to Norway on his appointment as engineer and assistant and acting mine manager to Traag Mines, Ltd., and after eighteen months he joined Telemarken Copper Mining and Smelting Co., Ltd., at Aamdal. He left Norway in 1908 to take up the position of assistant mine manager to Altai Gold Concessions, Ltd., in Siberia, returning to England in 1909 for a few months' work in London and Devonshire. In the same year he was appointed assistant engineer with Linchwe Concessions, Ltd., Bechuanaland, and in 1910 took over the management of the mines at Selukwe, Southern Rhodesia, for Rhodesia Chrome Mines, Ltd. In 1913 Mr. Musgrave inspected chrome deposits in Baluchistan, and then spent three months in practice at Selukwe as mechanical and mining engineer, while being retained by Rhodesia Chrome Mines, Ltd., as consulting engineer.

During the 1914–1918 war he served on the advisory side of the Ministry of Munitions in India, Burma, China, Japan, the U.S.A., and Canada, and returned to Selukwe in 1917 to continue his consulting work for Rhodesia Chrome Mines, Ltd., and Rhodesia Metals Syndicate, Ltd., his position with the former company being retained until his death.

Mr. Musgrave was the first chairman appointed to the Rhodesia Iron and Steel Commission, in which capacity he was responsible for setting up the steel works at Que Que, and he was also chairman of the Industrial Development Commission. He was a delegate to the Eastern Group Conference at New Delhi in 1940, and acted as assistant in Australia during the negotiation of a trade agreement with the Australian Government. He was awarded the O.B.E. in 1941. Until 1947 he was chairman of the Rhodesian National Industrial Council of the Mining Industry. He was elected a Member of the Institution in 1924, and was Member of Council for Rhodesia from 1946.

BOOK REVIEWS

Géologie des gites minéraux. 2nd ed. By E. Raguin. 641 p., 145 figs. Paper covers. Paris: Masson et Cie, 1949. 1,650 fr.

This edition follows the same general plan as its predecessor, published in 1940, whilst wisely omitting unimpressive short chapters on geological maps, geophysics, and ore microscopy. The first 150 pages deal with a slightly amended classification of mineral deposits, the circulation of underground waters, and with various types of orebodies of igneous, hydrothermal, and sedimentary origin. The remainder of the volume discusses the metals and other economic materials seriatim, their uses, production statistics (up to 1945), and geological occurrence. Little new is added to the descriptions of individual mining fields, with the notable exceptions of the Moroccan lead deposits and the Bolivian tin lodes, and the accounts of some districts, such as Rio Tinto and Sudbury, are still out of date. This volume is only moderately successful in its attempt to cover the origin and nature of mineral deposits, and on the whole it compares unfavourably with analogous works by Lindgren and Bateman.

David Williams.

Cobalt. By Roland S. Young. (A.C.S. Monograph No. 108.) New York: Reinhold Publishing Corporation, 1948. 181 p., illus. $5.00.

The work under review is one of a series of useful scientific and technological monographs published under the auspices of the American Chemical Society. Having regard to the rapid expansion in new knowledge the policy followed in their production has been to ensure complete and critical treatment of relatively restricted fields. These monographs, therefore, serve two purposes; to make available a thorough treatment of selected areas for the use of those working in wider fields and at the same time to stimulate further research in the specific field selected. To achieve these ends the authors are asked to give extended references to the literature of their own particular subject.

In this compilation of the present position with regard to the metal cobalt, these objects have been fully attained to the date of publication (1948), although the close attention now being given to the metallurgy and uses of this element is constantly bringing forward new information, such, for instance, as the recent (1949) contribution by P. S. Bryant of Murex Ltd. to the Institution's Symposium on Refining. A glance through the 260 or so references which appear at the ends of the 13 chapters reveals the care with which the author has covered the very wide field in which cobalt and its compounds are now of growing importance. Chapters 1–5 deal with the history, occurrence, metallurgy, and chemical, physical and mechanical properties of the metal, while those which follow record in some detail the various uses to which it is being put. These latter range from its inclusion as an important constituent in both ferrous and non-ferrous alloys—in high-speed cutting tools, permanent magnets and in alloys specially resistant to corrosion, to heat and to abrasion—to the important part it plays as a catalyst in synthetic reactions and in its biological and biochemical functions in regard to the nutritive values of plant life by its presence in soils and fertilizers. These latter aspects of the distribution of cobalt have received particular attention from scientific agriculturalists in New Zealand.

It is, however, to the valuable summary of the progress made in recent years in the extraction and refining of the metal, given in Chapter 3 under 'Metallurgy', to which particular attention may be directed in this short review. Here are to be found detailed descriptions of the operations in the Belgian Congo, in Northern Rhodesia, and in French Morocco. The long-established plants of the Deloro Smelting and Refining Company of Ontario and the part played by that company in more recent years in the recovery of cobalt from the copper-cobalt-iron alloy produced in Northern Rhodesia receives due recognition. Exploratory work in other countries is also described and particularly certain investigations made by the United States Bureau of Mines on the hydrometallurgy of cobalt. Electrodeposition from purified solutions, as carried out successfully at Jadotville by the Union Minière, is described, although the recent account of cobalt refining by the Murex Co., to which reference has been made above, shows with what success the carefully-purified oxides can be reduced either to rondelles or to metal powder.

S. W. Smith

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*Books (excluding works marked *) may be borrowed by members personally or by post from the Librarian, 424, Salisbury House, London, E.C. 2.*

Books and Pamphlets:

*American Bureau of Metal Statistics. *Year book*, 1948. N.Y.: The Bureau, 1949. 112 p. $3.

Metall und Erz. *Generalregister der Zeitschrift 'Metall und Erz', Band 1–42, 1904–1945.* W. Andrae, comp. Clausthal-Zellerfeld: Ges. D. Metallhütten u. Bergleute e.V., 1949. 104 p. DM 15.

Mond Nickel Co., Ltd. *The nickel bulletin, vol. 21, 1948.* London: The Company, 1949. 196 p., illus., diagrs., tabs.

Sheffield University, Mining Society. *Mining magazine*, 1949. Sheffield: The Society, 1949. 78 p., illus., diagrs., tabs. 2s. 6d.

VERDINNE, Henri. *Le problème de l'organisation scientifique du travail dans les mines.* Bruxelles : Comité National Belge de l'Organisation Scientifique, 1947. 135 p., diagrs., tabs., biblio. 11s. 6d.

Government Publications :

CYPRUS, MINES DEPT. *Annual report of the Inspector of Mines for the year 1948,* by W. Parry James. Nicosia : Govt. Printing Office, 1949. 7 p., map, tabs., diagrs. 3s.

DEPARTMENTAL COMMITTEE ON TAXATION AND OVERSEAS MINERALS. *Report.* London : H.M.S.O., July 1949. 28 p., appendix. 6d. (Cmd. 7728.)

FEDERATION OF MALAYA, GEOLOGICAL SURVEY DEPT. *Report of the Geological Survey Department for the year 1948,* by F. T. Ingham. Kuala Lumpur : Govt. Printer, 1949. 59 p., maps. 4s. 8d.

GT. BRITAIN, MINISTRY OF FUEL AND POWER. *Report of the Mineral Development Committee.* London : H.M.S.O., July 1949. 106 p., tabs., appendix. 2s. (Cmd. 7732.)

UNION OF SOUTH AFRICA, DEPT. OF MINES. *Annual report including reports of the Government Mining Engineer and the Geological Survey for the year ended 31st December, 1947.* Pretoria : Govt. Printer, 1949. 176 p., diagrs., tabs. 20s.

Proceedings and Reports :

ASSOCIATION OF MINE MANAGERS OF THE TRANSVAAL. *Papers and discussions, 1946–1947.* J'burg : Transvaal Chamber of Mines, 1948. 421 p., illus., diagrs., tabs.

TRANSVAAL CHAMBER OF MINES. *55th annual report, year 1944.* J'burg : The Chamber, 1949. 135 p.

Maps :

Lac de Gras, district of Mackenzie, North-West Territories. Geological survey, map 977A. Scale 1 in. = 4 ml. Ottawa : Dep. of Mines and Resources, 1949.

Moberly creek, West of sixth meridian, Alberta. Geological survey, map 963A. Scale 1 in. = 1 ml. Ottawa : Dep. of Mines and Resources, 1949.

Smithers - Fort St. James, British Columbia. Geological survey, map 971A. Scale 1 in. = 8 ml. Ottawa : Dep. of Mines and Resources, 1949.

INDEX OF RECENT ARTICLES

Classified according to the Universal Decimal Classification. All articles indexed are available in the Joint Library but the current issues of journals are not available for loan.

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Subject to revision.] [*A Paper to be submitted for discussion at the October*, 1949, *General Meeting of the Institution of Mining and Metallurgy.*

Investigations on the Production of Electrolytic Cobalt from a Copper-Cobalt Flotation Concentrate*

By H. L. Talbot,† *Member*, and H. N. Hepker‡

SYNOPSIS

This paper describes laboratory and pilot plant investigations on the production of electrolytic cobalt from a flotation concentrate produced at the concentrator of the Rhokana Corporation, Ltd., Nkana, Northern Rhodesia. Two processes are described; the first in which the concentrate is smelted to matte from which the cobalt is leached with sulphuric acid, and recovered by electrolysis from the purified solution. In the second process the concentrate is roasted and the cobalt extracted from the calcine by leaching with hot water, after which the cobalt is recovered from the purified solution by electrolysis. These processes have particular application for the recovery of cobalt from sulphide concentrates of relatively low cobalt content and high iron to cobalt ratio.

Introduction

The present paper describes investigations on the production of electrolytic cobalt from a concentrate obtained by selective flotation of the ores of the Rhokana Corporation, Ltd., at Nkana, Northern Rhodesia. This investigation was undertaken with the object of developing an improved method of treatment for the recovery of cobalt from the flotation concentrate, one which would be more suitable for the future ores containing a higher proportion of chalcopyrite and pyrite than those which have been treated in the past. These investigations were carried through from small-scale tests to continuous pilot-plant operation and the final data obtained have been utilized as a basis for the design of a full-scale plant for the production of electrolytic cobalt.

Cobalt occurs as a minor constituent of the Rhokana ores, which are mined essentially for their copper content. Although the cobalt content of the ores varies widely, it is not feasible to mine the higher-grade cobalt ores selectively for separate treatment. For this reason, concentrator practice must be adjusted to conform with the variation in cobalt head grade which has a marked effect upon the grade and percentage recovery of cobalt in the selective concentrate.

* Paper received on 20th April, 1949.

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‡ Research Metallurgist, Central Research Laboratory, Nkana, Northern Rhodesia.

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Investigations on the Production of Electrolytic Cobalt from a Copper-Cobalt Flotation Concentrate*

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SYNOPSIS

This paper describes laboratory and pilot plant investigations on the production of electrolytic cobalt from a flotation concentrate produced at the concentrator of the Rhokana Corporation, Ltd., Nkana, Northern Rhodesia. Two processes are described ; the first in which the concentrate is smelted to matte from which the cobalt is leached with sulphuric acid, and recovered by electrolysis from the purified solution. In the second process the concentrate is roasted and the cobalt extracted from the calcine by leaching with hot water, after which the cobalt is recovered from the purified solution by electrolysis. These processes have particular application for the recovery of cobalt from sulphide concentrates of relatively low cobalt content and high iron to cobalt ratio.

Introduction

The present paper describes investigations on the production of electrolytic cobalt from a concentrate obtained by selective flotation of the ores of the Rhokana Corporation, Ltd., at Nkana, Northern Rhodesia. This investigation was undertaken with the object of developing an improved method of treatment for the recovery of cobalt from the flotation concentrate, one which would be more suitable for the future ores containing a higher proportion of chalcopyrite and pyrite than those which have been treated in the past. These investigations were carried through from small-scale tests to continuous pilot-plant operation and the final data obtained have been utilized as a basis for the design of a full-scale plant for the production of electrolytic cobalt.

Cobalt occurs as a minor constituent of the Rhokana ores, which are mined essentially for their copper content. Although the cobalt content of the ores varies widely, it is not feasible to mine the higher-grade cobalt ores selectively for separate treatment. For this reason, concentrator practice must be adjusted to conform with the variation in cobalt head grade which has a marked effect upon the grade and percentage recovery of cobalt in the selective concentrate.

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Description of Present Practice

The principal copper minerals in the ore are bornite and chalcopyrite, cobalt occurring mainly as the sulphide mineral carrollite ($CuCo_2S_4$) and an oxide cobalt mineral, of similar composition to asbolite, which is not recoverable by flotation. Pyrite is present in varying proportions and since this mineral has flotation characteristics very similar to carrollite these two minerals cannot readily be separated in the concentrating process.

In the selective flotation process the bulk of the copper is removed as a 'copper' concentrate, after which a 'cobalt' concentrate is recovered, containing the recoverable cobalt for subsequent treatment in the cobalt circuit at the smelter. The percentage recovery of cobalt in the cobalt concentrate shown in Table I was 31·70. This Table shows the assays of copper and cobalt obtained at the concentrator for a typical period.

Table I

Typical Assays of Concentrator Products

	Copper %	Cobalt %
Mill Feed	3·22	0·104
Copper Concentrate	47·60	0·56
Cobalt Concentrate	31·22	2·75
Mill Tailing	0·38	0·04

The cobalt concentrate is smelted separately in a reverberatory furnace, the cobalt entering the matte along with the copper. During the converting of the matte the cobalt is slagged off with the iron and the converter slag is granulated, mixed with coke, and smelted in an electric arc furnace, where the cobalt is recovered in the form of an alloy assaying about 37 per cent cobalt, 12 per cent copper, and 50 per cent iron. This alloy is granulated, bagged and shipped overseas for refining. The overall recovery of cobalt from ore to alloy is about 14 per cent.

While this method gives a comparatively low production cost for cobalt as a by-product from copper smelting operations, the process losses are unavoidably high when treating concentrates of low cobalt content and having a high ratio of iron to cobalt. During the earlier years of operation the copper in the ore occurred mainly as the mineral bornite and cobalt grade was materially higher, the iron-cobalt ratio being between 3 and 4 to 1. Under these conditions smelting losses were not unduly high and alloy containing from 40 to 50 per cent cobalt could be produced. During the past several years, however, the cobalt grade has declined and the increasing proportion of chalcopyrite and pyrite in the ore has raised the iron to cobalt ratio to over 5 to 1 in the cobalt concentrate. In consequence smelter cobalt recovery has steadily decreased and it has become increasingly difficult to produce alloy of a grade acceptable to the refiners.

Detailed investigations were carried out at the smelter to determine whether it was feasible to effect any material improvement in the process. These studies indicated that high smelting losses are unavoidable when treating a concentrate with a high iron to cobalt ratio, and, since a further increase in this ratio was to be expected from the ores to be treated in the future, it was considered that an alternative process would have to be developed in order to effect any improvement in recovery of cobalt from the flotation concentrate.

Preliminary Experimental Investigations

Investigations on the concentration of Rhokana ores have not indicated any satisfactory method of separating carrollite from chalcopyrite and pyrite by either flotation or gravity methods. Thus, any alternative process should be independent of the iron to cobalt ratio in the cobalt concentrate. A leaching method appeared to offer the best possibility of securing the required separation between cobalt, iron, and copper.

Two parallel investigations were undertaken which are referred to in the present paper as the 'matte-leach' and 'roast-leach' processes. Small-scale tests had indicated that cobalt could be leached with sulphuric acid solution from a matte produced by smelting the cobalt concentrate under highly reducing conditions. These tests were confirmed on a larger scale and the investigation was subsequently carried through to the production of electrolytic cobalt metal. An alternative method investigated was the roasting of the cobalt concentrate to convert the cobalt to water-soluble sulphate, followed by hot water leaching of the calcine. Although this process showed considerable promise, no definite conclusions could be reached until these results were confirmed by continuous tests on a scale large enough to be representative of plant practice. Since the necessary equipment was not yet available, investigation of the matte-leach process was carried forward and completed before the larger-scale tests on the roast-leach process were undertaken.

A detailed description follows of the two processes. It is considered that both processes might have economic application, depending upon the type of cobalt-bearing material to be treated and local plant conditions, and it is hoped that the technical data presented may be of assistance to other investigators in this field.

The Matte-Leach Process

The major steps in the matte-leach process may be briefly summarized as follows.

The cobalt concentrate, mixed with suitable proportions of limestone, coal, and converter slag, is smelted in a reverberatory furnace to produce a matte from which the cobalt can be extracted by leaching with sulphuric acid. This matte, dry ground to *minus* 65-mesh, is then batch leached with hot sulphuric acid solution. In the leaching operation cobalt and iron are dissolved, while the copper remains as sulphide in the residue, which is returned to the smelter for copper recovery. The separation of cobalt from iron in the leach solution is accomplished by the addition of a soluble sulphide at a controlled *p*H whereby the cobalt is precipitated as hydrosulphide and recovered by filtration. The filtered cake is then subjected to an oxidation roast to convert the cobalt to sulphate which can be extracted from the calcine by leaching with hot water. The resulting solution is freed from the final traces of iron by oxidation with air and the addition of limestone. The purified solution, containing from 20 to 25 g. per litre of cobalt, is then electrolysed. The stripped cathode metal is prepared for the market by melting and granulation.

(A) *Laboratory Scale Furnace Tests*

The successful application of this process is primarily dependent upon a high recovery of cobalt from the reverberatory matte. When the cobalt concentrate is smelted under normal conditions, with the addition of limestone flux and 2–3 per cent by weight of coal, the extraction of cobalt from the matte by sulphuric acid leaching was too low to be economically acceptable. A detailed investigation was undertaken to determine the variables upon which the leaching properties of the matte are dependent. It was already known that the addition of cold converter slag and additional coal to the reverberatory charge had a marked effect on the leaching characteristics of the resulting mattes. The effect of these constituents when added to the charge in varying proportions was then studied by means of small-scale laboratory tests carried out in an electric arc furnace using a graphite crucible and a standard basic charge of 500 g. of cobalt concentrate. In the course of these tests the effect of the following variables was investigated :

(1) Increasing coal without addition of converter slag ;

(2) Increasing converter slag without addition of coal ;

(3) Increasing coal with a constant percentage of converter slag ;

(4) Addition of pyrite and converter slag with and without coal ;

(5) Finer grinding of converter slag before mixing into initial charge ;

(6) Maintaining the charge molten for varying periods after smelting.

From the results of these tests the following conclusions were drawn :

(*a*) A matte of acceptable leaching characteristics was not produced with up to 6 per cent coal in the charge without the addition of converter slag, or by adding up to 15 per cent of converter slag when no coal was used. In either case the maximum leach extraction obtained was less than 50 per cent.

(*b*) With the addition of 5½ per cent coal and a minimum of

14 per cent converter slag a leach extraction of over 95 per cent was obtained.

(*c*) The matte obtained with the addition of up to 10 per cent pyrite with a constant percentage of converter slag (14 per cent) did not give a satisfactory leach extraction unless a minimum of 5 per cent coal was used.

(*d*) Finer grinding of the converter slag before mixing with the charge did not improve cobalt recovery or leach extraction.

(*e*) Maintaining the charge in a molten state for longer periods resulted in higher cobalt slag losses and lower cobalt leach extraction.

The results of the laboratory furnace tests were used as a basis for further experimental work which was carried out in a 15 ft. long by 8 ft. wide reverberatory furnace, which was erected at the smelter for this purpose. It is of interest to note that the experimental data obtained in the small-scale laboratory smelting tests were reproduced very closely in the experimental reverberatory furnace both with respect to the leaching characteristics of the mattes and the distribution of metals between the matte and slag.

(B) *Experimental Reverberatory Furnace Tests*

An experimental reverberatory furnace was erected in the smelter building, so that charging could be carried out by one of the installed cranes and pulverized coal drawn from one of the smelter unit pulverizers. Side and end walls were constructed of magnesite brick 12 in. thick, and the flat-suspended roof of alumina-silica brick was 20 in. thick, with four charge openings, 6 in. by 12 in. on either side at the firing end. The internal dimensions were 15 ft. long, 8 ft. wide, and 4 ft. high above the slag bottom, which was poured in place to a depth of 3 ft. All charges were premixed on the floor and delivered by overhead crane to a hopper above the furnace by means of charging boats. The furnace was fired by a single pulverized coal burner delivering into a combustion chamber which extended 4 ft. outside the furnace on the charging end. The operation of this furnace was very satisfactory and the unit was of sufficient size to give results which were comparable with full-scale practice.

Several campaigns were run using increasing proportions of converter slag, and each run was continued for a period of sufficient duration to establish a condition of equilibrium, which varied from 5 to 8 days. All mattes tapped during the progress of the campaign were examined for leaching characteristics, and selected samples of matte were set aside for complete microscopic and analytical examination. The following is a summary of the results from four selected furnace campaigns.

(1) *No. 9 Campaign*

This campaign was run with a charge corresponding approximately to normal smelter practice on the cobalt reverberatory furnaces to provide a standard for comparison.

FURNACE CHARGE

	Weight lb.	% of Total	Cu %	Co %	Fe %	S %	SiO_2 %
Cobalt concentrate	39,900	81·2	34·08	2·24	13·07	22·35	14·74
Converter slag	nil	—	—	—	—	—	—
Limestone	7,140	14·5	—	—	0·45	—	2·1
Coal.....................	2,100	4·3	—	—	2·0	—	3·0
Total	49,140	100·0					

FURNACE PRODUCTS

	Weight lb.	Cu %	Co %	Fe %	S %	SiO_2 %
Matte	27,695	48·17	3·18	17·61	26·18	—
Slag	13,770	1·92	0·33	4·95	—	44·27

PERCENTAGE RECOVERIES

	Copper	Cobalt	Iron	Sulphur
Matte	98·1	95·0	87·7	81·3
Unaccounted for ...		+3·8	+5·1	

The average cobalt leach extraction on the matte tapped during this campaign was 87 per cent. This percentage extraction corresponds closely to that obtained on average cobalt mattes from the smelter reverberatory furnaces.

The following three campaigns were run with increasing percentages of converter slag in the charge while the percentage of coal was maintained at approximately 5·5 per cent.

(2) *No.* 10 *Campaign*

FURNACE CHARGE

	Weight lb.	% of Total	Cu %	Co %	Fe %	S %	SiO_2 %
Cobalt concentrate.	43,974	68·4	24·75	2·09	13·44	20·91	21·31
Converter slag	6,853	10·7	5·50	3·37	48·12	—	20·69
Limestone	10,000	15·5	—	—	0·45	—	2·1
Coal.....................	3,500	5·4	—	—	2·0	—	3·0
Total	64,327	100·0					

FURNACE PRODUCTS

	Weight lb.	Cu %	Co %	Fe %	S %	SiO_2 %
Matte	27,981	38·87	3·81	26·18	25·96	—
Slag	26,964	1·43	0·37	10·28	—	41·17

PERCENTAGE RECOVERIES

	Copper	Cobalt	Iron	Sulphur
Matte	96·6	91·4	72·5	79·0
Unaccounted for ...		+1·6	+8·3	

The average cobalt leach extraction on the matte tapped during this campaign was 70 per cent.

(3) *No.* 11 *Campaign*

FURNACE CHARGE

	Weight lb.	% of Total	Cu %	Co %	Fe %	S %	SiO_2 %
Cobalt concentrate.	29,296	63·7	27·21	2·23	13·27	21·83	18·62
Converter slag	7,557	16·4	7·81	2·58	45·65	—	19·51
Limestone	6,690	14·5	—	—	0·45	—	2·1
Coal.....................	2,500	5·4	—	—	2·0	—	3·0
Total	46,043	100·0					

FURNACE PRODUCTS

	Weight lb.	Cu %	Co %	Fe %	S %	SiO_2 %
Matte	29,949	38·32	3·62	27·23	26·00	—
Slag	17,007	0·91	0·33	11·24	—	42·01

PERCENTAGE RECOVERIES

	Copper	Cobalt	Iron	Sulphur
Matte	98·2	93·4	75·8	89·2
Unaccounted for ...		+0·4	+6·4	

The average cobalt leach extraction on the matte tapped during this campaign was 90 per cent.

(4) *No.* 12 *Campaign*

FURNACE CHARGE

	Weight lb.	% of Total	Cu %	Co %	Fe %	S %	SiO_2 %
Cobalt concentrate.	57,654	60·9	29·99	2·82	13·81	23·24	16·01
Converter slag	18,829	19·9	7·90	2·47	46·98	—	18·56
Limestone	13,000	13·7	—	—	0·45	—	2·0
Coal.....................	5,225	5·5	—	—	2·0	—	3·0
Total	94,708	100·0					

FURNACE PRODUCTS

	Weight lb.	Cu %	Co %	Fe %	S %	SiO_2 %
Matte	47,964	38·17	3·68	28·35	25·66	—
Slag	34,229	1·37	0·40	12·17	—	38·43

PERCENTAGE RECOVERIES

	Copper	Cobalt	Iron	Sulphur
Matte	97·5	92·8	76·6	91·9
Unaccounted for ...		—8·9	+4·8	

The average cobalt leach extraction on the matte tapped during this campaign was 95 per cent.

this can be accomplished by increasing the amount of iron in the charge and, at the same time, smelting under more strongly reducing conditions. Provided that sufficient iron is present in the matte, the cobalt will preferentially go into solid solution in the iron and iron sulphides rather than in the copper sulphide.

During the course of this investigation, the method of Smirnov and Mishin for the determination of metallic cobalt was applied to the cobalt mattes. The results of these determinations showed close agreement with the volume percentages of metallic cobalt-iron alloy as estimated under the microscope, and these results could be directly correlated with the leaching characteristics of the mattes. This assay method was therefore adopted for control purposes in the experimental smelting operation.

(D) *Acid Leaching of Cobalt Matte*

The matte was crushed to *minus* ½-in. and 200-lb. charges were wet ground in a ball-mill to give a *minus* 65-mesh product, of which about 70 per cent was *minus* 325-mesh. The pulp after settlement and decantation contained about 10 per cent moisture and was used without drying for the leaching tests. Standard leaches were carried out using 60 lb. of ground matte in 25 imperial gallons of solution.

The initial tests were carried out with a solution containing 340 g. of acid per litre and heated to 80° C. The matte was added slowly to the agitated solution over a 20-minute period, but no reaction, as indicated by evolution of hydrogen sulphide, was observed until agitation and heating had been continued for about two hours. At this point the matte suddenly commenced to dissolve with explosive violence and practically the whole charge was blown out of the agitator. Other tests proceeded with less violence, but the commencement and progress of the reaction were erratic and unpredictable. It was then observed that if, after the first addition of matte, a small quantity of dry pulverized matte was added, steady solution of the charge was promoted and the remainder of the matte reacted in a normal manner. Even when steady dissolution was obtained by this means, the cobalt leach extraction averaged only about 80 per cent as compared with over 95 per cent obtained in previous small-scale tests on the same matte. Obviously it was essential to determine the cause of the unsatisfactory behaviour of the larger-scale leach tests.

Tests were then carried out to compare the cobalt leach extraction on wet and dry ground samples of matte after varying periods of exposure to atmosphere. The poor leach extraction on wet ground samples was immediately apparent. Dry ground matte showed no decrease in leach extraction after 20 days' exposure and the average extraction was 96 to 97 per cent. The wet ground material showed an initial extraction of 81 per cent, which decreased steadily to about 66 per cent after 7 days' exposure and then remained fairly constant. It was also noted that even after more prolonged exposure of wet ground matte the leach extraction did not drop below about 60 per cent.

It was concluded that the drop in cobalt leach extraction was due to some form of oxidation which is promoted and accelerated by the presence of moisture. It is believed, from information gained in the subsequent microscopic investigation, that the cobalt which is present in the form of metallic cobalt-iron alloy is affected by this oxidation and rendered insoluble under these conditions of leaching. The remainder of the leachable cobalt, which is held in solid solution in the iron sulphides, is not subject to this oxidation. The microscopic examination also revealed that after oxidation the iron sulphide was totally surrounded by a film of metallic cobalt-iron alloy of varying thickness. The delayed reaction encountered when leaching oxidized matte may be ascribed to the slow penetration of the oxidized film surrounding the iron sulphide. When this film is penetrated, which we consider is brought about by the reducing action of the liberated hydrogen sulphide on the oxide film, the reaction proceeds very rapidly. This would also explain why steady dissolution of oxidized matte is promoted by the addition of a small amount of unoxidized dry material.

Leaching tests, carried out on samples of dry ground matte, indicated that the only significant variables were acid concentration, time, and temperature. The effects of these variables are illustrated in the following tabulations :

(*a*) *Acid Concentration*

Temperature 80° C., Leaching Time 2 hours

Acid concentration g/litre on...	290	240	195	160	110
Acid concentration g/litre off...	197	143	96	57	16
%Cobalt leach extraction......	97	97	91	81	64

(*b*) *Leaching Time*

Temperature 80° C., Acid concentration 290 g/l.

Leaching time — minutes ...	30	60	90	120	180
% Cobalt leach extraction ...	84	93	97	98	99

(*c*) *Temperature*

Leaching Time 2 hours, Acid concentration 290 g/l.

Temperature of leach °C. ...	50	70	80	90	98
% Cobalt leach extraction ...	83	96	98	98	97

The high acid concentration required to obtain good extraction of cobalt will be noted. Preliminary small-scale tests employing a ' neutral ' leach—that is, using only a slight excess of acid over the theoretical requirement—gave a maximum leach extraction of only about 65 per cent. Hydrogen sulphide evolution in these leaches appeared to be normal, and it was therefore concluded that only the cobalt contained in the iron sulphide was extracted, and that the metallic cobalt-iron alloy was not attacked by acid of this concentration.

(*E*) *Selective Precipitation of Cobalt from Iron*

Leaching tests carried out on dry ground matte with an initial acid concentration of 240 g. per litre, and leached for 2 hours at 80° C., gave a leach extraction of 97·4 per cent of the cobalt and 86·4 per cent of the iron. The leach solution after filtering was copper free and contained 12 g. of cobalt, 50 g. of iron, and 140 g. of acid per litre. It was intended to precipitate cobalt as the sulphide in neutral solution, and various reagents were tried out for this purpose. The high acid content must first be reduced, for which purpose ground limestone was used, and the resulting precipitate of calcium sulphate was removed by filtration to avoid dilution of the cobalt precipitate produced in the subsequent process step.

It was found that a satisfactory selective separation between iron and cobalt could be effected by the addition of sodium sulphide, calcium sulphide, or by passing hydrogen sulphide gas through the solution, and calcium sulphide was adopted as being the most economical. The calcium sulphide was prepared as a slurry by absorbing hydrogen sulphide gas in a milk of lime suspension. In practice, sufficient hydrogen sulphide would be generated during the leaching of the matte, and this gas would be utilized for the preparation of the calcium sulphide slurry. The precipitate assayed 9 per cent cobalt and 9 per cent iron with 96 per cent recovery of cobalt and 77 per cent rejection of iron in the waste solution.

After filtering and washing, the precipitate was subjected to an oxidizing roast under controlled temperature conditions to convert the cobalt to water soluble sulphate and the iron to insoluble oxide or ferric sulphate. When these tests were conducted, no suitable equipment was available for carrying out a detailed investigation of roasting technique. Roasting tests were conducted on a small scale at a temperature of 800° C. A hot water leach of the resulting calcine gave 90 per cent extraction of the cobalt and 3·5 per cent of the iron, and yielded a solution containing 25 g. of cobalt and 1 g. of iron per litre. The authors have no doubt that with suitably-designed roasting equipment cobalt extraction could be increased to over 95 per cent without additional iron extraction. The iron content of the solution must be reduced to about 0·02 g. per litre before electrolysis, and this was effectively accomplished by oxidation with air and addition of powdered limestone, preferably with the solution heated to about 60° C. After filtering to remove the sludge, the cobalt was recovered from the purified solution by electrolysis. The recovery of cobalt by electrolysis is outlined in the description of the roast-leach process which follows.

The Roast-Leach Process.

During the course of the investigational work, small-scale tests had been carried out on the extraction of cobalt from roasted concentrate by leaching. The results of these tests showed promise, but could not be confirmed without a furnace capable of continuous operation. When an 8-ft. diameter, single hearth Huntington Heberlein round furnace became available, the investigation was undertaken on the extraction of cobalt from the cobalt concentrate by roasting and leaching. The major steps in this process may be briefly summarized as follows :

The starting product is the same as for the matte-leach process—namely, the cobalt concentrate obtained from the selective flotation of the ore. The cobalt concentrate is subjected to a sulphatizing roast under closely controlled conditions, converting the cobalt to water soluble sulphate and rendering the copper and iron relatively insoluble.

The resulting calcine is leached with hot water, the solution recovered by filtration, and then purified to remove the small amounts of iron and copper dissolved in the leaching operation. The cobalt is precipitated from the purified solution as hydroxide by the addition of milk of lime, and, after filtering, the cobalt hydroxide cake is redissolved in spent solution from the electrolysis. By this means, acid build up and cobalt concentration in the circulating electrolyte are controlled, and at the same time this step assists in maintaining the solution balance and preventing the build up of magnesium in the electrolyte. The circulating solution is electrolysed and the cathode metal melted and granulated for the market.

(1) *Roasting of Concentrate*

The roasting tests were carried out on cobalt concentrate from current production having the following analysis :

Chemical Analysis of Cobalt Concentrate

	%
Copper	32·49
Cobalt	3·21
Iron	12·71
Sulphur	22·75
Silica	15·92
Alumina	4·00
Calcium Oxide	1·15
Magnesia	1·62

Arsenic, antimony, molybdenum and tungsten were present only in trace quantities.

The sulphatizing roast takes place in three principal stages and will be carried out in multiple hearth furnaces with concurrent flow—that is, with the concentrate and gases travelling in the same direction. A description is given of the roasting in three separate stages but in practice some overlapping takes place.

(a) Desulphidizing

During the first stage of roasting, the copper, iron and cobalt sulphides are converted to the oxides with conversion of the sulphur to sulphur dioxide. It is essential that the temperature should be kept from rising above 550° C., since the Fe_2O_3 when formed at higher temperatures loses much of its efficiency as a catalytic agent for the reaction $SO_2 + O = SO_3$ which is essential for the following stage. In addition, higher temperatures promote the formation of cobalt ferrites which are not converted to water soluble sulphate in the later stages and are therefore lost.

(b) Sulphatizing of the Copper

Under the catalytic influence of Fe_2O_3, SO_2 is oxidized to SO_3 which, in turn, reacts with CuO to form $CuSO_4$. At the same time a small amount of $CoSO_4$ is formed by a similar reaction. The optimum temperature range for these reactions is between 580° to 640°C.

(c) Sulphatizing of the Cobalt

The reactions taking place in this stage result in the decomposition of the $CuSO_4$ to CuO, which is not water soluble, and the formation of water soluble $CoSO_4$. The optimum temperature range is between 680°C. and 710°C. These changes may be expressed by the following chemical reactions :

$$CuSO_4 + CoO = CoSO_4 + CuO \quad \text{.........................} \quad (1)$$

$$CoO + SO_3 = CoSO_4 \quad \text{.......................................} \quad (2)$$

With the 8-ft. diameter single-hearth furnace, optimum temperature conditions could not be attained simultaneously on the single hearth to suit all three stages of the roast. The best results were obtained when the maximum bed temperature on the hearth was between 620°C. and 650°C. Under these conditions the percentage cobalt extracted from the calcine by subsequent hot water leaching was between 70 and 75 per cent. It is anticipated that the extraction will be above 85 per cent with a suitably designed multiple hearth roaster.

(2) *Leaching and Purification*

Although continuous leaching was tried, it was found that considerably higher extraction was obtained by batch leaching and this method was adopted for the large-scale tests. The time necessary to effect complete solution of the cobalt sulphate is extremely short, but it was found advantageous to extend the leaching period to about 45 minutes, at a temperature exceeding 80°C., in order to promote reprecipitation of dissolved copper. This is brought about by the action of the basic lime and magnesia constituents of the calcine. However, if the leaching time is extended for too long, some precipitation of cobalt takes place and this limits the extent to which copper can be eliminated by this means. The leach solution after filtering averaged 20 to 25 g. cobalt, 7 to 10 g. copper and 0·3 to 0·4 g. iron per litre. The filtered and washed residue would be delivered to the smelter for recovery of its copper content.

The copper and iron still remaining in solution were removed by oxidation with air and the addition of milk of lime to a controlled *p*H of 5·2–5·5. By this means the copper can be reduced to less than 0·2 g. and iron to less than 0·02 g. per litre. A small amount of cobalt is precipitated with the copper and iron, and the sludge from this purification step is sent to the residue retreatment section where this cobalt is recovered. The clarified cobalt solution is then passed over granulated cobalt metal to reduce the copper content to less than 0·001 g. per litre.

While this purified solution is now suitable for electrolysis, additional process steps are required to maintain the electrolyte solution balance and cobalt concentration, to control the build up of acid, and to eliminate impurities such as magnesium from the electrolyte.

There is no discard of solution from the electrolyte circuit and hence an amount of water must be continuously removed equivalent to that entering the circuit with the purified make up solution.

Initially it was proposed to remove the excess water by direct evaporation, but it was evident that the capital cost of the necessary corrosion-proof equipment together with the high fuel consumption, would make this step very costly. An alternative method was therefore adopted in which the cobalt was precipitated as hydroxide from the purified leach solution by the addition of milk of lime, the precipitate recovered by filtration, and the barren solution discarded to waste thereby eliminating soluble impurities, principally magnesium. By adding the cobalt to the electrolyte circuit in the form of hydroxide filter cake the evaporation problem is greatly simplified.

(3) *Electrolyte Neutralization*

The excess acid generated during electrolysis must be removed so that the *p*H of the cell discharge is maintained at a constant value. This is accomplished by using the cobalt hydroxide precipitate to neutralize the circulating electrolyte. At the same time, the cobalt dissolved in the neutralization step serves to maintain the cobalt concentration of the electrolyte at a constant value. The cobalt hydroxide precipitate contains a high proportion of calcium sulphate and lime residue which is not dissolved in the neutralization step and remains as a sludge. Owing to the slowing down of the dissolution rate as the electrolyte approaches neutrality, some undissolved cobalt hydroxide remains in the sludge. The sludge is removed in a thickener and sent to the residue retreatment section where the undissolved cobalt is recovered.

(4) *Residue Retreatment Section*

The feed to this section is made up of the sludge from the copper-iron purification and neutralization steps. The cobalt and a large proportion of the copper are redissolved by acidifying to a final pH of about 3·5. The copper is then reprecipitated by neutralizing with milk of lime to a pH of 5·5. The pulp is then filtered for recovery of the cobalt bearing solution, and the residue sent to the smelter for recovery of its copper content.

(5) *Cobalt Electrolysis*

The tests on electrolysis were carried out in a rectangular lead-lined cell 20 in. long, 12 in. wide, and 14 in. deep, containing 4 anodes and 3 cathodes. Anode to anode spacing was 4½ in. and the clearance between the electrodes and the cell lining was 3 in. at the sides, 4½ in. from the bottom, and 3¼ in. at either end. The cathodes were mild steel grooved plates 6 in. wide, $\frac{3}{16}$ in. thick, with 9 in. submergence, and the lead anodes were 6 in. wide, ¼ in. thick with 9 in. submergence. The anodes were of grid construction to maintain the anode current density at about 20 amp./sq. ft. in order to prevent the deposition of cobalt oxides at the anode.

Various conditions of current density, electrolyte temperature, cobalt-concentration and acidity were investigated. The data given in the following table are representative of the results obtained under the best average conditions which would be applicable to plant operation. The cobalt concentration of the electrolyte feed to the cell was maintained at 25 g. per litre.

Data on Cobalt Electrolysis

Duration of run—hours	72
Amp.-hours used	2342·5
Average amps through cell	32·6
Average voltage—cell busbars	2·98
Cathode current density (amp./sq.ft.)	14·5
Average temperature of cell discharge, °C.	58·4
Circulation rate—gallons per lb. Cobalt deposited	100
Average pH of cell feed	5·95
Average pH of cell discharge	1·77
Cobalt metal deposited—lb.	4·65
Cathode current efficiency, %	81·8
Total kWh. consumed	7·00
kWh. per lb. Cobalt deposited	1·51

A considerably higher current efficiency was obtained by increasing the circulation rate to reduce the acid concentration. For instance, with a flow rate equivalent to 190 gallons per pound of cobalt deposited, the pH of the cell discharge increased to 2·7 and a current efficiency of 87 per cent was obtained. However, it was considered that the gain in current efficiency at this high rate of circulation would be more than offset in plant practice by the increased size of auxiliary equipment required to handle the circulating electrolyte.

A governing factor in determining the effects of the foregoing variables was the physical character of the cathode deposit, which should be firm, adherent, and readily stripped from the blank. The cathode deposit exhibits a strong tendency to crack and peel away from the blank in the initial stages of deposition. Pickling of the cathode blanks and surface roughening with a steel brush did not prevent peeling of the deposit. It was found that light machine grooving of the mild steel plates resulted in a firm adherent deposit, which could be built up to about ¼ in. thickness and readily stripped. It was noted, however, that when the current density was increased up to 20 amp. per square foot some cracking and peeling occurred even when grooved plates were used. At a current density of 15 amp. per square foot, 12 day cathodes were built up without difficulty and no stripping trouble was encountered. The cell temperature was maintained at about 60°C. since the deposit at lower temperatures was less dense and adherent. The cathode metal was of excellent purity and had the following average analysis.

Analysis of Electrolytic Cobalt Metal

	%
Cobalt	99·89
Copper	0·041
Iron	0·032
Sulphur	0·019
Calcium	0·006
Nickel	not detected
Zinc	,, ,,
Manganese	,, ,,
Magnesium	,, ,,
Silicon	,, ,,

The experimental work was not carried beyond the production of cathode metal which in practice would require further processing to convert it to a form suitable for market. This is done by melting in an electric furnace, granulating in water, sizing and burnishing the granules. Two grades are usually marketed, namely, about *plus* ⅛-in. *minus* ¼-in., and *plus* ¼-in. *minus* ¾-in.

Summary and Conclusions

(1) Although cobalt is present only as a minor constituent of the Rhokana ores, the total cobalt content is appreciable since the tonnage of ore treated is between 3,000,000 and 3,500,000 tons per year.

(2) The leaching processes investigated would increase the production of cobalt metal by approximately 50 per cent as compared with the present alloy process. The indicated recovery of refined cobalt metal was slightly higher for the roast-leach than for the matte-leach process.

(3) A major advantage of both the new processes lies in their ability to treat ores of low cobalt grade and varying composition

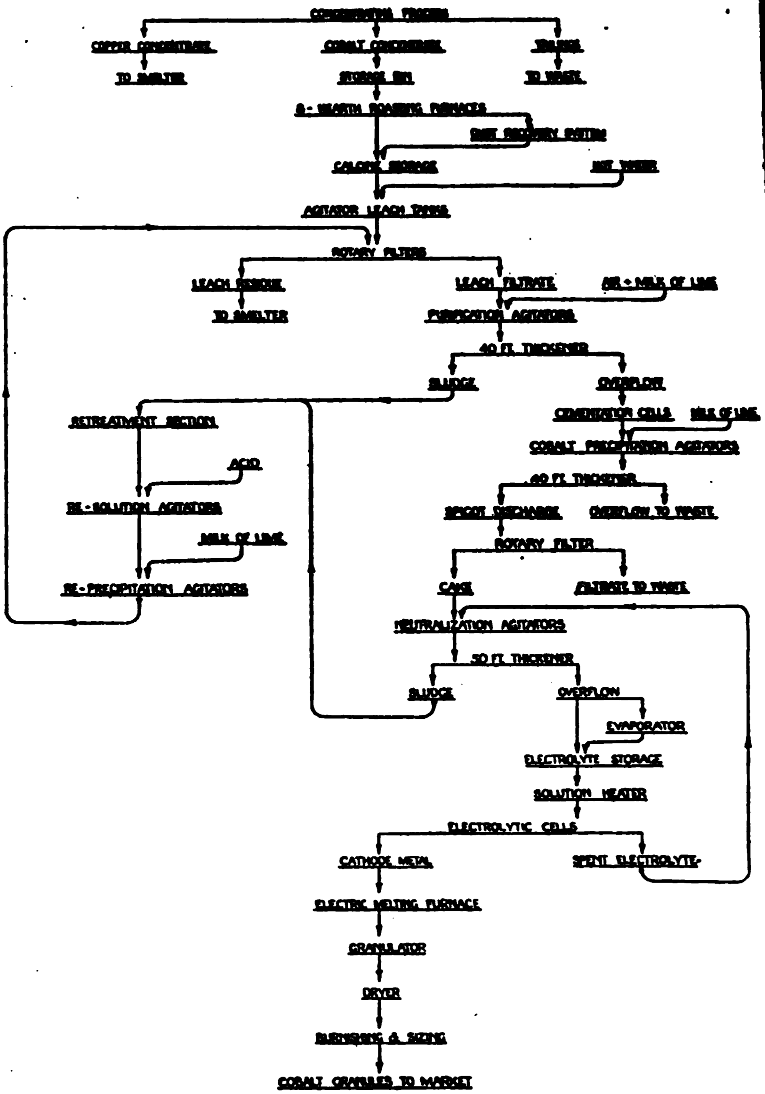

Fig. 1.—Flow-sheet for cobalt roast-leach process.

to better advantage than the present process. Their successful application would not be materially affected by the increasing ratio of iron to cobalt in the concentrate.

(4) A proposed flow-sheet for the roast-leach process is shown in Fig. 1.

Acknowledgements

The authors wish to acknowledge the assistance given to this investigation by Mr. R. R. Porter during his association with the Central Laboratory and to Miss A. J. Hall for the detailed microscopical and structural analysis of matte specimens, which proved invaluable in furthering this investigation.

They also express their thanks to the Rhokana Corporation, Ltd., and to the Anglo American Corporation of South Africa, Ltd., for their permission to publish the paper.

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Subject to revision.] [*A Paper to be submitted for discussion at the October,* 1949, *General Meeting of the Institution of Mining and Metallurgy.*

The Use of the Wet Kata Thermometer on the Witwatersrand*

By P. H. Kitto†

SUMMARY

(1) A description is given of the standard procedure for using the wet kata thermometer adopted by the Transvaal Chamber of Mines, and possible sources of error in its use.

(2) Formulae are derived correlating wet kata reading, air velocity and unventilated wet-bulb temperature which give better practical results under local conditions than existing formulae.

(3) The effects on the wet kata of high dry-bulb temperatures and of radiation from a surface at 125–130° F. were investigated. At constant wet-bulb temperature the former has little effect, while the latter only becomes important when the kata reading is very low.

(4) Experiments to find the differences in heat loss per unit area between large and small wet bodies are described, together with results obtained by other workers. They indicate that the relative heat losses per unit area of a wet kata and a man do not differ much at different air velocities.

Introduction

The wet kata thermometer has been used as a routine instrument for many years in the mines of the Witwatersrand and has proved of considerable value as an arbitrary measure of underground working conditions. From previous experience it is possible to form a very good idea of what the atmospheric conditions in a particular place are like from a wet kata reading and whether conditions are such as to necessitate prior acclimatization of the workers. It has been found, however, that readings with different instruments and by different observers under apparently similar conditions often showed large discrepancies, while attempts to correlate air velocity, air temperatures, and kata readings by means of the usually accepted formulae were not very successful.

The experiments described in the present paper were undertaken in an endeavour to improve the accuracy of the readings obtained by the instrument and to gain more knowledge of the actual significance of these readings in relation to the cooling power of the air on a man.

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(1) Accuracy and Calibration of the Instrument

It has been known for some time that to obtain good results with the instrument great care must be taken to standardise the procedure adopted when taking a reading. For example, Buist[1] described a number of ways in which errors might arise—such as the thickness of the sleeve used, how much the bulb was heated before taking a reading, the method of removing excess water from the bulb, and other minor points—and even then he found that two instruments in the same air stream did not always give the same result—in other words, the factor marked on the instrument was not always correct.

To obtain more comparable results between different observers the Chamber of Mines has adopted a standardized procedure for the use of the wet kata in Rand mines and the sleeve used is always made of the same material (a fine net) supplied by the Chamber. Furthermore, all the instruments are sent to the Chamber for calibration, which is done in the following manner.

A mark is made on the small bulb above the stem to correspond to a temperature of 117·5° F. (47·5° C.), and the instrument is always heated just sufficiently to raise the liquid to this mark before taking a reading. To check the factor the 'katas' are then tested against an arbitrarily-chosen standard instrument in a duct where the air velocity is 100 f.p.m. and the temperature and humidity are similar to those likely to be found underground (wet-bulb temperature about 85° F. and relative humidity about 95 per cent). This has been found necessary owing to the fact that katas vary somewhat in size and shape, particularly in different makes, and the factors, which are obtained by calibration of the dry instrument in still air, do not always compare well, one with another, when they are used wet in moving air. The velocity of 100 f.p.m. has been adopted arbitrarily as one similar to the velocities in which the instrument is likely to be used.

Compared under these conditions the factors of a large number of instruments are found to be correct, but variations of up to ± 10 per cent are often found. If carefully used, successive readings with the wet kata should not vary by more than about $\pm$ 1 per cent and any greater variation than this may be taken as an indication that the actual conditions affecting the cooling of the instrument have changed.

There is one other source of error which only becomes important when the wet-bulb temperature is more than about 90° F., and it is due to the fall in *rate* of cooling of the kata from 100° to 95° F. It should be remembered that, theoretically at any rate, the kata is supposed to represent a body constantly supplied with heat internally to keep it at a mean temperature of 97·5° F. while cooling from the surface. The time taken for the kata to cool from 100° to 95° F. is a fairly good measure of the rate heat would

[1] See references at the end of the paper.

be lost from such a body until the wet-bulb temperature of the surrounding air begins to approach 95° F. The temperature of the surface of the wet kata cannot fall below that of the ordinary wet bulb in the same air stream, so that if this is near 95° F. the rate of cooling over the last portion of the 5° is much too slow, and the average time taken to cool over 1° between 100° and 95° F. would be longer than (say) the time to cool from 98° to 97° F., although in both cases the cooling takes place over the same mean temperature—viz., 97·5° F. The shorter time of cooling per degree measured from 98°–97° is a better comparison with a body at a

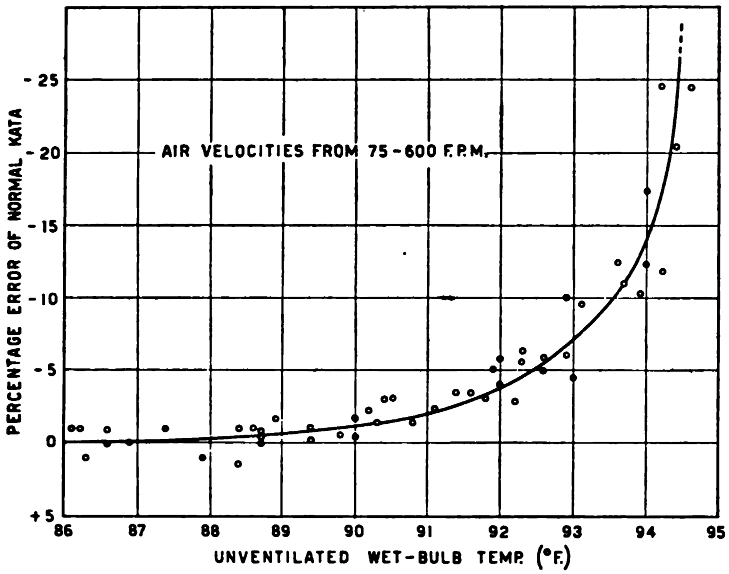

FIG. 1.—Errors in cooling power obtained by normal kata when compared with readings taken over a 1° drop (98° F.—>97° F.).

constant temperature of 97·5° F., and is therefore the more correct reading. Fig. 1 shows from actual experiments the errors which might arise in this way by the conventional use of the instrument, when compared with figures obtained by timing the cooling from 98° to 97° only.

For this reason it is desirable to have marks on the stem of the kata at 98° F. and 97° F. and to use these when the wet-bulb temperature is high or even as a matter of routine in all cases. Although readings will not be quite so accurate when timed over 1° if the rate of cooling is fast, the errors in such cases are not likely to be important as the atmospheric conditions will be good.

The ordinary formula for obtaining the cooling power of the air—

$$H' = \frac{F}{T}$$

where

H' = cooling power of the air on the wet kata in millicalories per sq. cm. per sec.

F = factor of the instrument

T = time in seconds to cool from 100° to 95° F.

will then become—

$$H' = \frac{F}{T' \times 5}$$

where T' = time in seconds to cool from 98°F. to 97° F.

The wet-bulb temperature is not likely to approach near enough to 97° F. in Rand mines to affect to any great extent readings taken in this way.

(2) The Correlation of Wet Kata Readings with the Factors Affecting its Rate of Cooling

When Professor Leonard Hill first introduced the kata thermometer he devised formalae for its use, both wet and dry, in which he gave the relationship between the cooling power of the air (on the kata), the air velocity and the air temperature (wet-bulb and dry-bulb). Since that time many other workers have produced formulae of a similar nature which they claimed gave better agreement with observed facts.

The chief causes of discrepancies between these different formulae are probably :

(*a*) Differences in technique in using the instrument, including thickness of the sleeve used ;

(*b*) differences in the measurement of the other factors involved, such as air velocity, temperature, etc. :

(*c*) failure to take into account factors which might affect the result in certain instances, such as air pressure and radiation ;

(*d*) the calculation of formulae from results obtained over widely different ranges ;

(*e*) differences between the instruments themselves, particularly as regards shape and size.

These various causes will be discussed in more detail as the different points arise.

Although a great deal of work has gone into the devising of these formulae, the results have had more academic interest than practical application in underground mining work on the Witwatersrand, because none of the existing relationships seemed to agree very well with measurements taken underground in these mines, and it seemed desirable therefore to try to find a more satisfactory relationship for use under the conditions in which the kata is used on the Rand.

Experimental Procedure

As mentioned earlier in this paper the first essential was to standardize the procedure of using the instrument and so eliminate the possibility of errors from this source, after which it was necessary to check the other instruments to be used in the experiments. For the measurement of medium and high air velocities ordinary vane anemometers were used, checked one against the other, together with the ' Velometer ' and Pitot tube to test the constancy of the air speed across the area of the anemometers. This was necessary because of the much smaller cross-section of the kata thermometer as compared with the anemometers.

For the measurement of low air speeds the Rees torsion anemometer, a low-speed vane anemometer, the velometer, and smoke were used and checked against each other. The whirling arm method was not practicable, because all measurements involving air flow were carried out in a duct in which it was possible to control air flow and wet- and dry-bulb temperatures using recirculated air (Fig. 2). For air speeds below 50 f.p.m. the most reliable method was found to be the timing of a puff of smoke over a distance of a few feet, care being taken that the smoke passed the actual position in which the kata bulb was to be suspended. Observation of the smoke showed also that there was very little turbulence in the air flow in the duct, and this is an important factor in experiments of this kind where the kata would be affected by turbulence and the other methods give linear velocity only.

Temperatures were measured with small-bulb mercury thermometers similar to those used in whirling hygrometers, all calibrated against a standard thermometer to $\pm$ 0·1° F. A cotton sleeve was placed over the thermometers for wet-bulb measurements, but the standardization of this sleeve is not of such great importance as in the case of the kata thermometer, for whereas the type of sleeve affects the *rate* of cooling considerably it has very much less effect on the actual wet-bulb temperature reached. Unventilated wet-bulb temperatures were taken with the thermometer hanging freely, but for the measurement of the true wet-bulb temperature the thermometer was inserted across a tube through which the air was sucked at not less than 600 f.p.m. by means of a suction pump. When, as in this case, the end of the sleeve dips into a small reservoir of water to keep it damp, this reservoir must be allowed to reach equilibrium near the wet-bulb temperature before taking a reading, otherwise the temperature measured may be affected by diffusion from the reservoir.

' Still air ' measurements were carried out in a large box or in the open room when conditions were steady. The air surrounding the wet kata when in use is rarely actually still during the cooling period, because the temperature of the kata surface is constantly changing and almost always different from the temperature of the air, and this causes convection currents round the bulb. The

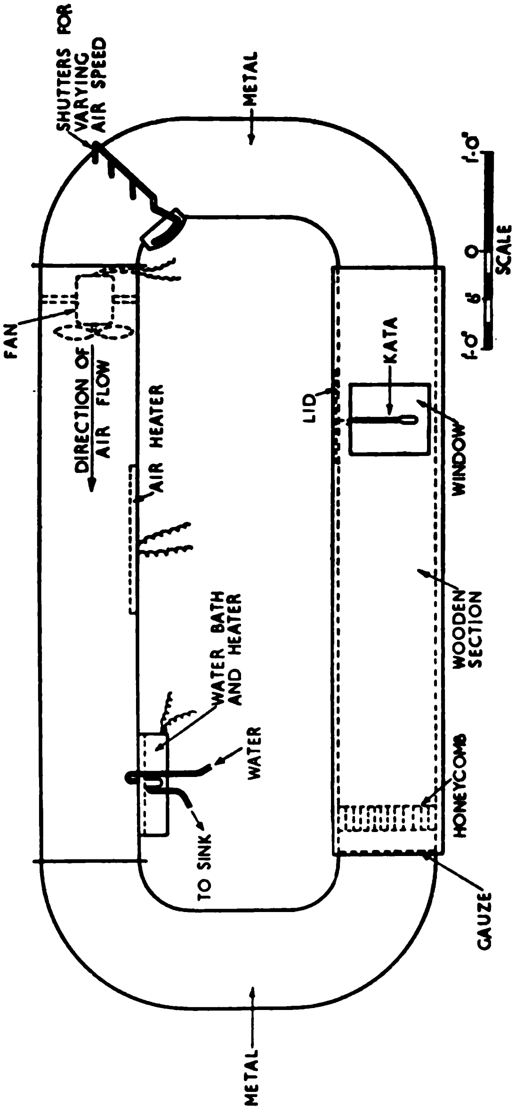

FIG. 2.—Duct for kata calibration.

experimental results given later show what effect this has when the kata is used at very low air velocities.

Method of Investigation and Results Obtained

The formula given by Rees[2] for the wet kata had been largely accepted for use at high wet-bulb temperatures and the method described by him was used to test his formula under local conditions.

(a) The Wet Kata in Still Air

Experiments were first carried out to find the still air relationship between the cooling power of the air as measured by the wet kata and the unventilated wet-bulb temperature. In Rees's formula this was given as—

$$H' = 0{\cdot}36\theta':$$

where

H' = cooling power in millicalories per sq. cm. per sec.

and θ' = the difference between 97·5° F. and the unventilated wet-bulb temperature.

This result was obtained for the unventilated wet-bulb temperature range of about 83° F. to 93° F., and showed also that dry-bulb temperatures as much as 28° higher than the wet-bulb temperatures had a negligible effect.

The author's experiments indicate that if a wider range of temperature is covered $\frac{H'}{\theta'}$ is not a constant, and that if H' is plotted on a graph against the unventilated wet-bulb temperature a slight curve is obtained (Fig. 3). It can be seen from Fig. 3 that at 90° F. $\frac{H'}{\theta'} = 0{\cdot}44$, at 80° F. $\frac{H'}{\theta'} = 0{\cdot}46$ and below that the ratio declines until

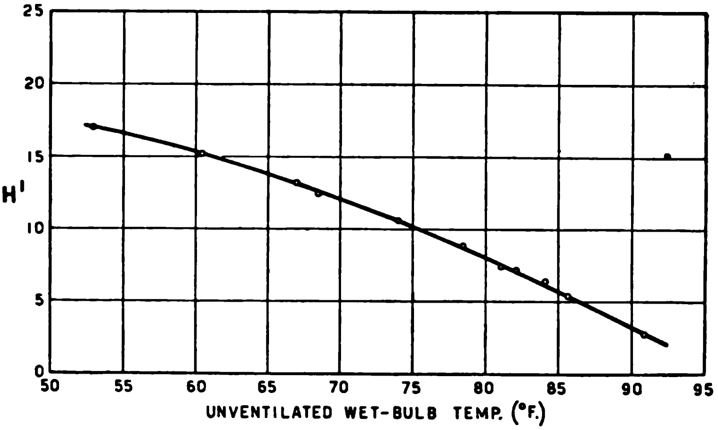

FIG. 3.—Wet kata readings in still air.

at 55° F. it is 0·39. This agrees with the result found by Buist[1], viz., that $T \times \theta'$ increases as θ' increases, because $H' = \frac{F}{T}$ and therefore $\frac{H'}{\theta'} = \frac{F}{T \times \theta'}$. As F is a constant $T \times \theta'$ is inversely proportional to $\frac{H'}{\theta'}$. However, for practical use it is the temperatures above 75° F. which are important, and in this range the ratio can be taken as 0·45 with very little error.

Nevertheless this is very different from the ratio of 0·36 obtained by Rees, and the most likely solution seemed to be the effect of the much higher altitude at which the experiments were carried out. The lower atmospheric pressure would increase the rate of evaporation and hence the cooling power of the air. Experiments were therefore carried out in still air at different depths in the mines, and these confirmed the assumption that the ratio $\frac{H'}{\theta'}$ decreased with depth—i.e., with increase of atmospheric pressure—though not to the extent expected. It was found to vary inversely approximately as the square root of the atmospheric pressure, i.e., at a pressure of 24·5 inches of mercury $\frac{H'}{\theta'} = 0{\cdot}45$, whereas at a pressure of 30·0 inches of mercury $\frac{H'}{\theta'} = 0{\cdot}41$.

The figure 0·41 still does not agree with the figure 0·36 obtained by Rees, but the difference may be accounted for by differences in technique, thickness of sleeve used, and the instruments used.

(b) *The Wet Kata in Moving Air*

Keeping the wet-bulb temperature constant, the wet kata readings were obtained over a large range of velocities and the results plotted graphically for both low and high wet-bulb temperatures. For velocities higher than about 20 f.p.m. it was found that H' plotted against the square root of the velocity gave very nearly a straight line, although H' plotted against the cube root of the velocity was even better (Figs. 4 and 5). It will be shown later that the difference is not important.

The graphs in Figs. 4 and 5 show clearly that at an unventilated wet-bulb temperature of 75° F. and dry-bulb temperature a few degrees higher, there is a marked change in the relationship between air velocity and cooling power below an air velocity of about 20 f.p.m. The reason is that at very low air speeds the air rising past the kata bulb due to convection interferes with the horizontally-flowing air current, whereas above 20 f.p.m. or so the effect becomes negligible. The figure of 20 f.p.m. varies somewhat according to the temperature of the surrounding air and hence the velocity of the convection currents, but the effect is still noticeable even when the temperatures

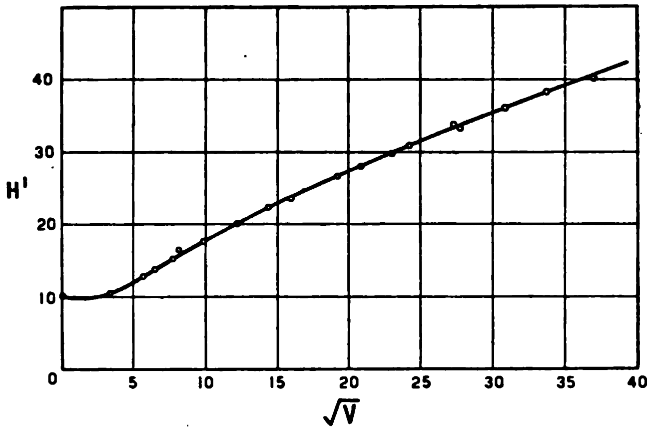

FIG. 4.—Wet kata readings at constant wet-bulb temperature (74·8° F.—75·2° F. variation).

are much higher. In the case shown, when the unventilated wet-bulb temperature is 75° F., it will be seen that from the straight-line relationship between H' and $\sqrt[3]{V}$ a wet kata thermometer in still air would indicate a velocity of about 14 f.p.m. As a general rule, the kata thermometer should not be used as a measure of air velocities below approximately 20 f.p.m.

Using the method given by Rees[2] for obtaining the complete formula, including the unventilated wet bulb, the author plotted

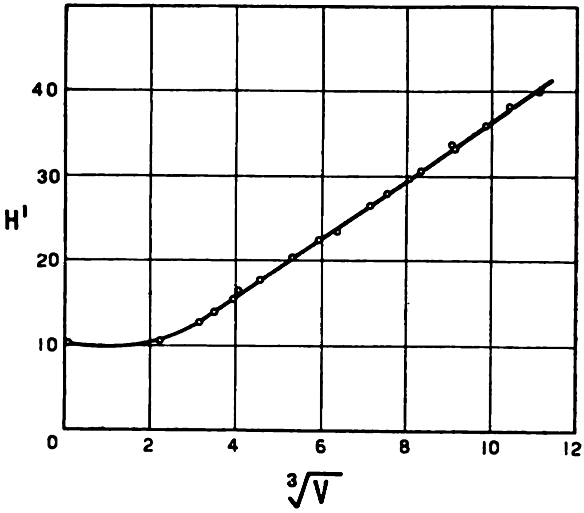

FIG. 5.—Wet kata readings at constant wet-bulb temperature (cf. Fig. 4).

TABLE 1

Air Velocity f.p.m.	Professor Hill $\frac{H'}{\theta'} = 0\cdot19+0\cdot082\sqrt[3]{V}$	J. P. Rees $\frac{H'}{\theta'} = 0\cdot36+0\cdot033\sqrt{V}$	P. H. Kitto $\frac{H'}{\theta'} = 0\cdot4+0\cdot04\sqrt{V}$.	$\frac{H'}{\theta'} = 0\cdot1+0\cdot15\sqrt[3]{V}$
27	0·44	0·53	0·61	0·55
64	0·52	0·62	0·72	0·70
125	0·60	0·73	0·84	0·85
	$\frac{H'}{\theta'} = 0\cdot056+0\cdot11\sqrt[3]{V}$			
216	0·72	0·85	0·99	1·00
343	0·83	0·97	1·14	1·15
512	0·94	1·11	1·31	1·30
729	1·05	1·25	1·48	1·45
1,000	1·16	1·40	1·66	1·60

$H' - 0{\cdot}45\theta'$ against the square root of the velocity and obtained a formula for use at this altitude. It was then necessary to introduce a factor to allow the formula to be used at any depth in the mines, but this did not prove to be easy. In the case of still air a correction could easily be applied, but when the air is moving the effect of pressure was found to be much less noticeable. The reason for this is not quite clear, but it is possibly due to some effect that the greater mass of air for the same velocity at high pressures has on the rate at which the heat is transferred from the kata to the air.

The author found that the ratio for $\frac{H'}{\theta'}$ of 0·45 did not give quite such good results for moving air as a somewhat lower one, about 0·40, and so the formula was worked out using this figure by plotting $H' - 0{\cdot}4\ \theta'$ against $\theta'\ \sqrt{V}$. It became

$$H' - 0{\cdot}4\ \theta' + {\cdot}04\ \theta'\ \sqrt{V}$$

or

$$H' = 0{\cdot}4\ \theta'\ (1 + 0{\cdot}1\ \sqrt{V}),$$

and this was found to give sufficiently accurate results at all the pressures tested (from 24·5 in. of mercury to 31·6 in. of mercury) to enable the correction for pressure to be left out entirely.

If $\sqrt[3]{V}$ is used a formula can be found in a similar manner. The straight line in Fig. 5 may be extrapolated to the axis to give a hypothetical figure of $H' - 0{\cdot}1\ \theta'$ when V is 0. Plotting $H' - 0{\cdot}1\theta'$ against $\theta'\ \sqrt[3]{V}$ the author obtained the graph shown

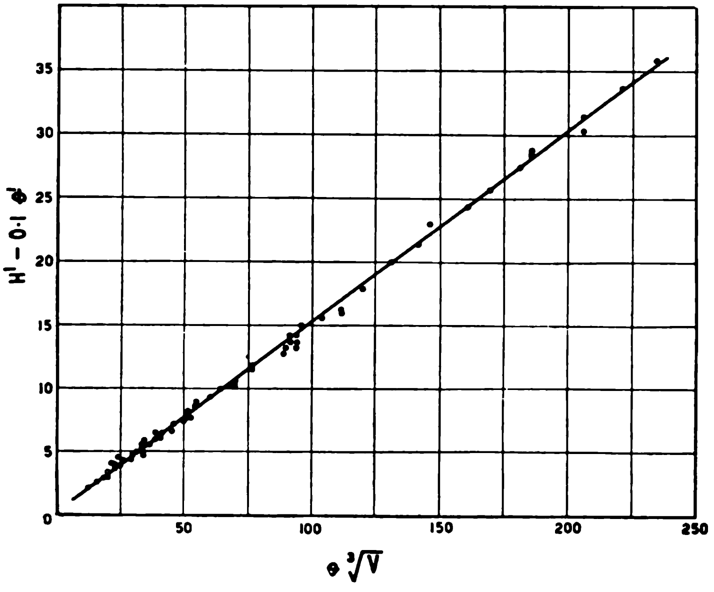

FIG. 6.

in Fig. 6, which includes velocities from 30 to 1,800 f.p.m. and values of θ' from 5° to 40° F. From this the formula obtained is $H' = 0{\cdot}1\theta' + 0{\cdot}15\,\theta'\,\sqrt[3]{V}$.

Although at first glance this seems to be quite different from the other formula given—viz., $H' = 0{\cdot}4\,\theta' + 0{\cdot}04\,\theta'\,\sqrt{V}$, the two give very similar results. Table I shows the values of $\frac{H'}{\theta'}$ obtained from these two formulae, that of Rees and those of Professor Hill, for different air velocities.

Professor Hill used the true wet-bulb temperature for his formulae, but in practice the author finds that in the moist conditions usually found underground on the Rand there is seldom need for such accuracy as to make it necessary to specify either the unventilated or the ventilated wet-bulb temperature, even for low air velocities and temperatures up to 90° F. At about 15 f.p.m. air velocity (due to convection currents only) the author obtained the following differences between a thermometer hanging freely exposed to the atmosphere and the same thermometer swung on the end of a piece of string at not less than 600 f.p.m.:

Wet Bulb Temperature (°F)

Unventilated	*Ventilated*
62·9	62·3 (dry-bulb 68·3)
64·6	64·0 (dry-bulb 70·0)
83·1	82·4 (dry-bulb 89·0)
88·1	87·4 (dry-bulb 96·0)

In circumstances where the dry-bulb temperature is much higher, as might be the case in dry mining, the differences would be greater.

The most important thing to remember with these formulae is that the air velocity measured by the kata is never 0, and that the kata should not be used to measure air velocities below about 20 f.p.m.

The formulae given in Table I have been compared with numbers of practical results taken underground in mines of the Witwatersrand by different workers and the best agreement is undoubtedly given by the last two. At very low air velocities the formula in the last column seems to give the best results. The general tendency is for the velocity calculated from the formulae to be slightly higher than that obtained by measurement, due to the fact that the measured velocity is linear and the kata takes account of velocity from any direction—i.e., turbulent air. From this point of view the velocity as measured by the kata would seem to give a better indication of the cooling effect this air would have on men working in it than the linear velocity.

(c) *The Effect of High Dry-Bulb Temperatures and of Radiation on the Wet Kata*

At the present time neither of these factors is of much importance in the Witwatersrand mines, where the air is very humid and rock surface temperatures are usually only a few degrees above the air temperature, but some experiments have been carried out to determine the relative effects of each. The results indicate that differences of up to 30° F. in the dry-bulb temperature have very little effect on the relationship between kata reading, wet-bulb temperature, and air velocity. One would expect therefore that changes in dry-bulb temperature of this order would have little effect on working efficiency providing the wet-bulb temperature remained constant, and this has been borne out by recent experiments.[3] It was found, however, that in very hot dry conditions the standard thin kata sleeve used was not suitable, as it started to dry out, with a resultant marked effect on the kata reading, and it can be assumed that the heating effect on a human being

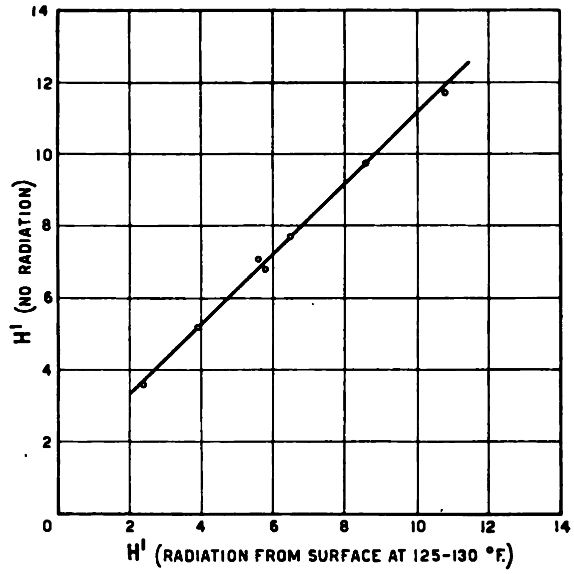

Fig. 7.—Effect of radiation on the wet kata.

would be just as marked if conditions were such that the sweat was evaporating faster than it could be produced.

The effect of radiation has been tested within the limits likely to be encountered in mining. The experiments were carried out in a duct of square cross-section which could be heated electrically over a short distance opposite to the point at which the kata was suspended. Conditions were therefore similar to those which might be obtained in a drive where heat was radiating from the rock surface.

The heated surface was maintained at a temperature between 125° F. and 130° F., and the kata readings compared with those taken at the same wet-bulb temperature and air velocity when there was no radiation. The temperatures were taken in a position unaffected by the radiation. The surface consisted of oxidized iron, with an emissivity of about 0·9 (blackbody emissivity = 1).

The results are shown in Fig. 7, which includes readings taken at air velocities from 50 to 200 f.p.m. and unventilated wet-bulb temperatures from 93·5° F. to 85·0° F. The radiation caused a drop in the cooling power of the air on the kata of approximately 1 millicalorie per sq. cm. per sec., and would become important therefore only when air conditions were nearing the limit of human tolerance.

(3) A Comparison between the Cooling Power of the Air on the Wet Kata and on Larger Bodies

From time to time the kata thermometer has been criticized on the grounds that it is too small an instrument to be used as a measure of the effect environmental conditions will have on a body the size of a man, and in particular that it exaggerates the effect low air velocities will have on such a body. The experiments described in what follows were undertaken to gain more information on the comparative rates of heat loss per unit area of different-sized bodies.

The absolute and accurate measurements of the heat lost from a body such as the kata thermometer over the range 100° F. to 95° F. when cooling from 117·5° F. to a temperature below 95° F. is not an easy matter, and for the purpose of this investigation it was not considered necessary. It was shown as long ago as 1924[4] that the kata thermometer does not accurately measure the cooling power of the air even on itself, but it does serve as a somewhat arbitrary method giving comparisons of the varying conditions between one place and another. The question that arises is how far these comparisons reflect the effect such conditions will have on a human being. Estimations by various means of the actual ratio of the heat lost per unit area per second from the wet kata and from a man in the same environment vary considerably, ranging from a heat loss for the kata four or five times that of a human being[5] to a heat loss of approximately the same for both[6]. The chief variable is the ability of acclimatization of a human being, which in itself varies from one person to another and from time to time, and for this reason no instrument can be designed which will give accurately the effect any particular set of conditions will have on any one person without that person's own reactions to different conditions being known.

It can be seen, then, that although the wet kata itself is very sensitive in its reaction to varying conditions, the reading obtained can at best only be approximately correlated with the possible reactions of men working or resting under the same conditions. The experiments which have been carried out, however, may help to throw some light on the controversial question of the usefulness of kata thermometer readings.

Previous Work

Much work has already been done on the difference in rate of heat loss from the surface of large and small bodies immersed in fluids of different kinds, including air, and some of the results obtained give a useful guide to the differences that might be expected when comparing a kata thermometer with a human being.

A start may be made by considering separately the ways in which a wet kata loses heat. They may be divided as follows :

(1) From the interior of the kata to the surface, by convection and conduction ;

(2) From the surface: (*a*) by evaporation of water; (*b*) by convection, both natural and forced; and (*c*) by radiation. There will also be a small amount of diffusion from the surface, but this can be neglected.

The relative amounts of heat lost by each method are not constant and may vary greatly from one place to another. The heat lost by evaporation (*a*) will decrease with increase in wet-bulb temperature and will also depend largely on (*b*), the amount of convection. The amount of heat lost under (*b*) is less important than the effect convection has on the rate of evaporation and hence the amount of heat lost under (*a*), a conclusion which can be easily demonstrated by comparing a wet kata with a dry kata under the same conditions. The quantity of heat lost by radiation (*c*) will depend on the temperature of the surrounding surfaces. In certain circumstances (*c*) might have a negative value—i.e., if the surrounding surfaces are at a higher temperature than the kata surface it will gain heat by radiation from them instead of losing it.

Experiments on men resting in still air[7] show clearly the big variations that may be expected in the relative amounts of heat lost by (*a*) and (*c*) as the conditions change, but in those working places of the mines of the Witwatersrand where conditions are such that the kata readings are low, radiation is at present unimportant, and the heat from the kata or a man is lost mainly by evaporation and by the effect of forced convection (i.e., air velocity) particularly insofar as it affects the rate of evaporation.

(*a*) *Evaporation*

The evaporation of water from surfaces of varying shapes and sizes has been studied by Powell[8]. He found that for each type of surface the rate of evaporation per unit area tended to assume a common value for large bodies but increased rapidly for small bodies. In the case of a sphere, for example, the rate of evaporation per unit area at 400 f.p.m. was almost three times as great for a sphere of diameter 2·5 cm. as it was for a sphere of diameter 20 cm. but above this size indications were that the change was small.

Furthermore, this ratio did not alter much over a wide range of air velocities, although the tendency was for it to decrease at the higher velocities.

The approximate relationship deduced by Powell for a sphere was that the rate of evaporation per unit area per unit vapour pressure difference (between the saturated surface and the air stream) is proportional to $V^{0.62}$ and $d^{-0.5}$, where V is the velocity and d the diameter, but this variation evidently does not hold for spheres of diameter greater than about 20 cm., which, he says, tend to assume a constant rate of evaporation per unit area.

(b) Convection

Natural convection—i.e., convection caused solely by the difference in temperature between the surface of the body and the surrounding air—is seldom of much importance in Rand mines, but it may be considered as a matter of interest. According to Péclet(9), working with spheres filled with water kept constantly stirred, the rate of heat loss by convection varies as $1{\cdot}778 + \frac{0{\cdot}13}{r}$ where r is the radius of the sphere in metres. He only worked with diameters between 5 and 30 cm., but if the formula is taken to hold at 2·5 and 50 cm. this gives a heat loss per unit surface area approximately 5 times as much for the small sphere as the large one. Later work on horizontal cylinders, however, indicates that the heat loss per unit area is independent of the diameter above approximately 15 cm.(10).

Finally there is the case of forced convection—i.e. the effect of air velocity. A lot of work has been done on its effect on wires and pipes where the diameter is small compared with the length, and also on other bodies, including the dry kata thermometer, which show that for bodies about the size of the kata the rate of heat loss varies as $V^{0.5}$, but in the case of larger bodies there is some evidence, though not entire agreement, that it varies according to a higher power of V—between 0·5 and 1·0(11, 12).

In the case of wires and cylinders the heat loss per unit area has been found to be proportional to $\frac{1}{d^{0.5}}$ for fine wires and something like $\frac{1}{d^{0.3}}$ for larger cylinders(12).

With a mean value of $\frac{1}{d^{0.4}}$ the comparative heat loss per unit area for cylinders of 2·5 and 50 cm. would be 4 to 1, which is similar to the differences obtained for natural convection and for evaporation.

Summary of Previous Work

Dealing with evaporation, which generally accounts for most of the heat loss, the relationship obtains that per unit area it varies as $V^{0\cdot 62}$ and as $d^{-0\cdot 5}$, whereas in the case of forced convection it is found that the heat loss per unit area varies as $V^{0\cdot 5}$ for small bodies —probably increasing to nearer V for large bodies—and as $d^{-0\cdot 3 \text{ to } -0\cdot 5}$. Thus it is likely that the proportional effect of velocity on large wet bodies will not be very different from that on small bodies, a result which is borne out by the experiments of Winslow, Herrington, and Gagge, who found that the convective heat loss from a man is proportional to the square root of the velocity(13), and those of Powell already quoted.

Experimental Procedure

Owing to the difficulty of getting larger instruments made of the same shape as a kata, the standard shape used was a sphere, and a number of glass bulbs of varying diameters was obtained, or made, each with a neck similar to that of an ordinary round-bottomed flask. These bulbs were covered with net and filled with liquid (see later), and their rates of cooling compared in still air and at different velocities.

In the early experiments the bulbs were filled with mercury and a small glass thermometer fixed with its bulb in the centre of the larger bulb, the rate of cooling being measured by the time taken for this thermometer to fall from 100° F. to 95° F. after being heated to 117·5° F., and this method gave good results for bulbs of the order of size of the kata thermometer and a little larger, but for much bigger bulbs (of diameter more than about 5 cm.) the readings became somewhat erratic, owing to the bigger difference in temperature between the top and bottom of the bulb during cooling. The larger bulbs also were not easy to handle, owing to the weight of the mercury.

Other liquids were then tried, including coloured alcohol, water, and malariol,* and the bulbs were calibrated in a similar manner to the kata thermometer by inserting a stopper with a stem of suitable thickness into the neck and marking on the stem the position of the liquid corresponding to 100° and 95°. Care was taken that no air bubbles remained in the bulb on warming and the amount of liquid was adjusted until it reached a suitable position on the stem at these temperatures. There must be another small bulb above the stem to hold the liquid which expands above the marks in the preliminary heating to 117·5° F. The top has a small opening to the atmosphere, and for this reason alcohol, particularly, was not very suitable, owing to evaporation, the bulbs needing frequent recalibration. Water was better in this respect, and results have been obtained with it up to a bulb size of 22·5 cm. diameter.

The method adopted was for each bulb in turn to be compared with a small spherical bulb of 2·5 cm. diameter and the ordinary kata thermometer (to give them a 'factor' in 'still air'—i.e., air

* Liquid of low vapour pressure used in mosquito control; a product of the Shell Oil Co.

moving by natural convection only), and then with the small bulb at different air velocities. Temperature conditions were those prevailing in the laboratory, and altered slightly from time to time. For this reason the difference in heat loss per unit area between each bulb and the small one was expressed as a percentage, and this percentage applied to an arbitrary heat loss which was about the mean of those obtained at any one velocity.

The results are shown in Figs. 8, 9, and 10, and in Table II. Fig. 8 shows the effect of bulb size on the measurement of the cooling power of the air at two different velocities, 100 and 200 f.p.m., if they are standardized to give the same value of H' in still air. It seems that above 12 cm. diameter the effect is negligible, and that the difference between a sphere comparable in size with the kata and one comparable in size with a human being is not likely to be more than about 20 per cent. Furthermore, the two curves follow one another closely, so that the effect of changes in velocity is not likely to be great.

Similar curves have been obtained at velocities from 30 to 500 f.p.m. In other words, when comparing the effect of different air velocities on bodies of various sizes, a small body does not exaggerate the effect of low air velocities to any great extent. This is perhaps shown more clearly in Fig. 9, in which three of the bulbs are assumed to give the same value of the cooling power of the air at 200 f.p.m., and the results given by each bulb at lower air velocities compared. The variation that does occur, however, takes place almost entirely between 0 and 100 f.p.m.

Table II gives the comparison between the surface areas of the bulbs and their rates of heat loss per unit area (i.e., the rate of cooling from 100–95° F. divided by the surface area) at 100 f.p.m., giving the bulb nearest in size to the kata a value of 1 in each case.

TABLE II

Bulb Diam. (*d cm.*)	$d^{0.5}$	Relative Surface Areas	Relative Rates of Heat Loss (100 *f.p.m.*)	Relative Rate of Heat Loss per unit area (R)	$\frac{1}{R}$
1·1	1·05	0·19	0·37	1·95	0·51
2·5	1·58	1·0	1·0	1·00	1·00
5·0	2·24	4·0	2·1	0·525	1·91
8·2	2·86	10·7	3·9	0·365	2·74
12·9	3·59	26·7	7·4	0·277	3·61
22·1	4·70	78·4	14·9	0·190	5·26

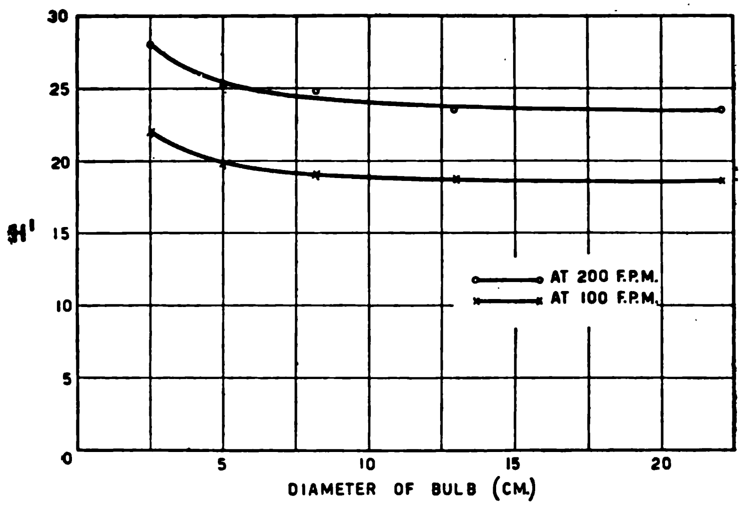

FIG. 8.—Effect of bulb size on the measurement of H'.

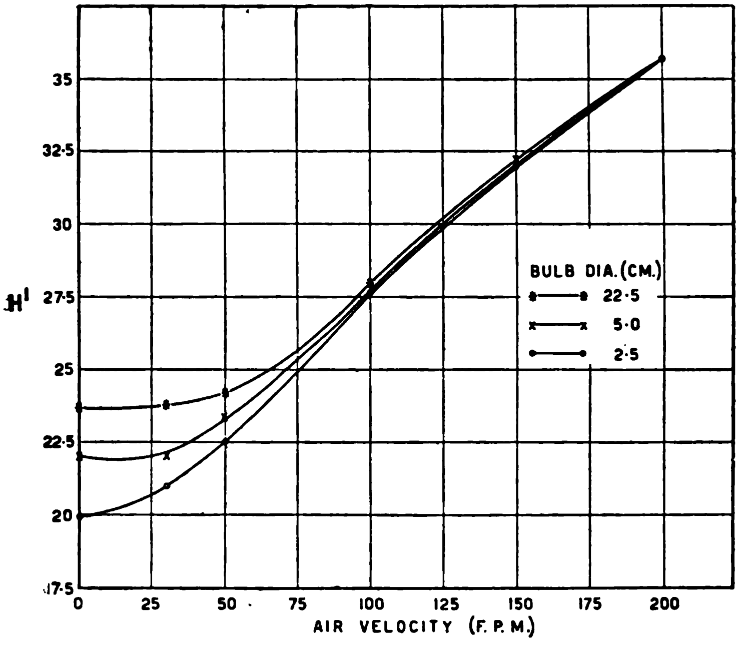

FIG. 9.

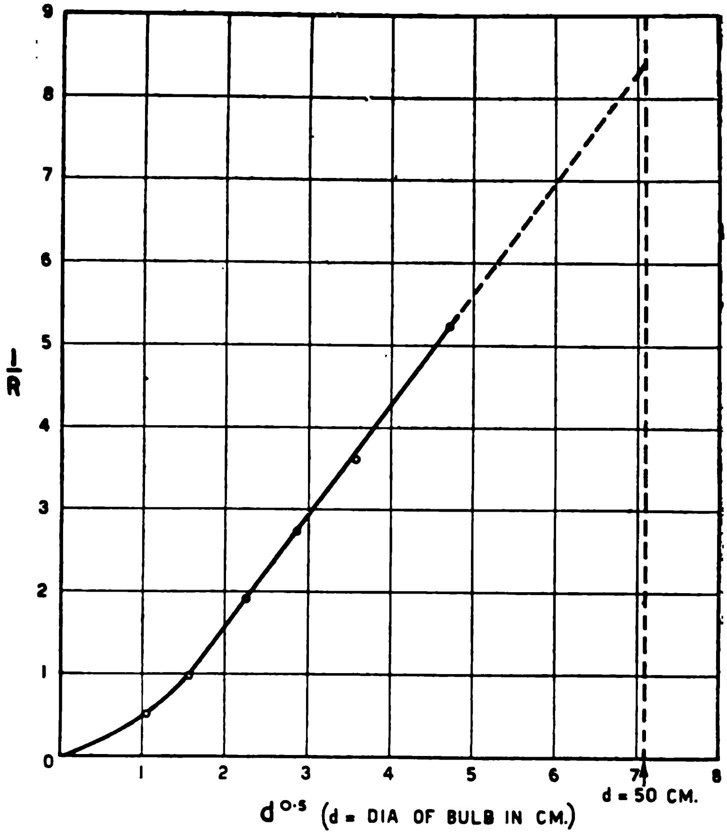

FIG. 10.—Effect of bulb size on the rate of heat loss per unit area (at 100 f.p.m.).

In Fig. 10 the reciprocals of these relative rates of heat loss per unit area for each bulb $\left(\frac{1}{R}\right)$ are plotted against $d^{0.5}$ (d = diameter of bulb in cm.), which gives a reasonably straight line for values of d above 2·5 cm. By extrapolation, a bulb of diameter 50 cm. would lose heat at approximately $\frac{1}{8}$th the rate per unit area that a bulb the size of a kata loses heat. If the rate of evaporation per unit area tends to assume a constant value for large spheres as stated by Powell[8], this ratio would be less than 8 to 1.

Conclusions and Discussions

From comparisons of different sized spheres it is shown that under the conditions tested the loss of heat per unit area for a wet spherical surface varies approximately as $d^{-0.5}$ (where d is not less than 2·5 cm.), and that for a sphere comparable in size to a human being the rate of heat loss per unit area might be approximately $\frac{1}{8}$th that of a sphere near the size of a kata. It is shown also that this ratio does not alter very much with changes in air velocity, so that as far as size alone is concerned the kata thermometer can be compared without very great error with a human being.

The actual ratio does not matter as long as the relative effect of varying air velocities is similar, and it has been shown that the difference between small and large bodies is not likely to be of much importance. Support is given to this result by the experiments which showed that, like the kata, the convective heat loss from a man is proportional to the square root of the velocity[13].

The effect of shape has not been thoroughly investigated, but it is apparent that for any type of body the bigger the surface-area/volume ratio the faster that body will lose heat at any particular air velocity, providing the air comes in contact with the whole of the surface. In the case of men working underground their surface-area/volume ratio is much bigger than the kata, but parts of the surface may be shielded from the air flow, which in turn may vary from practically streamlined to extremely turbulent. Turbulent flow will increase the rate of heat loss from a surface, some workers stating that for turbulent flow the rate of heat loss per unit area varies more nearly as V than as $\sqrt{V}$[14]. These possible variations are probably not of much importance when compared with the big difference in reaction to changes of environment that is found from one person to another. Although my experiments indicate that the wet kata thermometer can be used as a means of comparing the cooling powers of different air conditions on a man, it does not follow that it can therefore be used as a direct measure of the efficiency of a man working in those conditions, because a big change in heat loss per unit area from a man's body may have a relatively small effect on his work output. Much work has already been done on the comparison between kata readings and work output, but there is scope for a lot more, particularly in connection with the effects of radiation and high dry-bulb temperatures.

The inclusion of these factors in the correlation between kata readings and work output might help to increase the usefulness of wet kata readings as a comparative measure of working conditions.

The author's thanks are due to the Transvaal Chamber of Mines for its permission to publish this paper.

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[redacted]

FURTHER CONTRIBUTED REMARKS ON

Notes on Mining Education and Postgraduate Training*

By J. A. S. Ritson, O.B.E., D.S.O., M.C., *Member*

Mr. S. E. Willows-Munro : In South Africa, a well-established approach to the mining engineering profession is the three- or four-year full-time mining course taken at the University. It is generally agreed that this course must be supplemented by much practical experience, to enable the young men to develop ' mining sense ' as well as the art of controlling men. There are, however, many who enter the mining industry directly in junior capacities who are anxious to fit themselves for more responsible posts but whose educational background has not equipped them with a knowledge of the scientific principles needed to do so. The technical college is able to offer these junior learners adequate educational facilities on a part-time basis, as part of a nation-wide system whereby, in State-subsidized institutions, a full range of industrial and commercial education is available to all. The Union Government's subsidies amounted, in 1946, to just under £1,000,000.

A brief résumé of the development of mining education in the Union† might be of interest to members.

Until the discovery of the diamond mines in Kimberley in 1871 South Africa had few industries, and the only workshops in existence at that time were those attached to the Cape Government Railways. The discovery of gold in 1886 on the Witwatersrand, with the rapid development that followed, served to focus the attention of South Africans on mining. In 1894 a scheme for the training of mining engineers was adopted by the Government of the Cape of Good Hope, and this led to the establishment of the South African School of Mines at Kimberley in 1896, with Dr. James G. Lawn as the first professor, now one of our Past-Presidents. The work of the school was seriously interrupted by the South African War.

Following the report of the Technical Education Commission in 1903, the Kimberley School of Mines was transferred to the Transvaal Technical Institute in Johannesburg, which originally only undertook the training of mining engineers but soon introduced courses in other branches of engineering. This was the forerunner of the South African School of Mines and Technology, which was established in 1910, and which subsequently became the University of the Witwatersrand.

*Paper, *Bull.* 511, June, 1949; discussion, *Bull.* 513, Aug., 1949.

†Orr, Professor John. Technical education and training. *Trans. S.Afr. Inst. Elect. Engrs.*, April, 1932.

An offshoot of these institutions is the Witwatersrand Technical College, an institution constituted as a place of higher education under the Higher Education Act of 1923. The Mining Department of the Witwatersrand Technical College during the last ten years has had 7,335 registered students (excluding approximately 1,200 ex-volunteers). The maximum number during any one year was 1,594 spread over 110 classes, the area served extending over a distance of 200 miles, from Klerksdorp to Witbank. The war affected these classes adversely, the numbers falling at one time to 325.

Mining is a complicated art and science. Mining sense and the handling of men are developed by actual practical experience, and the science of mining by systematic study. People who control industry may be divided broadly into three groups: (1) consultants, (2) managers, and (3) producers, with their sub-groups. In mining there is a similar classification: consulting engineer, general manager, mine manager, sectional manager, underground managers, surveyors, study department, metallurgists, assayers, mine overseers, shift bosses, miners and plant operators.

The technical college should not poach on the preserves of the full-time University or School of Mines training, and its sphere should cover the field from Government Mining Training School trainees and plant operators to sectional managers. The subjects recognized as forming the integral parts of a vocational mining course should be mathematics (pure and applied), physics, chemistry, geology, mechanical engineering, surveying, metallurgy, scientific management, production control, labour control, and training of personnel.

Table II

VOCATIONAL MINING EDUCATION

NATIONAL DIPLOMA IN METALLIFEROUS MINING

The diploma will be awarded to candidates who satisfy the examination requirements in the following 14 subjects:

Group I. *Eight* subjects from:

Mining I*	Mining Economics I*
Mining II*	Surveying I
Mining III*	Surveying II*
Geology I	Practical Surveying*
Geology II	Mining Plant
Mathematics III	Power Plant
Mechanical Engineering	

Group II. *Five* subjects from:

Metallurgy of Gold: Theory*	Assaying I: Practical*
Metallurgy of Gold: Practical*	Ore Dressing
Determinative Mineralogy	Metallurgy I
Assaying I: Theory*	Metallurgy II

Group III. *One* subject from:

National Senior Certificate—Language A or Economics

*Compulsory subjects.

TABLE II—*continued*

ATIONAL DIPLOMA IN ASSAYING AND METALLURGICAL ANALYSIS

e diploma will be awarded to candidates who satisfy the examination rements in the following 14 subjects:

P I. *Eight* subjects from:

ematics III	Assaying I: Theory*
istry I: Theory*	Assaying I: Practical*
istry I: Practical*	Metallurgy II
istry II*	Determinative Mineralogy
istry III	Metallurgy of Gold: Theory
lurgy I	Metallurgy of Gold: Practical

P II. *Five* subjects from:

ing II: Theory*	Metallurgical Analysis II: Theory
ing II: Practical*	Metallurgical Analysis II: Practical*
lurigcal Analysis: Theory	Physical Chemistry
lurgical Analysis: Practical*	

P III. *One* subject from:

Vational Senior Certificate—Language A or Economics

NATIONAL DIPLOMA IN NON-FERROUS METALLURGY

e diploma will be awarded to candidates who satisfy the examination ements in 14 subjects:

P I. *Eight* subjects from:

istry I: Theory*	Assaying I: Theory
istry I: Practical*	Assaying I: Practical*
lurgy I*	Determinative Mineralogy
lurgy II*	Geology I
istry II*	Mechanical Engineering

P II. *Five* subjects from:

lurgy of Gold: Theory*	Physical Chemistry
lurgy of Gold: Practical*	Assaying II: Theory
istry III	Assaying II: Practical
Dressing*	Power Plant*

P III. *One* subject from:

Vational Senior Certificate—Language A or Economics

NATIONAL DIPLOMA IN FERROUS METALLURGY

e diploma will be awarded to candidates who satisfy the examination ements in 14 subjects:

P I. *Eight* subjects from:

istry I: Theory*	Chemistry II
istry I: Practical	Iron and Steel Manufacture I*
lurgy I*	Metallography I*
lurgy II*	Assaying I: Theory
minative Mineralogy	Assaying I: Practical

P II. *Five* subjects from:

and Steel Manufacture II*	Metallurgical Analysis I: Practical
trial Metallurgy I	Metallography II
trial Metallurgy II*	Assaying II: Theory
lurgical Analysis I: Theory	Assaying II: Practical

P III. *One* subject from:

Vational Senior Certificate—Language A or Economics

* Compulsory subjects.

Particulars of courses, covering a total period of five years, as laid down by the Union Department of Education, are given in Table II. The first and second years' courses cover the requirements for the Mine Overseers Certificate of Competency, while the first, second, and third years cover the requirements for the Mine Managers' Certificate of Competency.

That the above is an adequate approach to mining education is shown by the subsequent careers of many past students who hold positions ranging from mine managers and inspectors of mines, to other responsible positions in mining and associated industries. Many students on completion of their National Technical Certificates have continued their studies and obtained B.Sc. degrees in chemistry, geology, mathematics, or physics, externally, through the University of South Africa.

In contrast, one must record a lack of interest on the part of many junior officials due principally to (1) low general education, and (2) lack of encouragement on the part of the employer.

The objects of a vocational mining course should be :

(1) The training of junior mine officials in the scientific principles and technical methods which form the basis of economic productivity, safety and health.

(2) To be an important service department of the industry. On this account, contact should be kept between the industry and teaching institute, so that a steady flow of specially-trained personnel is available.

The following points should also be borne in mind in any scheme of training :

(1) As mines become deeper and more highly mechanized, success in management is largely dependent upon the technical training and executive ability of the officials.

(2) Mining officials of all grades must have considerable experience in practical mining, as well as instruction in theoretical subjects.

(3) The provision of essential practical experience and of instruction in scientific principles and technology are not the dual responsibilities of a single authority, but a divided responsibility, which should be shared by the management of the mine and by the teaching institute.

Mr. T. R. H. Nelson : I would suggest that the average mining man, whether belonging ' to coal or metal ', is of the ' middle class '. Most of us have to be content with earning a reasonable living as shiftbosses, surveyors, and superintendents in the mine, the mill, or on the surface, probably having married at an early age. A young graduate generally jumps at the best thing he can get—the ' middle class ' job, wherein, as he grows older, ambitions fade and he becomes disgruntled. Speaking personally, as one in a position this last few years to hire and fire mine and surface labour at his own discretion, I would not change an hour of my mucking and drilling experience for a month of my time as a sampler and surveyor.

It seems to me that the average metal mining student stays at school until he matriculates at 17 years of age or thereabouts and then goes on to a metalliferous school of mines, wherein he maintains uninterrupted contact with all that is desirable in the way of social life, with its healthful values and broadening effect. It may mean that he virtually starts his mining career from the comparatively sheltered life of home and school, lacking in contact with his fellow men, and ignorant of the fact that he has to make a living in a highly competitive world. He probably hasn't done a stroke of hard work in his life, but he has the advantage of background and probably the makings of a leader. What the educational authorities could do is to see that every budding mining engineer gets a chance for his latent capabilities before being crushed and stulted by a not particularly sympathetic and kindly world.

I suggest, therefore, that every would-be mining engineer, whether in coal or metals, should be granted the facility to remain at school until he is of an age to matriculate or to obtain the higher certificate. Then let him go to an operating mine for from 12–18 months as a learner-worker at a living wage. Therein he would learn something of all departments which would allow him to choose between mine, mill, and surface, and to concentrate thereon. Having qualified through practical examination he could then attend mining school and ultimately graduate, not only as a mining engineer with considerable practical experience but also as a young man less unsophisticated and more knowledgeable than would otherwise have been the case. At the same time the principals of the mining schools should take every opportunity of ' boosting ' mining engineering as a profession, so that the youngster of 15 knows of mining from the very day when he first begins to sit up and take sensible notice. Professor Ritson's suggestions for a timetable for young graduates are admirable, but I would think it very much better to have anything from 12 to 18 months' practical experience as a mucker and driller, as a timberman, or a mill hand prior to graduation, this to be followed by up to two years of intensive practical training in mine or mill or whatever other phase of mining on which the graduate has decided to concentrate.

A further point is that surely the smaller mines, which cannot afford to maintain schools of instruction, considerably outnumber the larger that can. How much better then to learn all that it is possible to learn prior to graduation, and then to spend a while not only on the final details, but, more particularly, to get experience of that most important of subjects ' labour relations '. After a couple of years or so the graduate would know whether his driller was ' pulling his weight ', and would also be able to design a sound steel headframe.

Mr. L. C. Hill: Professor Ritson is to be congratulated on introducing a subject of vital importance and there will be few who do not agree with his general thesis that mining cannot be learned solely out of books. A mining engineer is essentially a practical man and must needs practice his profession in order to become proficient in it. The initial stages of this practice, when a young man has just graduated and has obtained his first job, are what I imagine Professor Ritson had chiefly in mind when he wrote this paper. He puts in a plea for a course of postgraduate training which in his view should consist of two years' hard physical work during which the young engineer should himself carry out a large number of the various jobs that go to make up mining.

The practical experience that a mining engineer must have had before he can be considered in any way competent consists of two quite distinct phases in my view:

(1) He should try and obtain as much as possible of the artesan's aptitude by handling the various tools of his trade, and actually carrying out a variety of jobs with his own hands. This phase should, in my opinion, be carried out before he graduates and not afterwards, as Professor Ritson suggests.

(2) The other and much more important phase consists in getting practice in organizing and supervising the performance of these same tasks by other people, and this, of course, must necessarily be accomplished after graduation.

How these objects can best be attained will be discussed later, but meanwhile the point I wish to stress is that it is wrong to expect a young engineer to go back to school once he has graduated. Even where this is possible, it must have a stultifying effect on most young engineers, who at this stage in their careers are normally and rightly anxious to get to grips with a real job and to prove themselves.

It is surely better from the point of view both of the young man himself and of the company that employs him that he should be given some responsibility as soon as possible, even though this should be limited, at first, to looking after one development end. As his capacity increases so he can learn how to organize, on however small a scale; how to see a job through, however dull it may appear; how to improvise in an emergency when he has to; and how to clean up a mess when necessary. Above all, he is learning all the time how to work in with his fellow men be they his superiors or subordinates.

In this way the young engineer feels that he is, to some extent at least, earning his pay and that he is a part of a necessary interesting and active operation in which he is using his abilities and training to the best advantage. The management, on the other hand, has the opportunity of seeing how he tackles a job and of estimating the particular line that appears most suitable for his future employment.

There are many purely routine jobs on a mine such as surveying, sampling, ventilation measurements and so on, which are often regarded as the legitimate field for the employment of young engineers. Although this type of work can, of course, be performed adequately by a trained mining engineer, it is not wise to keep such a man on it for more than a short time. He is bound to become dissatisfied and will eventually seek other employment if he is not given a wider scope. One way out of this difficulty is to train a group of surveyors or samplers and measurers specifically for these routine jobs, and thus avoid having to employ engineers on them at all. These men are often workmen's sons who are sufficiently intelligent to profit by a limited specialized training in local technical schools, and are content to carry out routine jobs indefinitely.

Turning now to the elementary practical experience which a young engineer should certainly possess, I suggest that this should be gained before he graduates, either during his vacations as is the custom at the Royal School of Mines and elsewhere, or even between his school and college periods. This phase should be well organized to ensure that a lad does have the opportunity of learning how actually to carry out the various tasks in a mine, and, since there are few active metal mines in the U.K., this entails the arranging of visits to mines in Europe, Africa or N. America. Mining companies can (and many do) perform a useful service here by offering students accommodation; paying their passages or possibly paying them labourers' wages whilst working. In this way the mine staff can get a pre-view of next year's crop of graduates, whilst the students have the chance of comparing conditions in different mines.

It is for consideration whether the system current in most Universities at present, in which examinations take place in June each year, is the best. It means that only three months of the summer are available for practical work, which is scarcely enough if students have far to go to reach the mine in which they are to work. The so-called 'sandwich' system in use for the last 30 years in Glasgow University and elsewhere, in which lectures and academic work are limited to the period October–March, leaving six consecutive months available for practical work every year, has many good points, and might well be discussed by a joint committee of the various schools and Universities concerned.

One of the main difficulties in the way of any condensation of courses is the tendency to cram ever more subjects into the curriculum as new developments occur in the industry; and the consequent readjustment of the time that can be allotted to any one subject must be continuously under review by the Boards of Studies of mining colleges. For instance the courses in assaying and surveying at the Royal School of Mines are rightly regarded as being exceptionally detailed and thorough, and young graduates can be considered as fully qualified to carry out routine assaying and surveying competently in their first jobs. If, however, the number of young engineers who start as assayers or surveyors is now very much less than it used to be thirty years ago, and if this number is likely to be still further reduced by reason of the employment of men who are permanently limited to assaying or surveying, then it might be found possible to condense the courses on these subjects and to use the time thus gained for some potentially more useful subject.

Finally, I would urge that the Institution should take a lead in reconciling the demands of the industry as regards the most desired qualifications in young mining engineers with the mining schools' practical difficulties in arranging curricula to suit.

Mr. A. L. Austen: Professor Ritson's excellent summary, particularly of postgraduate training, is very timely. On many mines harassed, overworked senior officials, short of staff, do tend to keep young mining graduates for years on specialized production work, yielding the long-term advantages to be gained from postgraduate training to the necessities of the moment.

It is gratifying to see 'labour relations' stressed as of great importance. Management has been described as securing the whole-hearted and intelligent co-operation of the managed in the fulfilment of a given purpose. Although labour relations cannot be taught in a school, the student should at least be informed of the overriding importance of the subject. The advancing mechanization of mines will not change this fact. In any case, the Commonwealth contains numerous mine-fields worked by native races in which a great part of a supervisor's time is spent in administering labour.

The student should also be at least grounded in a few of the basic precepts of enlightened labour relations by being required to read some good book on the subject, such as Lee H. Hill's *Pattern for good labour relations.* These precepts, he should be told, apply to white and coloured labour alike.

If this were done, the fact that he would find them largely disregarded by mining companies might have less influence on him at an impressionable age.

No. 515 OCTOBER, 1949

BULLETIN OF THE INSTITUTION OF MINING AND METALLURGY

Principal Contents :

FACTORS AFFECTING THE RATE OF FORMATION OF ZINC FERRITE FROM ZINC OXIDE AND FERRIC OXIDE

By D. W. HOPKINS

Published monthly by The Institution of Mining and Metallurgy
Salisbury House, Finsbury Circus, London, E.C. 2

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BULLETIN OF THE INSTITUTION OF MINING AND METALLURGY

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CONTENTS

NOTICE OF GENERAL MEETING

The First Ordinary General Meeting of the Fifty-Ninth Session of the Institution of Mining and Metallurgy will be held, by kind permission, in the Apartments of the Geological Society of London, Burlington House, Piccadilly, London, W.1, on THURSDAY, 20TH OCTOBER, 1949, at 5 p.m.

The following papers (published in the September issue of the *Bulletin*) will be submitted for discussion: THE USE OF THE WET KATA THERMOMETER ON THE WITWATERSRAND, by Mr. P. H. Kitto; and INVESTIGATIONS ON THE PRODUCTION OF ELECTROLYTIC COBALT FROM A COPPER-COBALT FLOTATION CONCENTRATE, by Messrs. H. L. Talbot and H. N. Hepker.

Light refreshments will be provided at 4.30 p.m. for members and visitors attending the Meeting.

The Council invite written contributions to the discussion of papers from members who may be unable to be present at the Meetings of the Institution. The Council reserve the right to edit and condense such contributions.

INSTITUTION NOTES

AUSTRALIAN METALLURGY

A series of four lectures on 'Metallurgical Activities in Australia' will be delivered in the Metallurgy Theatre at the Royal School of Mines by Professor J. Neill Greenwood, D.Sc., M.Met.E., Research Professor of Metallurgy at Melbourne University, on the dates shown below. Each lecture will begin at 5 p.m., and the Chair at the first lecture will be taken by the President of the Institution, Mr. W. A. C. Newman, O.B.E. A cordial invitation to attend these lectures is extended to members of the Institution. The dates and subjects of the lectures are as follows:

Thursday, 3rd November, 1949: 'Historical survey of the development of metallurgical industries in Australia'.

Friday, 4th November, 1949: 'Metallurgy in Queensland'.

Tuesday, 8th November, 1949: 'The lead-zinc industry'.

Wednesday, 9th November, 1949: 'The iron and steel industry'.

MEETINGS IN THE SESSION 1949–1950

General Meetings of the Institution during the Fifty-Ninth Session will be held on the third Thursday in each month from October, 1949, to June, 1950. The dates of the Meetings are as follows:

20th October, 1949
17th November, 1949
15th December, 1949
19th January, 1950
16th February, 1950
16th March, 1950
20th April, 1950
18th May, 1950
15th June, 1950

PAPERS FOR DISCUSSION AT SUBSEQUENT GENERAL MEETINGS

The following programme of papers has been arranged for the November, December and January General Meetings:

November: 'Factors affecting the rate of formation of zinc ferrite from zinc oxide and ferric oxide', by D. W. Hopkins, M.Sc., A.I.M. (published in this issue of the *Bulletin*); 'Some notes on a mechanical concentrator', by T. Haden, *Associate Member*; and 'Gold concentration at the Amalgamated Banket Areas reduction plant', by G. Chad Norris, *Member*.

December: 'Management in industry', by F. G. Hill, B.Sc. (Min.), B.A., *Member* (Presidential Address to the Chemical, Metallurgical and Mining Society of South Africa, and republished in the July, 1949, issue of the *Bulletin*).

January: 'An outline of underground operations at Mufulira Copper Mines, Ltd.', by J. P. Norrie and W. T. Pettijohn.

MEMBERS FROM ABROAD

The Council are always anxious to meet members who come to England after a long absence abroad, and ask such members to make themselves known to the Secretary when attending General Meetings of the Institution at Burlington House.

ELECTION OF MEMBERS OF COUNCIL FOR THE SESSION 1950–51

As previously announced, the new Bye-laws governing the constitution of the Council and its mode of election will affect nominations for the election of Council for the session 1950-51. Nominations for the election of Ordinary Members of Council [*see* Bye-law 27 (iv)] and Overseas Members of Council [*see* Bye-law 28 (iii)] should be sent to the Secretary of the Institution to reach him not later than 1st November, 1949.

ERRATUM

It is regretted that the name of Mr. W. J. Alborno, who recently applied for transfer from Associate Membership to Membership of the Institution, was inadvertently printed in the August issue under the list of candidates for transfer *to* Associate Membership.

M. (OLD STUDENTS))CIATION

ıe Annual Dinner of the Royal ol of Mines Association will be at the Charing Cross Hotel on ›sday, 8th December, at 6.30 for p.m. Tickets, price one guinea, be obtained from the Hon. ›tary, Dr. J. H. Watson, Royal , London, E.C.3.

; NUFFIELD FOUNDATION: LOWSHIPS AND SCHOLAR- ?S FOR THE ADVANCE- ıT OF EXTRACTION METAL- GY

ıring the year 1950 the Nuffield ıdation is again offering a ed number of fellowships and arships with the object of ncing research and training in ıction metallurgy. Citizens of t Britain, the Commonwealth, Empire are eligible to apply.

avelling Fellowships are open to who are members of the teach- staff of Universities and ›ved Schools of Mines and llurgy.

avelling Postgraduate Scholar- are offered to junior members e profession who are graduates ıiversities and approved Schools nes and Metallurgy.

cation Scholarships are offered ning and metallurgical students ıiversities and approved Schools nes and Metallurgy.

plications for fellowships and arships to be taken up in 1950 be received before 1st Novem- 1949, by the Secretary, Nuffield dation, 12 and 13, Mecklen- ı Square, London, W.C.1, from ı full particulars and appli- ı forms can also be obtained.

,BY MEMORIAL AWARDS

ısideration is to be given to the ng of an award or awards from Sir George Beilby Memorial early in 1950, and the adminis- :s (representing The Royal ute of Chemistry, The Society hemical Industry, and The ute of Metals) will welcome nation before 31st December, to assist them in their selection. ards are made to British in- ;ators in science to mark appreciation of records of distinguished work. Preference is given to investigations relating to the special interests of Sir George Beilby, including problems connected with fuel economy, chemical engineering and metallurgy, and awards are made not on the result of any competition but in recognition of continuous work of exceptional merit, bearing evidence of distinct advancement in science and practice. In general, awards are not applicable to workers of established repute, but are granted as an encouragement to younger men who have done original independent work of exceptional merit over a period of years.

In recent years the amount of each award has commonly been 100 guineas.

Communications drawing the attention of the administrators to outstanding work of the nature indicated should be addressed to the Convenor, Sir George Beilby Memorial Fund, Royal Institute of Chemistry, 30, Russell Square, London, W.C.1.

CANDIDATES FOR ADMISSION

The Council welcome communications to assist them in deciding whether the qualifications of candidates for admission into the Institution fulfil the requirements of the Bye-laws. The application forms of candidates (other than those for Studentship) will be open for inspection at the office of the Institution for a period of at least two months from the date of the *Bulletin* in which their applications are announced.

The following have applied for transfer since 8th September, 1949:

To Membership—

Kenneth Christopher Griffith Heath (*London*).

Harry MacConachie (*Johannesburg, Transvaal*).

Wilton Merle McKeown (*Shinyanga, Tanganyika Territory*).

Hugh George McKerrow (*Johannesburg, Transvaal*).

William Arthur Odgers (*Johannesburg, Transvaal*).

James Haworth Taylor (*Johannesburg, Transvaal*).

John Wood Thorburn (*Mount Isa, Queensland, Australia*).

To Associate Membership—

David Jenner Hogg (*Que Que, Southern Rhodesia*).

Ronald Charles Aver Hooper (*Truro, Cornwall*).

George Alfred Rentoul MacIlhatton (*Geita, Tanganyika Territory*).

Kenneth Leslie Peller (*Leeds, Yorkshire*).

The following have applied for admission since 8th September, 1949:

To MEMBERSHIP—

Frank Rundle Beggs (*Sydney, Australia*).

Godfrey Bernard O'Malley (*Melbourne, Australia*).

To ASSOCIATE MEMBERSHIP—

John Francis Curtis (*Discovery, Transvaal*).

Charles Vivian Hickson (*Westcliff-on-Sea, Essex*).

Percy Charles Lockyer (*H.Q. Military Government, Germany*).

John Thomas Pilgrim (*Obuasi, Gold Coast Colony*).

James Arthur Sadler (*Geita, Tanganyika Territory*).

To STUDENTSHIP—

Patrick Denis Coakley (*Gwithian Towans, Cornwall*).

Bruce Jones (*Nkana, Northern Rhodesia*).

George Tombling Whincup (*Manchester, Lancashire*).

NEWS OF MEMBERS

Members, Associate Members, Affiliates, and Students are invited to supply the Secretary with personal news for publication under this heading

Mr. D. J. ADAMS, *Student*, has left England and has taken up an appointment in British Columbia.

Mr. C. T. BLACKWELL, *Associate Member*, has been transferred to the Union platinum mine at Rustenberg, Transvaal, as manager.

Mr. R. L. BLANDY, *Student*, has left England to join the staff of Rhokana Corporation, Ltd., Northern Rhodesia.

Mr. A. W. BOUSTRED, *Associate Member*, is returning to the Gold Coast as assistant manager to Taquah and Abosso Mines, Ltd.

Mr. E. J. D. BROWN, *Associate Member*, has left England to take up an appointment with the Cyprus Mines Corporation.

Dr. D. J. BURDON, *Associate Member*, has gone to Nicosia, Cyprus, to join the Water Supply and Irrigation Department.

Mr. S. B. C. EDWARDS, *Associate Member*, has joined the staff of the Kaduna Syndicate, Northern Nigeria.

Mr. L. T. EVANS, *Student*, has left England to take up employment with Mines Development Syndicate (West Africa), Ltd., Southern Nigeria.

Mr. W. E. EVANS, *Associate Member*, is shortly leaving the Gold Coast to return to United Kingdom.

Mr. H. FAHMY, *Student*, has arrived in England to take a course at Birmingham University.

Mr. D. F. FAIRBAIRN, *Associate Member*, has accepted an appointment on the Ruhuhu coalfields, Tanganyika Territory.

Mr. R. GOLDSMITH, *Student*, has left England to join Amalgamated Banket Areas, Ltd., Tarkwa, Gold Coast Colony.

Mr. V. W. HALL, *Student*, has taken up an appointment with Rhokana Corporation, Ltd., Northern Rhodesia.

Mr. Alan C. HARRISON, *Member*, has resigned from the Burma Corporation, Ltd., and is manager of the fluorspar mill now being erected at Nenthead, Cumberland, by Anglo-Austral Mines, Ltd.

Mr. H. C. HERBERT, *Associate Member*, is shortly leaving India on his return to England.

Mr. H. HOCKING, *Associate Member*, has relinquished his post with The Pahang Consolidated Co., Ltd., and will shortly arrive in England.

Mr. D. A. S. HOLDING, *Associate Member*, has returned to England on leave from India.

Mr. J. L. HOPKINS, *Student*, has left England to take up a position in Southern Rhodesia.

Mr. E. H. JAQUES, *Associate Member*, has returned to the Geological Survey, Nigeria.

Mr. F. L. JAMESON, *Associate Member*, has recently left the Southbury Smelting Works, Ltd., to set up on his own account as a metallurgical chemist and assayer.

[redacted]

G. R. JONES, *Associate* ɪr, has left England for British ı on a six months' visit.

N. R. JUNNER, *O.B.E.*, *M.C.*, ɪr, is away from England until d of next month on a visit to

R. KUTTNER, *Associate Mem-* ıs left Johannesburg to join ıff of De Beers at Kimberley.

R. LANDCASTLE, *Associate* ɪr, has left Tanganyika for six s' leave in England.

G. F. LAYCOCK, *M.C.*, *Mem-* as left England for a short ɔ Australia and expects to at the end of November.

E. LEE, *Associate Member*, has ;ypt for South Africa.

K. LEHMAN, *Associate Mem-* ʼill be returning early next to Australia on leave from

A. W. LEHMANN, *Member*, for Chile early this month, ʼisiting England.

J. McNEILL, *Associate Mem-* now in Uganda in the position ɔector of Mines, Tanganyika.

D. A. O. MORGAN, *Student*, ʼrived at Johannesburg from ıd.

Humphrey M. MORGANS, ɪr, has returned to England , visit to Peru.

H. D. OSBORNE, *Student*, has the staff of Ariston Gold (1929), Ltd., at Prestea, Gold

J. W. PARK, *Member*, has left Africa and is now in Ireland.

T. I. PINER, *Student*, has left ıd to take up a post with ın Tin Dredging, Ltd., Perak.

E. A. B. PRIOR, *Member*, is ing to Southern Rhodesia , visit to England.

Mr. R. F. V. READ, *Student*, has joined the staff of Noranda Mines, Ltd., Quebec.

Mr. C. REYNOLDS, *Associate Member*, has left England for Malaya, on accepting an appointment with The Pahang Consolidated Co., Ltd.

Prof. J. A. S. RITSON, *O.B.E.*, *D.S.O.*, *M.C.*, *Member*, has returned to England from his visit to Northern Rhodesia.

Mr. Stanley ROBSON, *Member*, who is at present on a short visit to Australia, has been elected President of the Society of Chemical Industry.

Mr. J. L. SHEARER, *Student*, has arrived in South Africa from England.

Mr. T. F. B. SPENCER, *Associate Member*, has joined the staff of the Cam and Motor Mines, Ltd., Gatooma, Southern Rhodesia.

Mr. A. TAYLOR, *Student*, has arrived in England from South Africa.

Mr. R. TEALE, *Student*, is on his way to England from Northern Rhodesia.

Mr. J. B. TOMS, *Associate Member*, has left England for Cyprus.

Mr. P. F. WHELAN, *Member*, has left the service of the Imperial Smelting Corporation and has accepted an appointment as Head of Coal Preparation Research for the National Coal Board.

Mr. F. T. M. WHITE, *Member*, has returned to England from Malaya.

ADDRESSES WANTED

D. S. Broadhurst.	R. B. Hicks.
J. A. Cocking.	G. C. Morgan.
N. F. Cox.	A. I. Scott.
E. Dickson.	A. Sloss.

ɔK REVIEW

ɛtraction for the small operator. ed. London : Imperial Chemi- ndustries, Ltd. 103 p.

ı is the second edition of a volume intended to help men ɛd in gold extraction on a scale mainly by the cyanide ɜ.

ıort history of the process is first and is followed by a ı containing very brief details of the principal rock formations in which gold appears. Succeeding sections deal in turn with crushing and grinding, amalgamation, use of strakes, refining of bullion, leaching with cyanide solutions, testing of such solutions, precipitation, filling and dressing of zinc boxes, clean-up, acid treatment, and smelting of the precipitate.

Two chapters are then devoted to

concentration and flotation and to the special methods needed for the treatment of difficult ores such as those, among others, containing pyrite, pyrrhotite, arsenopyrite, stibnite, lead and copper minerals and graphite. The final chapters deal with safety measures in handling cyanide, conversion tables, tank capacities and other useful data and the chemical and physical properties of common minerals.

Elementary, but necessary, precautions are plentifully given and economic limitations at various points are stressed.

The book is very short and so contains but the briefest outlines of the various stages in gold extraction. Nevertheless it is both readable and knowledgeable. It can be recommended as a potted guide.

W. A. C. Newman.

OBITUARY

Henry Edwards died suddenly at Wantage, Berkshire, on 18th September, 1949, at the age of 76. After completing his education in 1888 he studied mining and mechanical engineering under his father, who was then manager of coal, shale and gold mines in New South Wales and Western Australia. He received an appointment with Mr. R. G. Gibbons, of Sydney, in 1898, for four years travelling in New South Wales prospecting and developing properties and then for two years managing gold, silver and bismuth properties. From 1903 to 1904 he was mining engineer at the Empire Mines, North Queensland, and then left to take up the position of millman to the Ashanti Gold Corporation, West Africa. In 1906 he was engaged by the Cia Mineria del Neuquen, Argentine, and from 1908 to 1912 held the position of engineer and manager to La Concordia group of mines in Colombia, and during the same period was consulting engineer to La Socorro Mining Co., and El Transvaal Mining Co.

Mr. Edwards then set up in practice in London as consulting mining engineer to various companies, chiefly for developing gold and platinum fields. He also reported on properties in different parts of the world until 1916, when he joined the Cape Copper Co., Ltd., for special work in India. A year later he was appointed general manager of the Briton Ferry copper works of that Company in South Wales. From 1921 until his retirement Mr. Edwards practised as a consultant.

He was elected to Associateship of the Institution in 1907 and was transferred to Membership in 1918.

Russell Johnston Parker was among those killed in a Canadian air crash on 9th September, 1949, forty miles from Quebec. He was 52 years of age. He received his professional training at the Colorado School of Mines, and on graduating E.M. in 1919 joined the Société Internationale Forestière et Minière du Congo. He began prospecting for and developing alluvial diamond deposits and then worked in the technical office for field headquarters, before being placed in charge of the Muaba mine. He was then appointed acting head of the research department, to which position he returned after a period as manager of the Société Minière du Luebo. He came to the London office of Selection Trust, Ltd., for six months in 1925 before going to Northern Rhodesia to examine their properties, including Roan Antelope, and in August, 1926, was appointed manager of Mineralise Ventures, which later became Rhodesian Selection Trust, Ltd. During the five years Mr. Parker held this position he carried out a geological survey of the Nkana concession and the preliminary development of the Mufulira mine until a separate company was formed in 1930. His work also included development of Chambishi and Baluba mines. At the beginning of 1931 Mr. Parker returned to London as assistant to the managing director of Roan Antelope Copper Mines, Ltd., Mufulira Copper Mines, Ltd., and Rhodesian Selection Trust, Ltd.

In 1942 he took up the appointment of assistant to the President of the Kennecott Copper Corporation, Inc., in New York, and in 1948 became vice-president of the Corporation and president of its subsidiary, the Quebec Iron and Titanium Company.

Mr. Parker was elected to Membership of the Institution in 1933 and served as Member of Council from 1938 to 1941, as Vice-President from 1941 to 1944, and until 1948 had been Member of Council for the U.S.A. With Mr. Anton Gray he contributed a paper to the *Transactions* of the Institution entitled 'Prospecting and geological survey of the Nkana concession, Northern Rhodesia: 1927–1929' (vol. 45, 1935–36). He was a member of the American Institute of Mining and Metallurgical Engineers. He was in 1948 presented with the Medal of Merit of the Colorado School of Mines for achievements in the discovery and development of new orebodies.

ARTHUR DITCHFIELD STORKE, *C.M.G.*, was killed on 9th September, 1949, at the age of 54, in the same air crash near Quebec in which Mr. R. J. Parker lost his life. They were travelling together in company with Mr. E. T. Stannard, also killed, the president of the Kennecott Copper Corporation, which position Mr. Storke was to have taken over at the end of the year.

Mr. Storke attended the Leland Stanford University, California, and Colorado University from 1912 to 1915, and in 1916 was appointed engineer to the American Metal Co., Ltd., of New York. He joined the U.S. Army and served in France from 1917 to 1918, and on demobilization returned to Colorado as manager of the Michigan Copper Co. For five years subsequently he practised as consulting engineer in association with Mr. Carl O. Lindberg in Los Angeles, and in 1926 rejoined the American Metal Co., Ltd., as manager of the Climax mine at Climax, Colorado, and manager of Cape Copper Co., Ltd., Namaqualand, South Africa. In 1929 Mr. Storke took up the appointment of London manager and managing director of Roan Antelope Copper Mines, Ltd., Mufulira Copper Mines, Ltd., and Rhodesian Selection Trust, Ltd., and held these positions for eighteen years. He was also a director of Trepca Mines, Ltd., from 1939 to 1944. In 1947 he became president of Climax Molybdenum Co., New York, and had resigned on accepting the presidency of the Kennecott Copper Corporation. He had also for many years been chairman of the management committee of the Copper Development Association. After the Japanese surrender in 1945, Mr. Storke reported for the British Government on the mining industry of Malaya, and for his services during the 1939–45 war was awarded the C.M.G.

Mr. Storke was elected to Membership of the Institution in 1932 and served as a Member of Council from 1937 until 1940.

R. ARTHUR THOMAS, *O.B.E.*, died on 30th May, 1949, at Polstrong, Cornwall, at the age of 82. His early career after graduating from the Camborne School of Mines was spent in South Africa, where from 1888 to 1891 he was manager of City and Suburban Gold Mining Co., Ltd., Johannesburg, and acted as consulting engineer to several small companies. In 1891 he returned to manage lead and blende mines in Wales for two years, and in 1893 became manager of Dolcoath Mines, Ltd., Camborne, Cornwall. He was appointed to the board and later became managing director, but when the mine had to close down during the depression after the 1914–18 war he became principal of the Camborne School of Mines. He retired in 1933 and became a director of Jantar (Cornwall), Ltd. He served on several national committees set up to enquire into various aspects of the mining industry and was awarded the O.B.E. in 1933. He was consulting engineer to South Crofty for some years, and was well known in Cornwall for his long service on the Council of the Cornish Chamber of Mines, of which he had been Chairman since 1928.

He contributed three papers to the *Transactions* of the Institution: 'Causes and prevention of miners' phthisis' (written jointly with Dr. J. S. Haldane) (vol. 13, 1903–04); 'On crushing and concentration at Dolcoath mine, Cornwall' (vol. 7, 1898–99); and 'Dust in the air and gases from explosives in a Cornish mine (Dolcoath), and the efficacy of methods of dealing with them' (written jointly with W. P. O. Macqueen) (vol. 13, 1903–04).

Mr. Thomas was elected to Membership of the Institution in 1899, and served as Member of Council for the three sessions 1917–1920.

The Council regret to announce the death of MALCOLM FERGUSSON, *Member*, on 14th August, 1949; and THOMAS CAMPBELL SCRUTTON, *Member*, on 27th September, 1949.

ADDITIONS TO JOINT LIBRARY OF THE INSTITUTION AND THE INSTITUTION OF MINING ENGINEERS

Books (excluding works marked *) may be borrowed by members personally or by post from the Librarian, 424, Salisbury House, London, E.C.2.

Books and Pamphlets:

HOWE, James Lewis, and Staff of BAKER & Co., INC. *Bibliography of the platinum metals 1918–1930.* Newark, N.J.: Baker & Co. Inc., 1947. 138 p. 42s.

Government Publications:

FEDERATION OF MALAYA, MINES DEPT. *Annual report on the mining industry for the year 1948,* by A. Bean. Kuala Lumpur: Govt. Printer, 1949. 82 p., map, tabs. 5s. 10d.

INDIA, CHIEF INSPECTOR OF MINES. *Annual report . . . for the year ending 31st December, 1946.* Delhi: Govt. Press, 1949. 211 p., maps, diagrs., tabs., appendix. 8s. 6d.

NORTHERN RHODESIA, WATER DEVELOPMENT AND IRRIGATION DEPT. *Annual report for the year 1948.* Lusaka: Govt. Printer, 1949. 4 p., tab. 1s.

Proceedings and Reports:

AMERICAN INSTITUTE OF MINING AND METALLURGICAL ENGINEERS. *Transactions, vol. 175: Institute of Metals Division 1948.* N.Y.: The Institute, 1949. 924 p., illus., diagrs., tabs., biblios.

AMERICAN INSTITUTE OF MINING AND METALLURGICAL ENGINEERS. *Transactions, vol. 176: Iron and Steel Division 1948.* N.Y.: The Institute, 1949. 552 p., illus., diagrs., tabs.

AMERICAN INSTITUTE OF MINING AND METALLURGICAL ENGINEERS. *Transactions, vol. 178: mining geology 1948.* N.Y.: The Institute, 1949. 556 p., illus., maps, diagrs., tabs., biblios.

INSTITUTION OF MECHANICAL ENGINEERS. *Proceedings (vol. 157) War Emergency Issues, 25–36, 1947*; *Proceedings (vol. 158) January–December, 1948.* London: The Institution, 1949. 530 p., illus., diagrs., tabs; 516 p., illus., diagrs., tabs.

Maps:

Gap, west of fifth meridian, Alberta. Geological survey, map 978A. Scale: 1 in. = 1 ml. Ottawa: Dept. of Mines and Resources, 1949.

Geological map of the Maritime Provinces (New Brunswick, Nova Scotia, and Prince Edward Island). Geological survey, map 910A. Scale: 1 in. = 12 ml. Ottawa: Dep. of Mines and Resources, 1949.

***Pietermaritzburg and environs.* Union of South Africa, Dept. of Mines, Geological Survey, Map. Pretoria, 1948. Scale 1 in. = 0·789 ml.**

INDEX OF RECENT ARTICLES

Classified according to the Universal Decimal Classification. All articles indexed are available in the Joint Library but the current issues of journals are not available for loan.

An extra copy of this Index will be sent on application to members and subscribers who use it for maintaining their own cumulative index.

0 GENERALITIES

016 Bibliographies

016 : 550.83

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622.765 : 622.7–362.1

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622.766 Heavy media separation

622.766 : 549.621.9

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622.79 Roasting, drying, briquetting

622.794 : 553.97

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622.8 HAZARDS, ACCIDENTS

622.81 Gas, dust, explosions

622.811 : 622.474

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622.83 Rock pressure, subsidence

622.831 : 622.281

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622.86 Accidents, rescue work

622.861 : 622.235.38

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622.861 : 622.343(83)

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624.19 Tunnelling

624.19

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661.21

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669 METALLURGY

669.02/.09 Processes and equipment

669.041

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669.2/.7 Non-ferrous metallurgy

669.253

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669.717 : 622.67.078.2

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669.753

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93 HISTORY

93 : 622.73

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Factors Affecting the Rate of Formation of Zinc Ferrite from Zinc Oxide and Ferric Oxide*

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A paper published on 13th October, 1949, by the Institution of Mining and Metallurgy, to be submitted for discussion at the November General Meeting.

Synopsis

The factors affecting the rate of formation of a compound or compounds between zinc oxide and ferric oxide, and their composition, have been studied at temperatures between 600°C and 1,400°C, by heat treatment of mixtures containing between 5 per cent and 95 per cent zinc oxide. Quantitative assessment has been made possible by a method of chemical analysis designed to determine the state of the two oxides in all raw and reacted mixtures. It has been found that the rate of formation of the only compound conclusively proved to exist—zinc metaferrite ($ZnO.Fe_2O_3$)—is highly sensitive to temperature and profoundly affected by the temperature of preparation of the ferric oxide used. Ferric oxide, in reaction with zinc oxide, is either very active or relatively inert, depending on whether the temperature of preparation is below or above 678°C. The zinc oxide-ferric oxide ratio has practically no effect on the rate of combination when ferric oxide prepared at low temperature is used and only a small effect when the ferric oxide is prepared at high temperature. The low-temperature form of ferric oxide is rapidly transformed to the high-temperature form when heated at over 678°C. Zinc metaferrite is formed from the powdered oxides at temperatures over 600°C and is capable of dissolving a large amount of ferric oxide and a small amount of zinc oxide above 1,300°C. The apparent existence of compounds with more zinc oxide than the meta-ferrite is considered to be due to the difficulty of dissolving the free zinc oxide from the diffuse and enveloping structure of heat-treated mixtures. This is especially the case when the mixtures contain between 70 per cent and 90 per cent zinc oxide and are heated to over 1200°C. A summary of published data on the properties of zinc metaferrite is given and its effect on the stability of ferric oxide at high temperature determined.

Introduction

Before zinc can be extracted from the sulphide minerals wurtzite, sphalerite, and marmatite (the principal constituents of a zinc blende flotation concentrate), it is necessary to convert them to oxides, with or without a proportion of sulphate, according to the nature of the subsequent operation. This has to be done because zinc sulphide is not reducible by carbon and neither do the sulphide and oxide react, as in the case of copper and lead, to produce metal.

After oxidation the zinc oxide and sulphate can be leached out with sulphuric acid, or the sulphur-free oxide can be reduced with carbon, in the form of coke breeze or anthracite duff, at about

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1100°C to produce zinc. Zinc can be deposited from the leach liquor by electrolysis.

Oxidation is carried out by roasting the blende concentrate in air to temperatures above 700°C, under conditions conducive to the production of either oxide and sulphate or oxide alone. For sulphate production either a Herreshoff roaster or the 'flash roasting' process may be used, operating with a high SO_2 content in the furnace gases—either at a low temperature, as in the Herreshoff, or ample exposure to SO_2 at about 700°C on the bottom hearths of the flash-roasting furnace. Sulphur-free oxide may be produced from either of these furnaces if the SO_2 content of the gases is kept down by good draughting, and the speed of operation maintained by raising the Herreshoff temperature to about 850°C. The temperature in the combustion zone of the flash roaster is about 950°C at all times. Completely-oxidized material is also produced in large quantities by down-draught sintering on Dwight-Lloyd machines from a charge consisting of about five parts of return sinter mixed with one part of raw blende. The rate of production is high, owing to the intensity of combustion in the ignited mixture and the high temperatures reached during the operation.

The flotation concentrate which is the raw material for all these operations contains impurities, the nature of which will depend on its origin. The principal impurity is iron sulphide, either as the free sulphide (pyrite) or in solid solution with zinc sulphide as marmatite—2ZnS.FeS. Lead sulphide, copper sulphide, and silica are also present in smaller amounts. Table I indicates the percentage range of composition which may be expected.

TABLE I

Concentrate	Zn	Fe	Cu	S	Pb	Cd	SiO_2
Mexican	65·81	0·64	—	32·56	—	—	0·30
Australian	52·60	8·50	0·12	32·20	1·52	0·18	0·70
Canadian	52·60	3·70	0·69	31·60	3·22	0·18	1·20
Austrian	37·39	21·69	1·01	33·00	—	—	2·42

During the roasting or sintering operation these impurities will react with each other, either as the sulphides to form low melting-point mixtures, or as the oxides to form stable compounds which may have properties other than those desired in the furnace product. The liquid sulphides will delay the removal of sulphur and obstruct the mechanism of the furnace by the formation of agglomerates, while the oxide compounds may cause some of the zinc to be insoluble in the leach liquor or may reduce the efficiency of recovery by reduction and distillation.

The silica present is too small in amount to have any material influence on the zinc extraction and is in any case almost invariably found in combination with the lead oxide as glassy masses or 'stringers' in the roasted material. Cadmium, lead, and copper oxides do not form solid solutions or compounds with zinc oxide under these conditions but ferric oxide is capable of combining with zinc oxide to form zinc metaferrite ($ZnO.Fe_2O_3$)[1] which is insoluble in dilute mineral acid and is suspected of having an adverse effect on the recovery of zinc by distillation. This latter effect is due to catalysis of carbon monoxide breakdown to carbon dioxide and carbon, and the reoxidation of zinc vapour before condensation. It has been shown also that the amount of combined zinc oxide in a roasted pyritic blende is only one-third of that obtained when the same amount of iron is present as marmatite[2].

Zinc metaferrite has been synthesized by heating a stoichiometric mixture of the component oxides in fused boric acid[3], extracted from the insoluble residue after the leaching of roasted ferruginous blende[1, 2], and identified with the naturally-occurring mineral Franklinite ($Zn(Fe)O,Fe(Mn)_2O_3$). The minimum temperature of formation was given as from 700°–1200°C by early workers, but later work led to the recognition of 580°–600°C as the most probable minimum for formation from the powdered oxides[4]. X-ray examination of this compound showed it to have a disordered lattice, which became ordered on heating to 700°C[5]. The existence of this disordered compound has been noted[6] in the mixed hydroxide precipitate from aqueous solution, but the same hydroxides did not combine below 600°C when dried separately before heating, nor did the precipitated compound become ordered below 700°C.

Claims have been made for the existence of compounds other than the metaferrite[7, 8, 9, 10], but the data are scanty and contradictory. X-ray analysis is complicated by the similarity between the spectra of the ferrites and magnetite, which is certainly formed in samples having an excess of ferric oxide when heated to temperatures exceeding 1200°C. No account seems to have been taken of the possibility of solid solutions being formed between the original oxides, or between the metaferrite and either of them. A consideration of Table II, which contains analyses of the compounds claimed to have been found, will explain to some degree the ease with which figures over a wide range may approximate to simple molecular ratios. In all cases the metaferrite was found with the more complex compounds.

TABLE II

Compound	Zinc Oxide *Per cent*	Ferric Oxide *Per cent*
$2ZnO.3Fe_2O_3$	25·36	74·64
$ZnO.Fe_2O_3$	33·76	66·24
$4ZnO.3Fe_2O_3$	40·54	59·46
$5ZnO.3Fe_2O_3$	46·08	53·92
$2ZnO.Fe_2O_3$	50·47	49·53
$8ZnO.3Fe_2O_3$	57·56	42·44
$3ZnO.Fe_2O_3$	60·47	39·53
$4ZnO.Fe_2O_3$	67·10	32·90

(1) See bibliography at the end of the paper.

The solubility of zinc metaferrite in mineral acids, its decomposition by heating with basic oxides or carbon, and the prevention of ferrite formation when roasting blende, were investigated at the same time as the possibility of complex compound formation. The metaferrite was found to be insoluble in the common mineral acids of strength less than 15 per cent by weight in water and to be decomposed by heating with lime or magnesia at about 700°C[11]. Addition of these bases to the roasting material at almost the end of the sulphur elimination period resulted in the formation of calcium and magnesium ferrites and left the zinc oxide in the free state. Whether zinc ferrite was more or less easily reduced than zinc oxide was the subject of contradictory statements until Oldright[12] showed that zinc metaferrite dissociated in the presence of carbon to the constituent oxides at 650°–700°C. The ferric oxide was then completely reduced to metal before reduction of zinc oxide began.

The problem of the insolubility of zinc metaferrite in dilute mineral acids was overcome in the Tainton[13] process by separating the compound magnetically from the zinc oxide, dissolving it in the 28 per cent sulphuric acid electrolyte and neutralizing the liquor with the non-magnetic portion. A counter-current process is now in general use, in which the acid liquor, produced as a result of solution of the first-stage residue in strong electrolyte, is neutralized by a fresh batch of calcine, to leave further residue suitable for solution in electrolyte.

As a result of the work already outlined and of an extensive programme on the use of zinc metaferrite as a catalyst in the oxidation of nitrogen[14], a considerable number of data has been obtained on the properties of the compound. These are summarized in Table III.

TABLE III

PROPERTIES OF ZINC METAFERRITE ($ZnO.Fe_2O_3$)

Colour: Deep orange powder. Reddish brown by transmitted light. Black in mass with metallic lustre.

Refractive index: $7_{Li} = 2{\cdot}31 \mp 0{\cdot}02$ and $2{\cdot}36 \mp$

Specific Gravity: 5·09 to 5·290.5·322 and 5·349 calculated from lattice parameter.

Hardness: 5·5 on Moh's scale.

Unit Lattice: $a_0 = 8{\cdot}423Å \mp 0{\cdot}01Å$ (mean of 7 values.)

Melting Point: 1590°C. (Checked, the reported value of 1720°C is inaccurate.)

Heat Capacity: 0·73j/gm/°C. (50° — 1025°C.)

Coefft. of Thermal Expansion: 99×10^{-7} (25°—850°C.)

Magnetic Properties: Ferromagnetic, Curie Point 61°C.

Crystal Form: Cubic octahedra. Recrystallization temperature, 700°C.

Solubility: Insoluble in dilute mineral acids if prepared at over 700°C. Hygroscopic and soluble if prepared between 600° and 700°C. All forms soluble in 20 per cent sulphuric or hydrochloric acid in water.

Data on the proportion of insoluble zinc oxide in roasted or sintered ferruginous blendes indicate that the quantity depends on :

(1) the quantity of iron sulphide in the raw blende ;
(2) the form in which this is present ;
(3) the maximum temperature attained in the roasting or sintering ; and
(4) the length of time at the maximum temperature.

Obviously the particle size of the raw blende will have some effect, but in the examples studied there were no data on this factor. Since all were flotation concentrates it is reasonable to assume a roughly constant value.

Zinc metaferrite formation is increased by the presence of iron sulphide as marmatite, by a high temperature, and by a long time of roasting. The effect of temperature is much greater than that of time, as is shown by the fact that almost 100 per cent of the iron oxide forms ferrite in the sintering process when the maximum temperature of about 1300°C is maintained for only about five minutes, whereas after about six hours in a reverberatory furnace at 850°C, only about 35 per cent is combined with zinc oxide. For this reason, zinc oxide for lithopone manufacture or any process involving leaching is still prepared by the more expensive reverberatory roast.

The results of such investigations as have been carried out on the rate of formation of zinc ferrite are incapable of general application, generally owing to a lack of knowledge of the state of the reactants, or of the precise conditions of experiment. This is particularly the case where the raw material has been a sulphide concentrate, as the uncontrollable variables involved in the oxidation process are added to those affecting ferrite formation.

Scope of Investigation

While it is hoped to obtain a solution to the problem of the mechanism of zinc ferrite formation during blende roasting as an ultimate goal, it is obvious that the starting point has to be such that all materials are in defined and reproducible states and that experimental conditions allow the highest accuracy of control. By the synthesis of information obtained from a series of investigations into simple systems of pure materials, it should be possible to predict the behaviour of a given blende from the results of one or two elementary tests.

For these reasons the present work has been carried out on pure zinc and iron oxides, prepared and reacted with each other under carefully-controlled conditions. The pure sulphides may appear to offer a better starting point, but the effects of residual sulphur and the probable variation in the temperature of preparation of the oxides as a result of non-uniform roasting are eliminated by the method chosen.

The experimental work has been carried out in the following ways :

(1) Determination of the effect of temperature on the rate of combination of zinc oxide and ferric oxide between 600°C and 1300°C.

(2) Determination of the effect of the temperature of formation of the ferric oxide on the rate of combination between 600°C and 1300°C.

(3) Determination of the effect of the zinc oxide-ferric oxide ratio on the rate of combination and on the composition of the compounds formed.

Experimental Procedure

The preparation of ferric oxide by the ignition of ferrous oxalate was not possible, owing to the high and uncontrollable temperature in the final oxidation process. Ferric oxide made from ferrous oxalate was therefore dissolved and precipitated from acid solution before drying at 150°C and storing in a well-stoppered glass bottle. Samples were taken and ignited at the specified preparation temperature for four hours. Uniformity of mean particle size and size distribution was ensured by grinding, measurement on a Fisher sub-sieve sizer, and reaction with zinc oxide under standard conditions. The zinc oxide was purified in the same way by solution and precipitation and the stock of oxide was ignited at 500°C. There was no need to store the hydroxide, as the ignition temperature had no effect on the reactivity below 950°C, and then only as a result of slight sintering. The particle size, apart from loose clots, was remarkably uniform without crushing.

Table IV

Ferric Oxide prepared at:	550°C	650°C	850°C	1050°C
Mean Particle Size:	4·72μ	4·84μ	5·00μ	4·68μ

Analysis: From 99·64 per cent Fe_2O_3 + 0·32 per cent FeO to 99·96 per cent Fe_2O_3.

Zinc Oxide Mean Particle Size: 1·13μ.

Analysis: Pure, but for a trace of carbon dioxide.

Mixtures for 'rate' experiments were prepared by tumbling the weighed oxides together in a rotating glass bottle for 30 minutes under conditions which eliminated grinding. Mixtures for compound formation were made in the same apparatus, but with the addition of steel rods, so as to reduce the mean particle size to 0·5μ. The former were placed in the furnace container as loose powders, and the latter pressed into cylinders 0·5 in. in diameter by 0·5 in. high. Heating-up periods were reduced to 1–2 per cent of the total time of any treatment by varying the size of sample and using furnaces of very large heat capacity relative to the charges and containers.

The loose powders were cooled rapidly on removal from the furnace by spreading on a polished steel slab. Pressed cylinders were cooled by an air blast at first, but it was later found that they could be quenched in water without disintegration. Wire-wound resistance furnaces were used for temperatures up to 1000°C and a furnace with ' Globar ' elements from 1000 to 1400°C. The former were controlled mechanically to within 5° at 1000°C, but the latter had to be manually controlled to keep within 10° at 1400°C. Fused silica containers were used up to 1000°C, but above this temperature platinum dishes had to be used. The slight reducing effect of platinum on samples rich in ferric oxide, as found by White[15], was noted and the affected portions of the samples were discarded.

Mixtures containing between 5 per cent and 95 per cent zinc oxide, the balance being ferric oxide, were prepared and heated over this wide range of temperature in order to simulate all likely conditions of sintering and roasting, and to give the maximum opportunity for the formation of compounds. The ' rate of combination ' was determined by analytical estimation of the insoluble —i.e. combined—zinc oxide in the samples after definite periods of heating at a fixed temperature. The zinc oxide content of the residue after treatment in a solution intended to dissolve free zinc oxide, is divided by the original content of zinc oxide in the sample and multiplied by 100 to give the ' percentage combination '.

Sampling and Analysis

All raw and reacted mixtures were sampled and analysed. In the early stages the ferrous oxide was determined in all samples, but this was later confined to samples prepared at temperatures over 1200°C. The samples were crushed in an agate mortar and dried at 105°C, before storing in a desiccator. Between crushing each sample, the pestle and mortar were thoroughly wiped with a wet cloth to remove adherent particles which were not affected by dry cleaning.

Total Iron and Zinc Determinations

For this purpose 0·5 g. of sample was treated with 50 per cent H_2SO_4 and 50 per cent HCl and simmered until solution was complete. The diluted liquor was then boiled and filtered, the residue being washed and redissolved, before separating the iron from the zinc by ammonia precipitation.

The iron in the precipitator and the zinc in the solution were then determined in the normal way by standard methods.

Solubility in Alkaline Ammonium Chloride Liquor

For this 0·5 g. of the sample was treated with liquor comprised of the following constituents: 5 g. of NH_4Cl, 50 ml. of distilled water and 20 ml. of NH_4OH (sp. gr. 0·880). The sample was allowed to soak overnight in this liquor. (A longer time of leaching had detrimental effects on the determination and a slow attack on the ferrite occurred.) The leach liquor was then filtered through a weighed Gooch crucible and washed thoroughly with distilled water containing a few drops of NH_4OH. All the insoluble matter was transferred to the pad, with the aid of a ' policeman ' if required.

The crucible was then washed well to remove all the NH_4Cl, removed, dried at 110°C for half an hour, cooled in a desiccator and weighed. The increase in weight represented the insoluble matter, which was composed of either pure zinc metaferrite, or a mixture of ferrite and excess ferric oxide, according to the original composition of the sample.

Soluble Zinc

The filtrate from the above test, which was collected in a clean Buchner flask, was washed into a conical beaker, 100 ml. of saturated H_2S water added, the solution diluted to 500 ml., and heated almost to boiling point. Any soluble zinc was precipitated as the sulphide.

For this determination HCl was added until the ZnS just dissolved and 7 ml. added in excess, before titrating almost at the boiling point with standard potassium ferrocyanide solution, using uranyl acetate as an external indicator.

Ferrous Iron

For samples containing 80–95 per cent Fe_2O_3 a 0·5-g. sample was used, for 50–80 per cent Fe_2O_3, 1·0 g., and for 5–50 per cent material the maximum available.

The weighed sample was transferred to a clean 250 ml. conical flask fitted with a two-hole rubber stopper, through which inlet and outlet tubes passed for the passage of carbon dioxide. Ten ml. of 50 per cent H_2SO_4 and 50 ml. of 50 per cent HCl, both made with air-free distilled water, were added and the flask placed on a hot plate. The CO_2 flow was started as soon as the acids were added and the heating continued until solution was complete. The flask and contents were then quickly cooled in a stream of cold water, with the CO_2 flowing until ready for titration. When cold the CO_2 was cut off and the stopper and fittings removed and washed with distilled water into the flask. The volume was increased by the addition of about 120 ml. of distilled water and titrated immediately with N/10 potassium dichromate, using potassium ferricyanide as an external indicator. (Diphenylamine had been tried here as an indicator, but was found to be seriously in error in that it gave an end point when the external indicator did not give one.)

Standardization

The standard solutions were prepared under the same conditions as those above, in order that the factors for iron and zinc should be of the maximum accuracy.

Reporting of Results

(A) If ferrous iron was determined, the results were reported as follows:

(1) Total FeO (2) Total Fe_2O_3 (3) Total ZnO	} = 100 per cent	Matter insoluble in leaching solution. }	(1) Total FeO (2) Total Fe_2O_3 (3) ZnO (Original less soluble)
(4) Soluble ZnO		(The above reported as percentage of insoluble residue)	

(B) If ferrous iron figures were not required:

(1) Total Fe (as Fe_2O_3) (2) Total ZnO	= 100 per cent	Matter insoluble in leach liquor	(1) Insoluble Fe (as Fe_2O_3). (2) Insoluble ZnO.
(3) Soluble ZnO			

When the samples were extremely small, the iron was determined by $K_2Cr_2O_7$ titration, but the zinc was determined gravimetrically by the mercuric thiocyanate method. Ferrous iron was determined by solution of the sample in 50 per cent HCl, in the presence of a known excess of dilute standard $K_2Cr_2O_7$ solution, separation of the iron with NaOH solution, filtration and subsequent colorimetric determination in an aliquot portion of unreduced $K_2Cr_2O_7$, using diphenyl carbazide.

Experimental Results

As preliminary trials had shown that there was a difference in the reactivity of ferric oxide with zinc oxide, according to whether the temperature of preparation was above or below approximately 700°C, the experiments under Section (1)* were carried out in duplicate, using two varieties of ferric oxide, one prepared at 650°C and one prepared at 850°C. In Section (2) the actual value of this transition temperature was determined, together with an indication of a further fall in reactivity at a higher temperature. The first part of Section (3) was derived from the use of a number of different mixtures in all of the experiments under (1) and (2). For the second part, it was necessary to examine a number of the samples microscopically as well as analytically, since the analytical data alone were not conclusive.

Graphs 1 and 2 illustrate the effect of temperature of reaction on the rate of combination of zinc oxide and the two kinds of ferric oxide. Until the effect of zinc oxide-ferric oxide ratio is discussed under Section (3), the groups of curves representing one type of ferric oxide are dealt with as if a mean curve was being considered. The exceptions are those curves which occur in a number of graphs and have a common peculiar form. The first of these is Curve 3 of Graph 2.

Graphs 3 and 4 show the result of examination of the rate of combination at one temperature, but for a period sufficiently long for the reaction to be substantially complete. The great difference in reaction velocity between the two kinds of ferric oxide is apparent and the form of the curve previously noted is repeated in Curves 2 and 3 of Graph 4.

In Graphs 5 and 6 the temperature of reaction is at the lower end of the range and a new factor appears, the failure of one type of ferric oxide to approach complete combination in any foreseeable

*See (1), (2) and (3) under the heading 'Scope of Investigation', pp. 5 and 6.

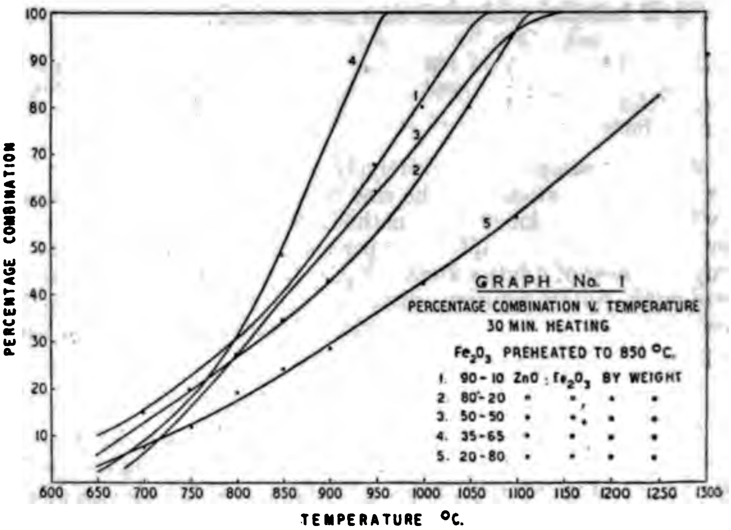

GRAPH No. 1
PERCENTAGE COMBINATION V. TEMPERATURE
30 MIN. HEATING
Fe_2O_3 PREHEATED TO 850 °C.
1. 90-10 ZnO : Fe_2O_3 BY WEIGHT
2. 80-20 " " " "
3. 50-50 " " " "
4. 35-65 " " " "
5. 20-80 " " " "
PERCENTAGE COMBINATION
TEMPERATURE °C.

Fig. 1.

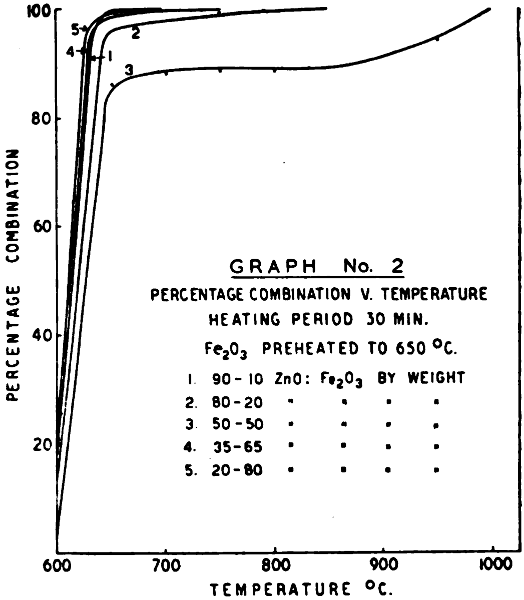

GRAPH No. 2
PERCENTAGE COMBINATION V. TEMPERATURE
HEATING PERIOD 30 MIN.
Fe_2O_3 PREHEATED TO 650 °C.
1. 90-10 ZnO: Fe_2O_3 BY WEIGHT
2. 80-20 " " " "
3. 50-50 " " " "
4. 35-65 " " " "
5. 20-80 " " " "
PERCENTAGE COMBINATION
TEMPERATURE °C.

Fig. 2.

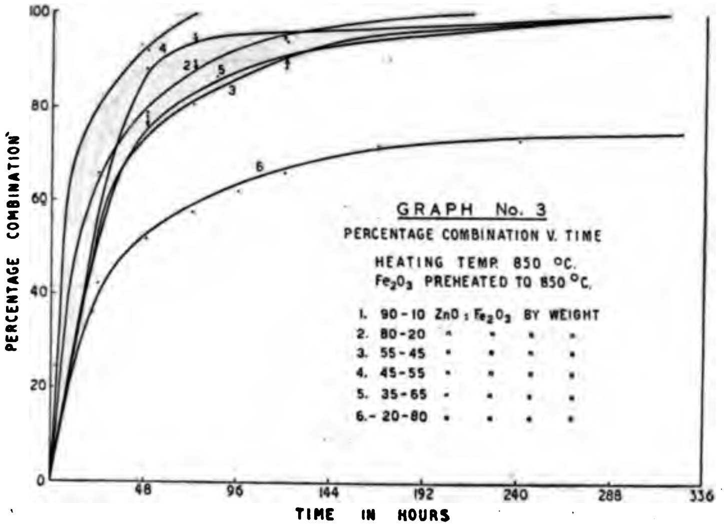


FIG. 3.

period. The velocity of reaction of the low-temperature ferric oxide appears to be the same at 650°C as it was at 850°C.

The effect of a higher temperature of reaction is shown in Graph 7, where the ferric oxide prepared at 650°C has reacted completely in a few minutes and the '850' ferric oxide curves have indications of being similar in form to those mentioned in Graphs 2 and 4.

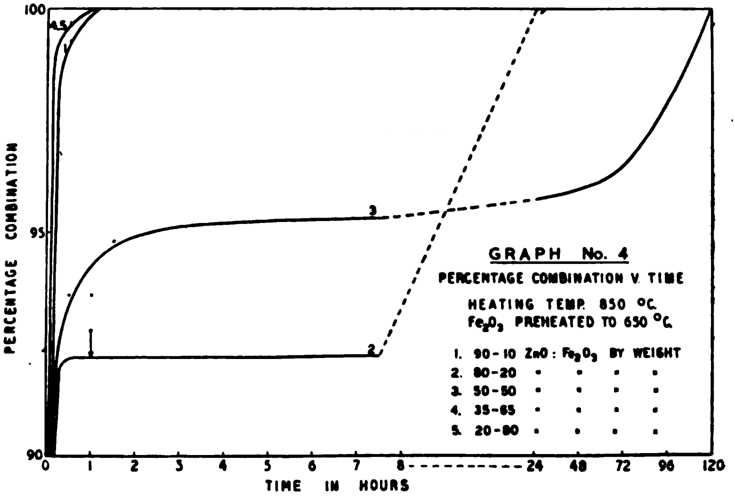


FIG. 4.

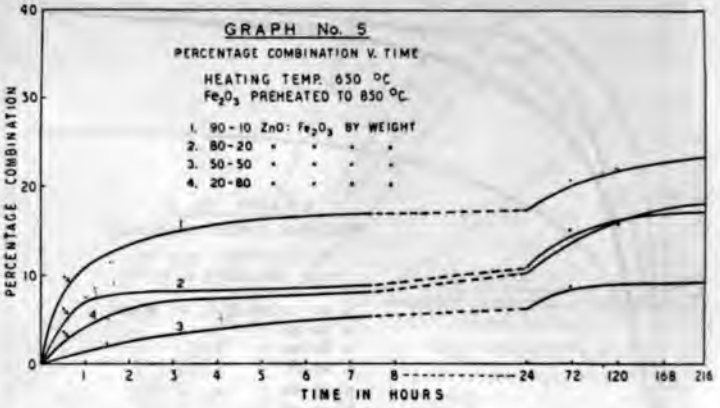


Fig. 5.

Graph 8 is a preliminary determination of the effect of the temperature of preparation of the ferric oxide on the rate of reaction at a constant temperature. The falling off in reactivity above 700°C and again above 1200°C confirm the previous approximation for the lower temperature, but the highest accuracy is only attained in Graph 9, where a combination of temperature of preparation and temperature of reaction produce a very sharp break in the rate curves.

Whereas the samples of ferric oxide had previously been prepared by heating at the requisite temperature for four hours, there was

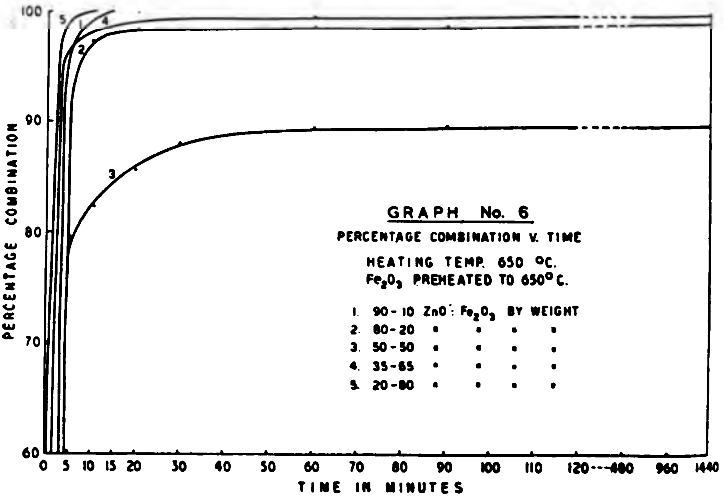


Fig. 6.

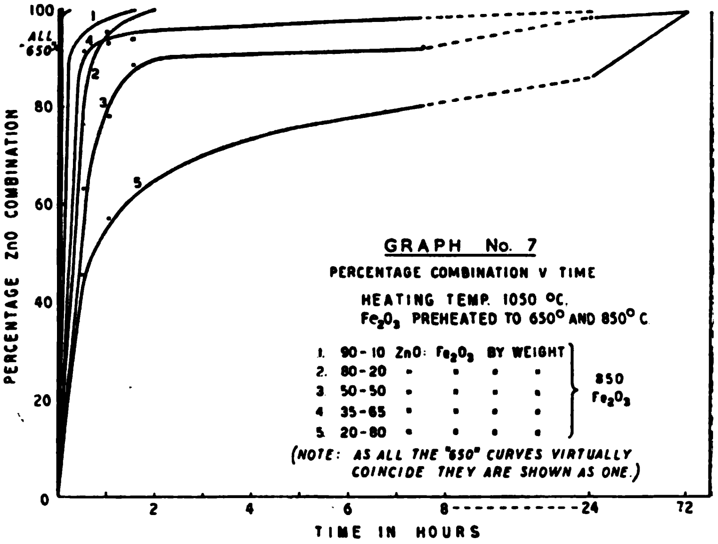

GRAPH No. 7
PERCENTAGE COMBINATION V TIME
HEATING TEMP. 1050 °C.
Fe2O3 PREHEATED TO 650° AND 850° C.
1. 90-10 ZnO: Fe2O3 BY WEIGHT
2. 80-20
3. 50-50
4. 35-65
5. 20-80
850 Fe2O3
(NOTE: AS ALL THE "650" CURVES VIRTUALLY COINCIDE THEY ARE SHOWN AS ONE.)
ALL 650°
PERCENTAGE ZnO COMBINATION
0
20
40
60
80
100
2
4
6
8
24
72
TIME IN HOURS

FIG. 7.

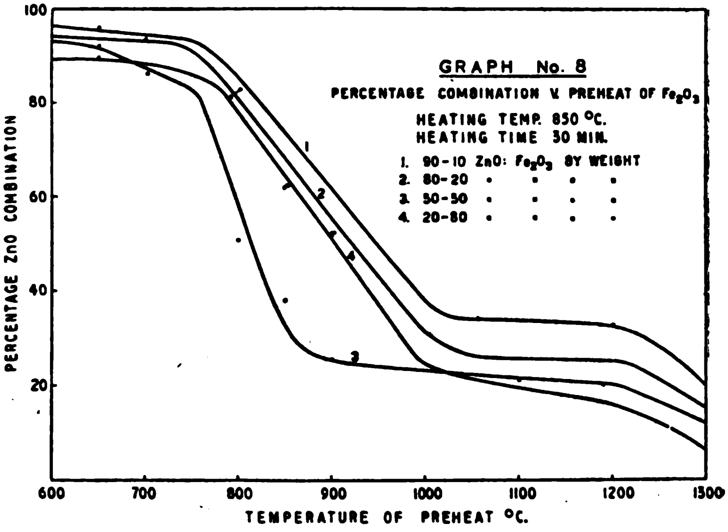
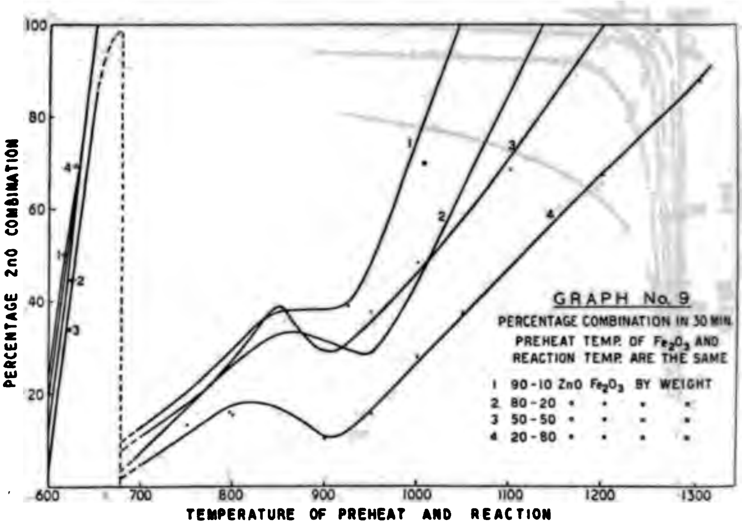
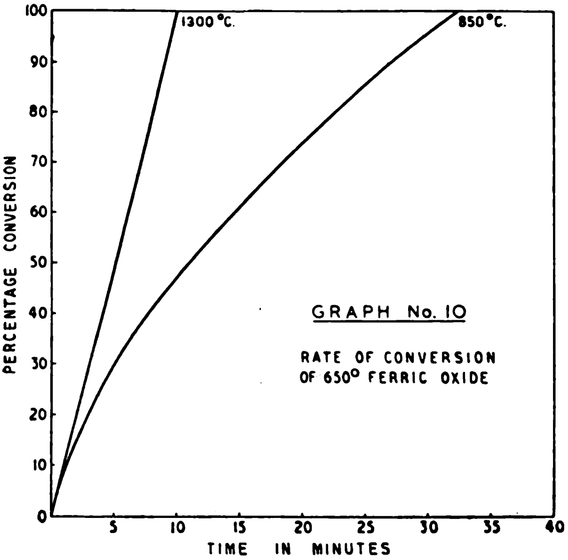

GRAPH No. 8
PERCENTAGE COMBINATION V. PREHEAT OF Fe2O3
HEATING TEMP. 850 °C.
HEATING TIME 30 MIN.
1. 90-10 ZnO: Fe2O3 BY WEIGHT
2. 80-20
3. 50-50
4. 20-80
PERCENTAGE ZnO COMBINATION
20
40
60
80
100
600
700
800
900
1000
1100
1200
1300
TEMPERATURE OF PREHEAT °C.

FIG. 8.



GRAPH No. 9
PERCENTAGE COMBINATION IN 30 MIN.
PREHEAT TEMP. OF Fe_2O_3 AND
REACTION TEMP. ARE THE SAME
1 90-10 ZnO Fe_2O_3 BY WEIGHT
2 80-20 " " " "
3 50-50 " " " "
4 20-80 " " " "
PERCENTAGE ZnO COMBINATION
TEMPERATURE OF PREHEAT AND REACTION

Fig. 9.



1300 °C.
850 °C.
GRAPH No. 10
RATE OF CONVERSION
OF 650° FERRIC OXIDE
PERCENTAGE CONVERSION
TIME IN MINUTES

Fig. 10.

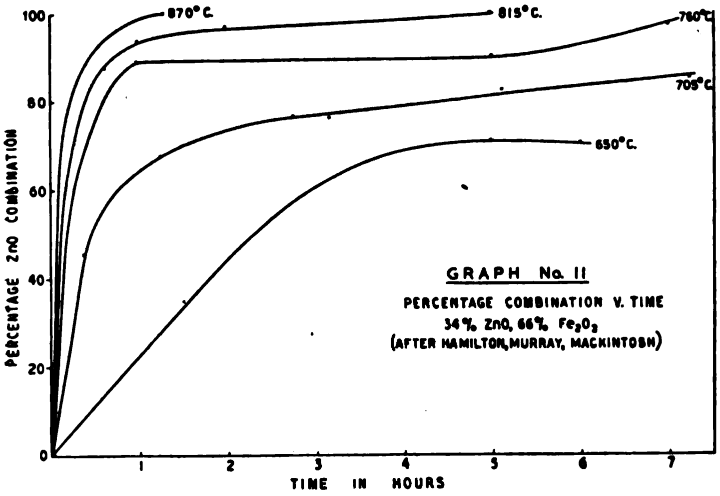


FIG. 11.

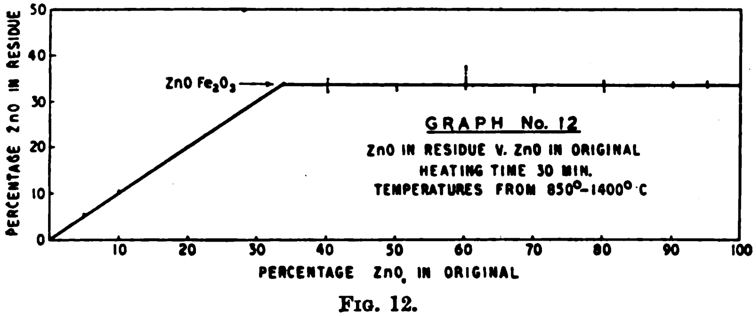


FIG. 12.

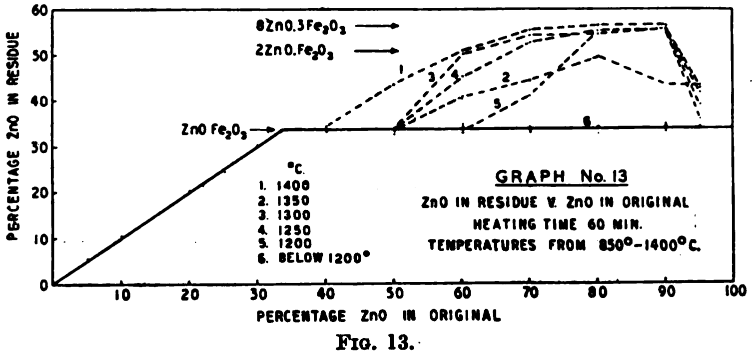


FIG. 13.

no exact knowledge of the rate of change of one form into another. Graph 10 is the result of taking ferric oxide prepared at 650°C, heating it at the temperatures given for various periods and then reacting with zinc oxide under standard conditions. Knowing the reactivity of the original material and of the forms stable at the two temperatures used, it was possible to determine the degree of conversion from the reactivity of the treated material.

As can be seen the rate is such as to affect the uncombined ferric oxide in a mixture after only a few minutes' exposure to the temperature of reaction, provided that this is sensibly higher than the temperature of preparation. Graph 11[16] is drawn from published work on blende roasting and is of interest in that the 760° curve is of the unusual form previously mentioned.

Graph 12 and Graph 13 summarize the results of the very large number of analyses made in order to detect compound formation over the whole range of mixture compositions used. Graph 12 indicates very clearly that the initial compound to be formed is the metaferrite ($ZnO.Fe_2O_3$) containing 33·76 per cent zinc oxide.

Graph 13 appears to provide support for the formation of $2ZnO.Fe_2O_3$ and $3ZnO.3Fe_2O_3$, but there are weaknesses in the argument based on these data which are discussed later.

Graph 14 is compiled from data incidental to the attempts at compound formation at the highest temperatures. Zinc oxide is seen to have a stabilizing effect on ferric oxide similar to that of magnesium oxide. It should be noted that in the zinc oxide-ferric

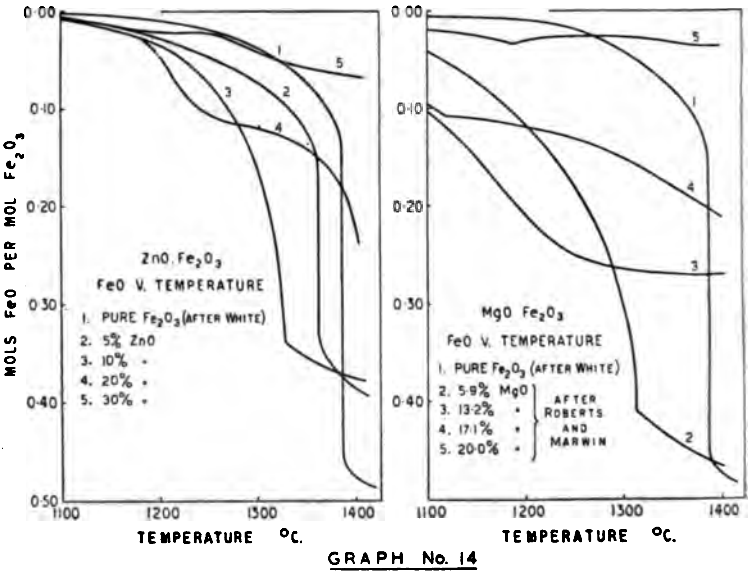
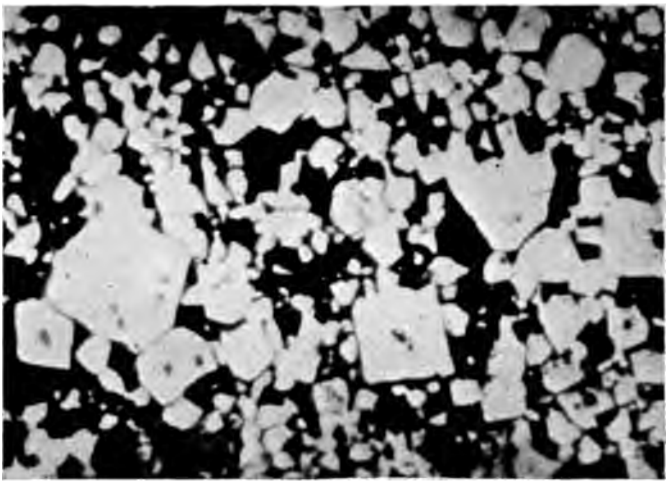
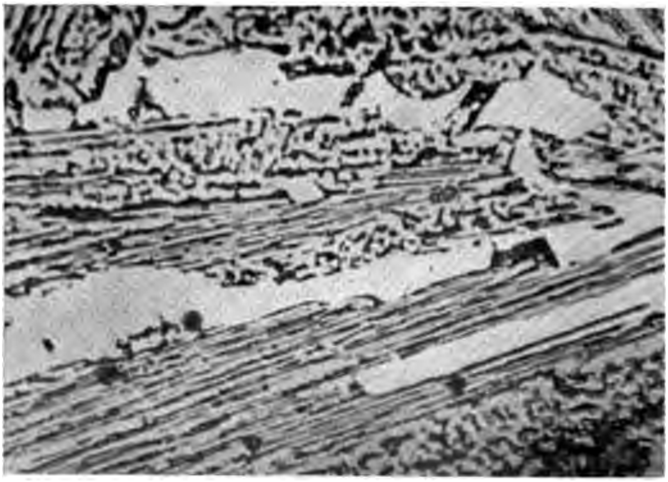


Fig. 14.



Heavily etched × 250

Fig. 15.—Zinc ferrite.



Etched × 500

Fig. 16.—Complex structure.

oxide graph, the ferric oxide content of any sample is taken as 100 per cent at below 1200°C and not as the actual content, as in the work of Roberts and Merwin[17].

Figs. 15 and 16 (Plate I) are included to show the structure of the pure metaferrite and the complex nature of the high-temperature products containing between about 86 per cent and 90 per cent zinc oxide. The large white masses in both cases are the metaferrite, and in Fig. 16 the remainder consists of a lamellar aggregate of white metaferrite and darker zinc oxide solid solution.

Discussion of Results

Under Section (1), the results from which Graphs 1–7 were prepared indicate that the reaction

$$ZnO + Fe_2O_3 = ZnO.Fe_2O_3$$

has a large temperature coefficient, irrespective of the character of the ferric oxide. The coefficient is naturally greater for the more active form, as is shown by the fact that the percentage combination in 30 minutes is increased from 20 per cent to 90 per cent in 50° (600–650°C) for the low-temperature form, and only from 40 per cent to 55 per cent in 50° (850–900°C) for the high-temperature variety (Graphs 1 and 2).

Comparison of the mean rates in Graphs 3 and 4 shows that the '650' ferric oxide reacts with an initial rate which is about 100 times that of the '850' variety.

The general form of both sets of curves is indicative of diffusion control of the rate of combination, but there are curves in Graphs 2 and 4 which appear to indicate an initial rapid reaction, followed by an interval of time or temperature over which no further combination takes place, and then by renewed reaction at a slower rate. For example, in Graph 2, Curve 3, there is no increase in the total combination above 650°C until the temperature exceeds 900°C. In the same way, Curve 3 of Graph 4 appears to indicate a suspension of activity between about 2 and 72 hours after the commencement of heating. It is interesting that Graph 11, constructed from published data, contains a curve of this unusual form (760°).

Reference to Graph 10 will assist the explanation advanced to account for this phenomenon. In this graph it can be seen that the rate of transformation of the low-temperature form of ferric oxide into that stable at the temperature of reaction takes place at about the same rate as the reaction to form the metaferrite. Thus, during the heat treatment of a mixture of zinc oxide and ferric oxide prepared at a temperature lower than that of the heat treatment, while part of the ferric oxide is combining with the zinc oxide, the rest is being transformed into the form stable at that temperature. If this latter form is less reactive than the original, then, after an initial phase of rapid reaction, a stage will be reached at which combination will stop, until the reactants diffuse through this layer of product to continue combination at a much slower speed.

It will be possible by increasing the temperature of reaction sufficiently to raise this speed of diffusion to a point such that the percentage combination increases beyond this steady value (Curve 3, Graph 2, over 900° C). The reverse—i.e. change to a more active form when the temperature of reaction is the appropriate stability zone—does not appear to take place at a measureable rate. This is illustrated in Graph 5, where the apparent rise in the curves is due to alteration in the scale.

This effect on the form of the reaction curves is not confined to the region of 700°C, as can be seen in Graph 7. Here the reaction was carried out at 1050°C and the form of curve previously mentioned is found in the case of the '850' ferric oxide. According to Hedvall[18] chemical combination is accelerated when either of the reactants undergoes an allotropic modification when heated to the temperature of reaction. Although some change in the ferric oxide certainly does take place, the negligible difference in the rates of reaction at 650°C (Graph 6) and 850°C (Graph 4), of the '650' oxide tends to eliminate this possibility.

Published data unanimously support a diminution in the chemical activity of ferric oxide if heated to over 'about 700°C'. There is nothing more specific on the actual temperature or on the cause of the change. For example, gravimetric analysis instructions stress the difficulty of redissolving a ferric hydroxide precipitate if it is ignited at temperatures over this value. Ralston[19], in a comprehensive account of the properties of ferric oxide, does not mention any change in the physical properties or structure at this temperature, but does note the loss of ferromagnetism and absorption of heat on heating at 678°C. The density, crystal structure and lattice parameter have no sharp change of value when passing through this temperature.

With this approximate figure in mind the raw materials for Graphs 1–7 were prepared at 650°C, equivalent to about the lowest possible temperature for roasting to form sulphate, and 850°C, the usual temperature for dead roasting to oxide in a reverberatory furnace. The difference in rates of reaction between these two varieties of ferric oxide indicated the probability of a drastic change occurring between these temperatures, and the experiments resulting in Graph 8 were intended to examine this, as well as the possibility of other changes, over a wide range. By preparing ferric oxide at temperatures from 600° to 1300°C at 50° intervals, and reacting with zinc oxide at a fixed temperature—850°C—two points were found, above which the reactivity fell more or less sharply. These were at about 700°C and about 1200°C. The latter temperature is that above which ferric oxide decomposes rapidly to magnetite, which is itself a spinel and inert toward zinc oxide. Analysis of the 'ferric oxide' prepared at over 1200°C confirmed the presence of an increasing proportion of magnetite with increasing temperature of preparation. The position of the lower inflexion point was rather indefinite, although roughly in the position predicted.

In order to eliminate the effect which the difference between the temperature of preparation and the temperature of reaction may have had on the position of the inflexions, the experiments resulting in Graph 9 were carried out. The temperatures of preparation of the ferric oxide and of reaction were the same for individual tests. This was also of interest, in that it might be described as a close approximation to roasting practice, where the oxides produced from the sulphides, at an approximately constant temperature, are maintained in intimate contact for a further period before serious cooling takes place. The difference in reactivity and temperature of transformation are both brought out very clearly, subsequent work at 660° and 690°C having confirmed the position of the change point. The dip in the curves at 850°–950°C is considered to be due to sintering of ferric oxide before combination and the resultant slow reaction between coarser particles. From Graph 9 it is clear that the change in the reactivity of ferric oxide takes place at 678°C and is related to the other phenomena occurring at this temperature. Nothing can be added to the knowledge existing on the relationship between them from the results of this work.

In Section (3) Graph 12 is self explanatory, in that the residue analysis was constant above a content of 33·76 per cent zinc oxide in the original mixture. Since this is the zinc oxide content of the metaferrite, it may be accepted that this is the initial compound formed between 850°C and 1400°C, from mixtures containing any proportions of the two oxides.

Graph 13 is noted as the record of experiments conducted for 60 minutes at the specified temperatures, but the form of the curves was essentially unchanged when heating periods of up to six hours were used. At first sight it appears that more complex compounds of the two oxides are formed, probably by reaction between the metaferrite and excess zinc oxide.

If the ranges of composition over which apparent complex formation occurs are compared with the compositions given in Table II, it will be seen that constancy of residue analysis similar to that in Graph 12 is not attained in the range including the supposed compound. For example $2ZnO.Fe_2O_3$ contains 50·47 per cent ZnO, but the approximate horizontal in the residue analysis curve appropriate to this value exists only between 70 per cent and 90 per cent ZnO in the original mixture. The possibility of incomplete combination is eliminated by the persistence of this situation, in an essentially unchanged form, after the longest period of heat treatment. The same observations can be made for the $8ZnO.3Fe_2O_3$.

It is reasonable to expect that if a compound is formed with more than 33·76 per cent ZnO, the residue analysis curve should rise steadily from the origin to the new value, and remain at this level to the minimum iron oxide content used in the original mixture.

It is appreciated that this state of affairs will not of necessity prevail in the final equilibrium condition, but the time required to reach equilibrium, except with special preparations, will be so long as to obviate the necessity for consideration. Microscopic examination of samples analysed in the course of preparation of Graph 13 indicate that the metaferrite is capable of dissolving ferric oxide above 1100°C and zinc oxide above 1300°C. The maximum zinc oxide content of the ferrite phase at 1400°C is about 38 per cent and from this figure to about 90 per cent zinc oxide the structure of a heat-treated mixture is that of a finely-disseminated agglomerate of metaferrite saturated with zinc oxide and a solid solution of metaferrite in zinc oxide. The latter contains about 90 per cent zinc oxide and is so intimately mixed with the insoluble constituent as to make complete extraction by a solvent highly improbable.

That this is able to take place in the absence of ferrite is shown by the fact that the residue analysis falls practically to the metaferrite value when the original zinc oxide content exceeds 90 per cent. Persistence of envelopment, due to heterogeneity, probably accounts for failure to fall completely to this value. This latter diminution in the combined zinc oxide runs contrary to the expected stability of a high zinc oxide compound, when heated in contact with great excess of this material. Figs. 15 and 16 show the structure of heavily-etched pure metaferrite and of the complex structure referred to. From this evidence it is considered that the metaferrite is the only compound formed in the system and that the others are due either to solid solution of ferric oxide or to failure to extract the free zinc oxide by solution.

The effect of the zinc oxide-ferric oxide ratio on the rate of combination is very difficult to determine. Only in the case of the high-temperature ferric oxide is there some indication of the rate falling with the value of this ratio. Graph 14 is of interest in that the effect of zinc oxide in retaining the iron in the ferric state results in the introduction of a greater amount of oxygen into the distillation retort charge than would be the case if the uncombined ferric oxide had been converted to ferrous oxide by the heat treatment.

Acknowledgements

The author's thanks are due to the Imperial Smelting Corporation for a grant towards the cost of apparatus and to the Research Department of the same company for assistance in the supply of zinc oxide, as well as valuable aid in carrying out a large number of the analyses.

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Recovery of Sulphur from Smelter Gases by the Orkla Process at Rio Tinto

H. R. POTTS, MEMBER, and E. G. LAWFORD, A.R.S.M., MEMBER

Further contributed remarks on paper published in Bulletin 509, *April* 1949.

THE TECHNICAL STAFF OF ORKLA METAL-AKTIESELSKAP: As a contribution to the paper we give below as a comparison a short summary of the Orkla process as it works to-day at Thamshävn. The calculation shown in Table XII is based on the technical report for March, 1949.

TABLE XII

CHARGED TO THE FURNACES AND PRODUCTS
(in metric tons)

	Tons	*Cu* %	*Fe* %	*S* %	SiO_2 %	*Zn* %
Pyrites	29,980	1·866	38·149	41·018	12·95	1·922
Quartz	4,826				95·00	
Limestone (95% $CaCO_3$, 16% H_2O)	1,837					
Coke (11% ash)	4,144					
Poor matte	4,591	11·355	54·00	24·00		
Products:						
Rich matte	1,384	37·44		24·24	0·04	
Slag from matte furnace	4,426	0·50		2·13	29·55	
Slag from pyrite furnace	21,555	0·27		2·67	34·33	
Dust from gas mixer	105			21·06		
Dust from Cottrells	152			15·22		

EXIT GASES FROM THE FURNACES

Gases	*Per cent by Vol.*	*g. per cu. m.*	*g. per cu. m.*	
			C	*S*
CO_2	13·2		71·0	
CO	0·2		1·1	
O_2	0·5			
SO_2				24·92
H_2S				11·62
CS_2		28·8	7·1	21·70
COS		22·3	4·0	18·83

As it appears from the feed balance the re-smelting of the poor matte is included there in order to make the following calculation easier. The poor matte is re-smelted in its own furnace, but the amount of fluxes and coke used is included in the feed balance and the gas from the re-smelting furnaces is mixed with gases from the pyrite furnaces.

In the following calculation the distillation of the loose atom sulphur is also calculated as follows :

$$7\,FeS_2 + \text{heat} = Fe_7S_8 + 3\,S_2 \qquad (1)$$

$$2\,Fe_7S_8 + 23\,O_2 = 14\,FeO + 16\,SO_2 \qquad (2)$$

Some authors, however, calculate with :

$$2\,FeS_2 + \text{heat} = 2\,FeS + S_2$$

At Orkla we also calculate with the following reactions :

$$SO_2 + C = CO_2 + \tfrac{1}{2}\,S_2 \qquad (3)$$

$$CO_2 + C = 2\,CO \qquad (4)$$

$$SO_2 + 2\,CO = \tfrac{1}{2}\,S_2 + 2\,CO_2 \qquad (5)$$

TABLE XIII

DETERMINATION OF THE VOLUME OF GAS PER TON OF PYRITES SMELTED BY MEANS OF THE CARBON BALANCE

	kg. of Carbon per ton pyrites
Carbon charged to the furnaces as coke	100·9
„ „ in limestone	7·0
Total carbon charged to furnaces	107·9
	g. Carbon per cu. m.
Carbon in Exit Gases	
CO_2, 13·2 per cent by volume	71·0
CO, 0·2 „ „ „	1·1
COS	7·1
CS_2	4·0
Total	83·2

The volume of gas per ton pyrites is :

$$\frac{107{\cdot}9 \times 1{,}000}{83{\cdot}2} = 1{,}300 \text{ cu. m.}$$

TABLE XIV

Sulphur Engaged As :

SO_2	= 24·92 × 1,300 = 32·40	kg. per ton pyrites.
COS	= 18·83 × 1,300 = 24·48	„ „ „ „
CS_2	= 21·70 × 1,300 = 28·21	„ „ „ „
H_2S	= 11·62 × 1,300 = 15·11	„ „ „ „
Total	100·20	

Sulphur Balances

		kg. per ton pyrites
Charged sulphur		410
Sulphur combined with Cu, Zn, etc.		15
Sulphur in FeS_2		395
Volatile sulphur (42 per cent in FeS_2)		166
Sulphur in Fe_7S_8		229
Sulphur combined with Cu, Zn, etc.		15
Fixed sulphur entering smelting zone		244
Sulphur in matte	$\frac{1,384 \times 24{\cdot}24}{29,980 \times 1,000} = 11{\cdot}2$	
Sulphur in slag	$\frac{669{\cdot}800}{29,980} = 22{\cdot}3$	
Sulphur in dust	$\frac{62,100}{29,980} = 2{\cdot}1$	35·5
Sulphur oxidized to SO_2		208·5
Sulphur as SO_2 not reduced in the gases from the furnaces:	$\frac{24{\cdot}92 \times 1,300}{1,000} =$	32·4
Sulphur produced by reduction of SO_2		176·1

Table XV

Carbon Entering into the Reduction Reaction

	kg. per ton pyrites
Total carbon in coke charged to the furnaces	100·9
Carbon combined as COS, CS_2, and CO in gases from the furnaces, about	15·9
Carbon for reduction of SO_2	85·0
If 10 per cent of the carbon reaching the focus is burnt to CO_2, not reduced to CO, and passes into gases as CO_2—that is	10·0
Carbon to reduction of SO_2 is	75·0

Table XVI

Calculation of the Proportion of the Total Sulphur Reduced from SO_2 by CO and by Solid C

x = weight of carbon reducing SO_2 by equation (3).
y = weight of carbon burnt to CO_2 at the focus, but subsequently reduced to CO, which then reacts with SO_2 by equation (5):

$$x + y = 75$$

$$\frac{32x}{12} + \frac{32\,(y - 0{\cdot}1\,y)}{24} = 176{\cdot}1$$

Solving this equation gives:

$$x = 58{\cdot}7 \text{ kg./ton pyrites.}$$

$$y = 16{\cdot}3 \quad ,, \quad ,,$$

Thus sulphur reduced from SO_2 by carbon as CO = 21·1 kg./ton pyrites ... 12 per cent
and reduced from SO_2 by solid C = 155·0 kg./ton pyrites ... 88 ,,

It is evident from the calculations set out in Tables XII to XVI that we have a better reduction of SO_2 per unit of carbon than is the case at Rio Tinto. This depends to a certain degree upon the fact that we have a higher ratio of concentration (mineral to matte) in our furnaces.

Cleaning Dust from the Gases

The heaviest particles of dust are settled out in flues and in the gas mixer (recovery 4–5 kg./ton pyrites). Then the gases pass through Cottrells (8 Cottrells for 4 furnaces) which clean the gases of the finest particles of dust. We have a very good efficiency, as the sulphur condensed only contains 0·02 per cent ash (recovery of dust is 5–6 kg./ton pyrites).

Reaction of Catalysing

After the Cottrells the gases enter into a catalyser chamber at a temperature of 450°C. and pass through a catalysing mass.

The analysis of the gas before the catalyser is :

Sulphur as :		*g./cu. m.*	
SO_2	=	24·92	
H_2S	=	11·62	
COS	=	18·83	
CS_2	=	21·70	
		——	77·07
CO_2	=	13·2 per cent volume.	
CO	=	0·2 ,, ,,	
O_2	=	0·3 ,, ,,	

The analysis of the gas after the catalyser is :

Sulphur as :		*g./cu. m.*	
SO_2	=	21·25	
H_2S	=	12·59	
COS	=	8·13	
CS_2	=	10·47	
		——	52·44
CO_2	=	13·70 per cent volume.	
CO	=	0·10 ,, ,,	
O_2	=	0·20 ,, ,,	

Recovery = 77·07 — 52·44 = 24·63 g. S/cu. m.

$$\text{Per ton pyrites} = \frac{24{\cdot}63 \times 1{,}300}{1{,}000} = 32{\cdot}02 \text{ or } 7{\cdot}8 \text{ per cent of total S.}$$

The gas is then cooled down in a condenser to about 140°C. The analysis of the condensed S is :

Ash	=	0·02 per cent.
As	=	0·14 ,,
Se	=	0·014 ,,

The gas is reheated in an oil-fired heat exchanger to 250°C. and passed through the second catalyser. The analysis of the gas after the second catalyser is :

Sulphur as:		*g./cu. m.*	
SO_2	=	16·67	
H_2S	=	7·35	
COS	=	2·77	
CS_2	=	8·95	
			35·74
CO_2	=	15·03 per cent volume.	
CO	=	0·00 „ „	
O_2	=	0·00 „ „	

Recovery = 52·44 − 35·74 = 16·7 g./cu. m.

Per ton pyrites $= \frac{16{\cdot}7 \times 1{,}300}{1{,}000} = 21{\cdot}71$ g./ton pyrites, or 5·3 per cent of total S.

The gas is then cooled down to 130°C., and the sulphur condensed and collected in gas coolers and scrubbing towers. The analysis of the sulphur is: Ash, 0·005 per cent; As, 0·009 per cent; Se, 0·002 per cent.

The tail gases from the scrubbers are cooled down in cooling towers to 30°C. by water, passed through absorption towers filled with limestone, and sprayed with seawater from the top. All SO_2 is here absorbed before the gases are passed out to the free.

It will be seen that the analyses of the different gases will not correspond to the reaction going on by catalysis, but this is because gas-samples are not taken at the same time, and the gases can differ from time to time. The recovery of sulphur after these analyses should be:

328 kg./ton pyrites

$$= \frac{328 \times 100}{410} = 80 \text{ per cent.}$$

However, we had a recovery of 341 kg./ton pyrites

$$= \frac{341 \times 100}{410} = 83{\cdot}1 \text{ per cent.}$$

It is therefore obvious that we have had better reaction than the analysis shows.

It is evident that it is impossible to recover more sulphur by catalysis of the exit gases from the furnaces before the sulphur containing almost all the arsenic has been condensed and collected. Afterwards, however, the gases can be re-heated and sulphur recovered by catalysis.

Another thing is worthy of attention. The low-grade matte should be concentrated by re-smelting in a closed-top furnace, the gas cleaned from dust by Cottrells, and this gas mixed with the gas after the condenser, and the gas-mixture passed through a catalyser. The gas from this furnace would be almost free from arsenic and contain a surplus of SO_2, which would react with other sulphur-compounds in the catalyser. The other furnaces could then be charged with more coke to diminish the content of SO_2 in the gas from them.

The fuel consumption for re-heating the gases from the first condenser will also be less as the gas from the re-smelting furnace has a temperature of about 300°C.

Experiments on the Removal of Selenium and Tellurium from Blister and Fire-refined Copper

W. A. BAKER, B.SC., F.I.M., and A. P. C. HALLOWES, B.SC., A.I.M.

Contributed remarks on paper published in Bulletin 513, *August* 1949.

MR. H. R. POTTS*: The paper written by Messrs. Baker and Hallowes contains some exceedingly interesting information and should be of great practical value to metallurgists at smelters which produce blister copper which contains Se and Te, as showing what can be done to eliminate these very objectionable elements.

Selective converting, which is usually carried out with a view to concentrating gold and silver values, is possible because of the fact that gold has a much greater affinity for copper than for 'white metal', while silver has a slightly greater affinity; as a result of this it is possible to concentrate more than 90 per cent of the gold and more than 50 per cent of the silver into less than 50 per cent of the blister copper produced from a given charge of matte. A similar action, although working in the opposite direction, evidently makes the process a practical one for concentrating Se and Te.

In Table I (first section) the weight of white metal produced from each 350-gram charge of matte is not given, but by working with the theoretical quantity of white metal which could be produced and applying to it the analysis given for the Se and Te content of the white metal it seems that in the case (taken at random) where 49 per cent of the copper in the matte had been brought down as blister, which contained 2 per cent of the original total Se and 9 per cent of the original total Te present in the matte, that 80 per cent of the original Se and 63·7 per cent of the original Te had passed into the remaining white metal.

These figures, if correct, give a total of 82 per cent Se and 72·7 per cent Te accounted for; the balance in each case must presumably have been slagged and volatilized.

Even if only approximately correct these figures do not agree with those given by the authors on page 2 under the heading 'Results', which show a maximum of 6 per cent Se and 20 per cent Te eliminated; it may be possible for the authors to clear up this point, because the writer may have misunderstood their data.

The problem of separating the white metal from the blister copper is a very real one in works practice; the writer has had

*Late Smelter Superintendent, The Rio Tinto Co., Ltd.

considerable experience with selective converting, using 12-ft. Great Falls upright converters, and it was found that, in order to get good and consistent results, it was essential to have a large quantity of white metal, as pure as possible, before beginning the selective blow; a very suitable quantity was two ladles, or about 15 tons. When the selective blow was finished, about three-quarters of a ladle of white metal, containing about 6 tons, was poured off, while a similar quantity of blister remained in the converter; the impoverished white metal could then be blown to blister separately, or fresh matte could be added to it.

In this way there was no necessity to tap the copper direct from the base of the converter; this method has been used in the past in more than one smelter.

The late Henry F. Collins used it at the Cueva de la Mora smelter in the south of Spain, two papers of his on this subject having been published in the *Transactions* of the Institution.*

*Note on the concentration of gold in bottoms in the converter. **24**, 1914–15, 489–94.

Note on the purity of selected copper made in converters. **26**, 1916–17, 256–8.

Notes on Mining Education and Postgraduate Training

J. A. S. RITSON, O.B.E., D.S.O., M.C., MEMBER

Further contributed remarks on paper published in Bulletin 511, *June* 1949

MR. REGINALD S. H. RICHARDS: As one who has been right through the mill, and held every position possible in a mine from mucker, driller, shiftboss, foreman, to superintendent and general manager, I have found very interesting the paper by Professor Ritson and the discussion which has followed.

Personally, I should never put a young graduate on to a two-year course of practical work with the status of a learner (although I did one year and nine months myself)—three months is quite sufficient. As a mine superintendent I have trained quite a number of young graduates, the majority of whom had received no previous practical training whatsoever. My method was to give them a fortnight or so on drifters, a week or so on jackhammers and stopers, a few days with drill-fitters and pipe-fitters, a week or two charging and blasting, a couple of weeks as assistant to a chargeman, and then time studies on mucking and tramming, followed by time studies of drilling, followed, perhaps, by timbering or time studies of haulage, hoisting, etc., reports being called for from each operation. At the end of, say, three months the graduate would be placed in charge of a development end where details and time studies of all operations were called for, the graduate being encouraged to make suggestions and criticism, often being allowed to try out his suggestions. In nine cases out of ten the graduate became quite an enthusiast in a very short time.

Before the graduate became tired of the single-end job he would be put in charge of two, three or more development jobs, including rising or sinking, whenever possible. During this period of training the graduate would have direct access to the mine superintendent for reporting and discussion, thus a certain number of working faces would receive special attention and the graduate already had an interesting job. After two or three months of this class of work the graduate would be put on to efficiency studies, such as explosive consumption per ton of mineral broken, per metre of advance, etc., metres drilled per machine shift, or tons per metre drilled, etc. Numerous possibilities will suggest themselves to the mine manager. Well before the end of one year the graduate was more than earning his salary, whilst he had the satisfaction that he was doing a job of work, steadily gaining experience and status, at the same time learning to handle men by close contact with them.

Occasionally I would send them through the mine to study the ventilation or consumption of timber, always choosing jobs for the graduate which would give me some valuable information, and for which I could not spare any of the normal mine staff. Whenever the graduate showed enthusiasm and promise I always increased his salary *before it was due by contract*, this in my opinion being a very important item in the training of good assistants.

MR. RALPH SYMONS: It was not my intention to contribute to this discussion but the drastic changes suggested by some members incline me to the opinion that, before any such changes are made, as many ideas as possible should be gathered from practising engineers.

The prospect of lengthening the University course in mining to six years is rather a frightening one and not likely to commend itself to parents who, in these difficult times, are seeking a suitable professional career for their sons in return for a reasonable expenditure in educating them for it. The probable result of introducing such a change would be a contraction in the numbers of graduates entering the industry.

Nor do I think a comparison with professions such as law and medicine a fair one. In those professions, suitable qualifications are a *sine qua non* whereas in mining, the qualified man has to compete with unqualified people who are able to rise to the top positions in the industry. Furthermore, mining is a hazardous occupation and therefore the University course should not be too long.

In certain continental countries, the title of 'Engineer' has the same status as 'Doctor' and is used in a similar way. It is granted only to those who, having passed a University course of four years, spend at least two further years practising and then submit a thesis on some mining project. But, in return for this extra time, the senior positions in the mining industry are only available to qualified engineers. If such a change were contemplated in British practice, I believe the effects would be entirely beneficial, but I would not welcome any lengthening of University courses otherwise.

I agree that the essential aim of mining schools must be to turn out their graduates with a sound theoretical training and to this end, unlike some contributors, I believe that the earlier the 'sheet anchors' of mathematics, chemistry, and physics are taught, the better. But there is no reason why the onus of providing the practical experience should fall wholly on the industry. In my day at the Royal School of Mines any man who had conscientiously carried out the course was a first-class surveyor, a fair assayer and had a good working knowledge of ore-dressing. In these branches, my experience has been that the R.S.M. man need fear no competition from other mining graduates. The reason for this is not far to seek—there was a long practical course in surveying in a mine owned by the School and the assay and ore-dressing laboratories were probably the finest in the world. On the other hand, the R.S.M. man did sometimes find himself at a disadvantage, particularly in comparison with Canadian and American graduates, in the all-important matter of practical underground mining.

To overcome this admitted defect, I would advocate that the Royal School of Mines supplement its course with a practical mining course in its own mine to be held during the vacations. The course should be supervised by a teacher who has had long experience as a mine captain, and practical and *viva voce* examinations should complete each section. The students should be arranged in gangs and each student should have his turn as leader in charge; in this way all would learn to give and receive instructions. Any student who failed the practical course should be obliged to repeat it. Throughout the course, safety and mining regulations should be stressed. Summer vacation courses at other mines should be abolished; I agree with Mr. J. A'C. Bergne—they are a joke.

I believe if some such scheme were carried out, a graduate leaving the Royal School of Mines would enter his first job with a good deal more confidence than he does now. Moreover, practical underground courses under the eyes of the College staff would quickly reveal the 'misfit', who could then be advised to transfer to some allied but more theoretical course.

If any changes are needed in the subjects taught at the Royal School of Mines, I would suggest courses in mining law (safety regulations) and the riduments of building and planning of surface layouts, including houses, compounds, change houses, sanitation, etc. Included in the survey course should be a *viva voce* examination in plan reading—it needs considerable practice to be able to grasp quickly a plan of a strange mine which was drawn by another person.

To counteract the tendency to overspecialize by students taking technical courses, I would suggest that they be encouraged to devote some time to activities such as sports, non-technical discussion groups and debating societies, etc., and that achievement in these directions should receive merit towards the student's degree. Non-participation in outside activities should bar progress in the course. Mining students should be made to realize that a manager's responsibilities necessitate an interest and a competency in matters divorced from mining. I cannot see that any useful purpose would be secured by continuing the study of classics at the University; I feel that if the student comes to the University with a sound general education he will continue 'liberal' studies when he has more time at his disposal, i.e., after graduation, and what is more important, he will then do so of his own free will. Nevertheless, I agree that some training in clear and concise expression both in speech and writing should form part of the University course.

The newly-fledged graduate on entering his first job will, in most cases, be confronted with the task of settling down in a strange country and absorbing the language and customs thereof. Moreover, he will be debarred by law from taking any direct responsibility until he has spent some considerable period acquiring underground experience and then passing examinations for the necessary certificates of competency. It is during this period, I submit, that the true function of the industry lies in preparing young graduates for their jobs. I agree with Mr. Thomas Pryor that this preparation should not be in the form of a standard postgraduate course but more of the type advocated by Messrs. S. E. Taylor and G. Keith Allen—viz., that the graduate should be given official status straight away as a ' technical assistant ' (the term ' learner ' is unnecessarily humiliating and should be dropped) and that the object of the training should be to turn him into an efficient shift boss in the shortest possible time. After some time spent on various jobs under the wing of experienced miners, he should become an assistant shift boss gradually acquiring responsibility until he has reached the stage where he can be appointed shift boss with full responsibility for a section of the mine ; the length of time this would take would depend largely on the capacity of the individual concerned, but if the R.S.M. course were revised as suggested the graduate should be able to reach the shift-boss stage within six to twelve months of starting, according to his progress.

I do not myself believe that it is necessary or desirable that a graduate should spend time in the ancillary departments of the mine such as survey office, assay office, study department, etc., at this early stage. With his training he will not be able to avoid learning much about the activities of these departments while he is in the mine. Later, after he has reached the shift-boss stage, he should be allowed and encouraged to pay visits to these departments and should particularly be given every facility to become thoroughly acquainted with the underground plans. I regard it as most important that the mining graduate should be identified with the underground department from the outset of his career. In travelling from one department to another on a fixed timetable, the young engineer is nobody's ' baby ' and tends to take only a peripatetic interest. Right through the training period regular reports should reach the manager about graduates' work and each graduate should have an interview alone with the manager once a month.

. 516 NOVEMBER, 1949

BULLETIN OF THE ISTITUTION OF MINING AND METALLURGY

Principal Contents:

LD CONCENTRATION AT THE AMALGAMATED BANKET AREAS REDUCTION PLANT

By G. CHAD NORRIS, *Member*

SOME NOTES ON A MECHANICAL CONCENTRATOR

By T. HADEN, B.E.M., *Associate Member*

Published monthly by The Institution of Mining and Metallurgy
Salisbury House, Finsbury Circus, London, E.C. 2

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BULLETIN OF THE INSTITUTION OF MINING AND METALLURGY

NO. 516–NOVEMBER 1949

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Telephone: Monarch 2096–7. Telegrams: Minanmet, Phone, London

CONTENTS

NOTICE OF GENERAL MEETING

The Second Ordinary General Meeting of the Fifty-Ninth Session of the Institution of Mining and Metallurgy will be held, by kind permission, in the Apartments of the Geological Society of London, Burlington House, Piccadilly, London, W.1, on THURSDAY, 17TH NOVEMBER, 1949, at 5 p.m.

The following papers will be submitted for discussion: *Factors affecting the rate of formation of zinc ferrite from zinc oxide and ferric oxide*, by Mr. D. W. Hopkins, M.SC., A.I.M., of the University College of Swansea (published in the October *Bulletin*); *Gold concentration at the Amalgamated Banket Areas reduction plant*, by Mr. G. Chad Norris, *Member*, of the West African Gold Corporation, Ltd.; and *Some notes on a mechanical concentrator*, by Mr. T. Haden, B.E.M., *Associate Member*.

Light refreshments will be provided at 4.30 p.m. for members and visitors attending the Meeting.

The Council invite written contributions to the discussion of papers from members who may be unable to be present at the Meetings of the Institution. The Council reserve the right to edit and condense such contributions.

The Institution as a body is not responsible for the statements made or opinions expressed in any of its publications

INSTITUTION NOTES

FILM SHOWING—1 DECEMBER

The Zinc Corporation, Ltd., have kindly lent the Institution their film of the activities of the Corporation and of New Broken Hill Consolidated, Ltd., at Broken Hill, and it will be shown on Thursday, 1st December, 1949, at the Mining Theatre of the Royal School of Mines, Prince Consort Road, South Kensington, S.W.7. The showing will begin at 5 p.m. and will last about an hour. Members and their friends are most cordially invited. Admission will be free, without ticket.

NEWS AND VIEWS

The Council of the Institution wish to begin a new section in the *Bulletin* which they hope will be used by members of all classes for contributing news, ideas and observations on their work and the industry. The Institution's publications do not at present provide space for brief notes and correspondence, and it is proposed to meet this need by introducing the new feature.

The Council trust that the new section will be welcomed by members and that there will be a ready response to this invitation for contributions. All communications for publication should be sent to the Secretary. It will be understood that the responsibility of deciding to publish communications and to edit them must remain with the Council.

LIBRARY SERVICE—TRANSLATIONS

The Council of the Institution wish to form a panel of translators from among members of the Institution for the translation of technical articles required by members, the cost of which would be borne by the members for whom the translations were made. Members willing to translate articles are invited to notify the Secretary of the languages and subjects they would undertake, and terms of remuneration.

It is hoped that in this way a translation service may be established.

THE COMMONWEALTH FUND FELLOWSHIPS

The Commonwealth Fund of New York have decided to offer in 1950 a number of Fellowships tenable for one year in the U.S.A. There is no fixed stipend, but the emolument attaching to each Fellowship provides for a minimum of $3,350 for twelve months, plus the cost of travel to and from the United States. It is calculated to cover the full expenses of residence, study and travel, and the Directors of the Fund are prepared to pay an additional stipend to married men or women.

Four categories of Fellowships are offered: 20 Ordinary Fellowships, 5 Home Civil Service Fellowships, 5 Dominion Civil Service Fellowships, and 2 Colonial Civil Service Fellowships. Candidates should be between 23 and 35 years of age. Ordinary Fellowships are offered to candidates domiciled in the United Kingdom who are graduates of British universities or graduates of recognized universities in other parts of the Empire. Home Civil Service Fellowships are open to candidates holding permanent appointments in the higher ranks of the Civil Service in Great Britain. The Dominion and Colonial Civil Service Fellowships are offered to candidates employed overseas by Dominion or Colonial Governments.

Full particulars and forms of application (returnable by 15th December, 1949) may be obtained from the Secretary to the Committee of Award, 35, Portman Square, London, W.1.

REMAINING MEETINGS IN THE SESSION 1949–1950

The dates of the General Meetings, to be held on the third Thursday in each month, during the remainder of the Session are as follows:

15th December, 1949
19th January, 1950
16th February, 1950
16th March, 1950
20th April, 1950
18th May, 1950
15th June, 1950

PAPERS FOR DISCUSSION AT SUBSEQUENT GENERAL MEETINGS

The following programme of papers has been arranged for the December and January General Meetings:

ɪʀ : *Management in industry,*
Hill, B.Sc. (Min.), B.A.,
residential Address to the
Metallurgical and Mining
South Africa, and repub-
ıe July, 1949, issue of the

: *An outline of underground*
ıt Mufulira Copper Mines,
ſ. P. Norrie and W. T.

GENERAL MEETING

ȶ Ordinary General Meeting
ion was held on Thursday,
ɔer, at Burlington House,
papers were submitted and
n the absence of the authors,
eadle of the Dust and Venti-
ɹrtment, Transvaal Chamber
ntroduced the first paper—
the wet kata thermometer on
rsrand, by Mr. P. H. Kitto—
W. Smith, *Member,* intro-
paper entitled *Investigations*
luction of electrolytic cobalt
pper-cobalt flotation concen-
ɪssrs. H. L. Talbot and H. N.
report of the discussion will
ɪd in the December issue of
·.

ʃ FROM ABROAD

ncil are always anxious to
ɔers who come to England
g absence abroad, and ask
bers to make themselves
he Secretary when attending
ɪetings of the Institution at
House.

ΓES FOR ADMISSION

welcome communications to assist
ling whether the qualifications of
admission into the Institution fulfil
ıts of the Bye-laws. The application
dates (other than those for Student-
ɔen for inspection at the office of the
a period of at least two months from
ɪ *Bulletin* in which their applications

lowing have applied for
ıce 13th October, 1949 :

ɪERSHIP

Airth (*Blyvooruitsig, Trans-*

Vatson Connor (*Banstead,*

Vivian Cunliffe (*Bushtick,*
Rhodesia).
ɹglas Dixon (*Marikuppam,*

Redvers Buller Greaves (*Bindura, Southern Rhodesia*).
Bertram Alan Miller (*London*).
James Malcolm Newman (*D'Aguilar, Queensland, Australia*).
Cyril John Douglas Veal (*Johannesburg, Transvaal*).

To Associate Membership

Philip Charles Metcalfe Bathurst (*Germiston, Transvaal*).
Dudley John Batzer (*Kingsbridge, Devonshire*).
Jack Menneer (*Bogota, Colombia*).
Dean Arthur Oliver Morgan (*Bulawayo, Southern Rhodesia*).
David John Penney (*Yauricocha, Peru*).

The following have applied for admission since 13th October, 1949 :

To Membership

Oliver Bosshardt Bennett (*Nkana, Northern Rhodesia*).
Eric Frank Marland (*Springs, Transvaal*).

To Associate Membership

William George Collins (*Carnwath, Lanarkshire*).
Bruce Crawford (*London*).
Norman Evelyn Dougans (*Johannesburg, Transvaal*).
Geoffrey Luther Evans (*London*).
Henry Alfred Page Hetherington (*Staines, Middlesex*).
Alexander George Scoular (*London*).
Dayatissa Seneviratne (*Ruanwella, Ceylon*).
Donald Lindsay Turnbull (*Mufulira, Northern Rhodesia*).
Edwin Charles Young (*Mbarara, Uganda*).

To Affiliateship

Robert Hermann Alexander Neuschild (*Enfield, Middlesex*).

To Studentship

John Anthony Eaton Allum (*London*).
Kenneth Ernest Sylvester Applin (*Camborne, Cornwall*).
John Lindon Ashford (*Chandlers Ford, Hampshire*).
Anthony René Barringer (*London*).
John Hunter Bennie (*Camborne, Cornwall*).
James Gerald Brading (*Purley, Surrey*).
Brian Buckley (*Doncaster, Yorkshire*).
Michael Henry Cleary (*Camborne, Cornwall*).
Harry Kemp Cole (*Bedford, Bedfordshire*).

John Hamilton Crawford (*Kalgoorlie, Western Australia*).
Michael Dore Cruikshanks (*Herne Bay, Kent*).
Keith Francis Gosling (*Mt. Hawke, Cornwall*).
Thomas Albert Winfield Haddon (*Camborne, Cornwall*).
Frederick John Trevor Hancock (*Kidderminster, Worcestershire*).
Robert William Alan Hansel (*St. Ives, Cornwall*).
Peter Henry George Hayward (*London*).
Thomas Andrew Henderson (*Hexham, Northumberland*).
John Michael Wren Humphreys (*London*).
Leong Sing Lim (*Camborne, Cornwall*).
Tony Bernard Lock (*Kingston-on-Thames, Surrey*).
John Carlo Loretto (*Montreal, Canada*).
Keith Edward Mantell (*Hayle, Cornwall*).
Thomas Oswald Martyn (*Wadebridge, Cornwall*).
John Walton Martyr (*Camborne, Cornwall*).
William Garth Barrington Phillips (*Framlingham, Suffolk*).
James Potter (*St. Ives, Cornwall*).
Frederick Raynes (*Camborne, Cornwall*).
Peter William Edward Richardson (*Camborne, Cornwall*).
Mohan Singh (*Camborne, Cornwall*).
Michael Wyndham Stephenson (*Camborne, Cornwall*).
Arthur Raymond Tron (*Gateshead, Co. Durham*).
Peter Lloyd Walker (*London*).

TRANSFERS AND ELECTIONS

The following were transferred (subject to confirmation in accordance with the conditions of the Bye-Laws) on 13th October, 1949:

To Membership

Henry Thomas James Edward Barker (*Selukwe, Southern Rhodesia*).
James Herbert Bennetts (*Camborne, Cornwall*).
William David Evans (*Nottingham*).
Henry Nisbet Lightbody (*Que Que, Southern Rhodesia*).
Ben Lightfoot (*Maidenhead, Berkshire*).
Frank Pinney Longmire (*Wareham, Dorset*).
Richard Bradford McConnell (*Kaduna Junction, Northern Nigeria*).
Charles Harold William Martyn (*London*).
William Edward Sinclair (*Koegas, C.P., South Africa*).
Frederick Charles Willoughby (*Filabusi, Southern Rhodesia*).

To Associate Membership

Anthony Vernon Bradshaw (*Alemtejo, Portugal*).
Denis Bridger (*Kitwe, Northern Rhodesia*).
Ronald Frederick Jarvis (*Marikuppam, South India*).
Fadil Kheiry Kabbani (*Jeddah, Saudi Arabia*).
John Eddy Kernick (*Champion Reefs, South India*).
Wilfred Henry John Luck (*Que Que, Southern Rhodesia*).
David Ronald Mitchell (*Crowborough, Sussex*).
Dennis Frederick Reeves (*Bukuru, Northern Nigeria*).
James Ryan (*Luanshya, Northern Rhodesia*).
Lindsay Lee Shearer (*Mufulira, Northern Rhodesia*).

The following were elected (subject to confirmation in accordance with the conditions of the Bye-Laws) on 13th October, 1949:

To Membership

Robert Lepsoe (*Trondheim, Norway*).
Donald McDonald (*Beckenham, Kent*).

To Associate Membership

George Milton Barnett (*Nicosia, Cyprus*).
Lance Gordon Earle (*Bristol, Gloucestershire*).
Daniel Aloysius Harkin (*Dodoma, Tanganyika Territory*).
D. Thyagaraja Iyer (*Champion Reefs, South India*).
Syed Kazim (*Hyderabad, India*).
John Edward Howse Keylock (*Jos, Northern Nigeria*).
John Cecil McMullin (*Mbarara, Uganda*).
Kabool Chand Maithal (*Ruwanwella, Ceylon*).
Kenneth Charles Braman Morrison (*Tarkwa, Gold Coast*).
George Edward Cunynghame Robertson (*Kisumu, Kenya*).

Robert Whatmough (*Kakamega, Kenya*).
Alan James Wilson (*Oorgaum, South India*).

To Affiliateship

David Faunch (*Wembley Park, Middlesex*).
Arthur Griffith (*London*).
Harry Albinson Mellor (*Orpington, Kent*).

To Studentship

John Walsh Barnes (*Wellington, Shropshire*).
William Edward Cliff (*Camborne, Cornwall*).
Robert Edwin Laurence (*Bendish, Hertfordshire*).
Alan By Nichols (*Camborne, Cornwall*).
Dennis Michael O'Donahue (*Camborne, Cornwall*).
Peter John Hedley Rich (*London*).
David Cowper Tennent (*Dunedin, New Zealand*).
Brian Harry Coles Waters (*Cambridge*).

NEWS OF MEMBERS

Members, Associate Members, Affiliates, and Students are invited to supply the Secretary with personal news for publication under this heading

Mr. J. D. Aitken, *Member*, is now in England on leave and returns to India at the end of the month.

Mr. K. L. Allen, *Student*, has left England to take up the position of mining engineer to Cerro de Pasco Copper Corporation, Peru.

Mr. H. Arnall, *Member*, expects to arrive in England from Nigeria towards the end of this month.

Mr. A. M. Babington, *Student*, has returned to England on leave on the completion of his agreement with Rosterman Gold Mines, Ltd.

Mr. J. W. Barnes, *Student*, has left England to take up the appointment of geologist to the Colonial Geological Survey of Uganda.

Mr. D. J. Batzer, *Student*, has left England to join A.O. (Malaya), Ltd., Kuala Lumpur.

Mr. R. Blanchard, *Member*, has returned to Sierra Madre, California.

Mr. C. M. Carroll, *Member*, sailed last month for Peru after a visit to England and the Continent.

Mr. J. T. Chappel, *Member*, is returning to Malaya this month.

Mr. A. J. M. Clesham, *Student*, is leaving England to take up an appointment with Roan Antelope Copper Mines, Ltd., Luanshya, Northern Rhodesia.

Mr. W. H. Collins, *Associate Member*, has returned to England on three months' leave from Northern Nigeria.

Mr. P. H. Cowdery, *Student*, is now employed at Hollinger Consolidated Gold Mines, Ltd., Timmins, Ontario.

Mr. A. L. Crockford, *Student*, is now in Holland in the employ of the Anglo Saxon Petroleum Company, Ltd.

Mr. H. N. B. Dover. *Associate Member*, has been transferred from Colombia to Venezuela by the Compania de Petroleo Shell de Colombia.

Mr. J. C. Erskine, *Associate Member*, has left for Nigeria to take up his duties as Inspector of Mines at Jos.

Mr. D. F. Fairbairn, *Associate Member*, has taken up an appointment with the Colonial Development Corporation in Tanganyika.

Mr. K. R. Fleischman, *Student*, has been appointed mining engineer and metallurgist to Dickson Primer and Co., Pty., Ltd., Sydney, N.S.W.

Mr. R. P. Grosscurth, *Student*, is now at Boksburg, Transvaal, with Van Dyk Consolidated Mines, Ltd.

Mr. H. J. E. Haggard, *Student*, has left England for Consolidated African Selection Trust, Ltd., Gold Coast.

Mr. C. E. H. Hallé, *Student*, has left this country to take up a three-year contract with Cerro de Pasco Copper Corporation, Peru.

Mr. E. G. Harding, *Member*, has been promoted from assistant consulting engineer to consulting engineer of African Associated Mines, Ltd., Bulawayo, Southern Rhodesia.

Mr. W. Hoatson, *Associate Member*, is on leave in the United Kingdom and intends to return to Southern Rhodesia early next month.

Mr. J. H. Hohnen, *Associate Member*, has been appointed general manager to New Guinea Gold Fields, Ltd.

Mr. W. T. Hooper, *Associate Member*, expects to return to England on leave from the Gold Coast at the end of this month.

Mr. J. L. Jolly, *Associate Member*, has recently accepted a position as Inspector of Mines, Broken Hill, with the New South Wales Mines Department.

Dr. R. B. McConnell, *Associate Member*, is returning to England on leave from Northern Nigeria.

Mr. P. H. McDouall, *Member*, has left England on a visit to Portugal.

Mr. R. K. McLeod, *Associate Member*, has returned to the Department of Lands and Mines at Tabora, Tanganyika Territory.

Mr. R. J. Matthews, *Student*, has taken up an appointment with the Consolidated Mining and Smelting Co., Ltd. of Canada, at Trail, B.C.

Mr. H. R. Miles, *Associate Member*, has left Amalgamated Banket Areas, Ltd., and has gone to Western Australia.

Mr. H. L. Mitchell, *Student*, has returned this month to the United Kingdom.

Mr. V. A. Phillips, *Student*, has left the U.S.A. and has joined the Cavendish Laboratory, Cambridge, under Dr. E. Growan.

Mr. J. H. Polglase, *Associate Member*, left England last month to resume his position in Malaya as assistant manager to the Sungei Besi Mines, Ltd.

Mr. J. S. Proud, *Associate Member*, has left South Africa for Canada via England.

Mr. D. H. Shute, *Associate Member*, expects to return shortly to England from the U.S.A.

Mr. D. Simpson, *Student*, is now assistant to the Head of Coal Preparation Research, National Coal Board, Cheltenham, Gloucestershire.

Mr. R. H. Skelton, *Member*, has returned to England from Ceylon.

Mr. N. R. Spendlove, *Student*, is now in Northern Rhodesia, having taken up an appointment with Rhokana Corporation, Ltd.

Mr. K. L. G. Terrell, *Associate Member*, is now engaged as surveyor and assayer with the Nanwa Gold Mines, Ltd., Gold Coast Colony.

Mr. P. A. W. Thuell, *Student*, is home on leave from Malaya.

Mr. D. V. G. Tregaskis, *Student*, has left England for India to take up the appointment of mining assistant to the Indian Copper Corporation, Ltd.

Mr. J. K. Walker, *Associate Member*, has recently returned to India to take up the appointment of underground agent for the Champion Reef and Ooregum Mine Joint Operation of the Kolar Gold Fields.

Mr. J. P. Warner, *Student*, has been appointed junior engineer in the Metallurgical Department of Falconbridge Nickel Mines, Ltd., Ontario, after taking the Ontario Mining Association's one-year postgraduate training course.

Mr. P. W. Watson, *Student*, is now in England, having returned from the Gold Coast.

Mr. J. T. M. White, *Member*, of the Malayan Mines Department, is visiting England before going to Australia.

Mr. G. Wilson, *Student*, is leaving the Transvaal to join the Geological Department, Kenya.

ADDRESSES WANTED

R. Milton Thomas	J. A. Cocking
K. A. Knight Hallowes	A. S. Rogers
D. S. Broadhurst	A. J. W. Walser

OBITUARY

Malcolm Fergusson died in Johannesburg on 14th August, 1949, at the age of 75. He was born in Norfolk, educated at Bedford Grammar School, and received his professional training at the Royal School of Mines from 1891 to 1894, graduating with the A.R.S.M. in mining. In 1895 he went to Johannesburg as sampler at the Robinson gold mine, and as assistant engineer with Consolidated Gold Fields of South Africa, Ltd., in the Heidelberg and Nigel Districts. From 1896 to 1898 Mr. Fergusson was manager of Central Lydenburg Goldfields, Pilgrims Rest, and then joined an expedition to Central Africa under the joint auspices of the Royal Society and Royal Geographical Society. In 1901 he took up the appointment of assistant manager of Ashanti Goldfields Corporation, Gold Coast, but in the following year returned to South Africa to serve as Inspector of Mines for the Transvaal and the Union of South Africa for fifteen years. In 1927 he was promoted Chief Inspector of Mines of the Union of South Africa, and retired from that position in 1930. He then became resident mining engineer of the British South Africa Co. in the Rhodesian copper belt and remained at Ndola until 1937, when he went to live in Johannesburg. He retained his interest in mining and was a director of Apex Mines, Ltd., at the time of his death.

Mr. Fergusson was elected to Membership of the Institution in 1933.

ARTHUR WARNER HOOKE died at Salisbury, Wiltshire, on 18th October, 1949, at the age of 68. After studying mining and mechanical engineering from 1898 to 1900 at Sydney Technical College he had four years' practical experience in various mines in New South Wales, including Peak Hill Proprietary mines and Commonwealth Gold Extraction Co. He was assistant manager and surveyor at Wolumla Gold Mines, Ltd., in 1906 and then for six months held the appointment of manager. In 1907 he became sub manager and assayer to Tasmanian Consols, Ltd., and in the following year was made general manager. From 1909 to 1910 Mr. Hooke held the position of manager of Montana Tin Syndicate, Tasmania, and later superintended sampling at the Dreadnought mine for two months. He then joined Juga (Nigeria) Tin and Power Co., Ltd., as manager, and at the same time was in charge of Lucky Chance Mines, Ltd., until 1913, when he was made manager of Forum River (Nigeria) Tin Co., Ltd. Three years later he accepted the additional appointment of manager of Bisichi Tin Co. (Nigeria), Ltd., and in 1918 was also consulting engineer to Ninghi (Nigeria) Tin Co., Ltd. From 1923 to 1924 Mr. Hooke was general manager of Batura and Monguna tin mines, Nigeria. In 1932 he carried out examinations in Guiana, and in the following year was appointed general manager of Lower Ancobra (Gold Coast) Areas. From 1935 he served for five years as general manager of Van Emden (Dutch Guiana) Gold Mines, Ltd. In 1942 he returned to Bisichi as general manager until 1947, taking over the managership of Ashanti Adowsena Goldfields, Ltd., in 1948. He had only recently returned to England from the Gold Coast.

Mr. Hooke was elected to Associateship of the Institution in 1908, and was transferred to Membership in 1920.

THOMAS CAMPBELL SCRUTTON died in London on 27th September, 1949, at the age of 72. From 1895 to 1896 he took the mechanical engineering course at University College, London, and during the following year worked in the engineering shops of Messrs. Coubro and Scrutton at Millwall. He then studied chemistry, metallurgy, mineralogy and mining at the Royal School of Mines for two years and in 1899 went to Sarawak prospecting for the Borneo Co., Ltd. In 1901 he became assistant manager of the Bidi cyanide works of that company, taking over the management in 1904. Later that year he joined Malay Exploration Syndicate as manager of development and prospecting operations on tin properties in Pahang, and from 1906 to 1908 was manager of Legeh Concessions Syndicate which operated in Legeh, Rahman and Siam. Mr. Scrutton then worked for seven years in the Congo in charge of prospecting and exploration for Cie. des Chemins de Fer Grandslacs, of Brussels. He spent some time in Uganda, and from 1921 to 1930 was again working in the Belgian Congo. From 1931 onwards he was in practice in London as a consultant, and was a director of many companies, including Lupa Exploration Syndicate, Ltd., Northern Transvaal Goldfields, Ltd., Watende, Ltd., Marsman Metal & Trading Co., Ltd., Anglo Eastern Tin, Ltd., Anglo Portuguese Diatomite Corporation, Ltd., and Hollybush Trust, Ltd.

Mr. Scrutton was elected to Studentship of the Institution in 1899, was transferred to Associateship in 1902 and to Membership in 1909. He served as Member of Council for the three sessions 1935–38. He contributed a paper to the *Transactions* of the Institution entitled *Notes on the occurrence and treatment of gold ore at Bidi, Sarawak* (vol. 15, 1905–6).

BOOK REVIEW

Powder metallurgy in Germany during the period 1939-1945. By R. A. HETZIG. B.I.O.S. Overall Report No. 20. London: H.M.S.O., 1949. 27 p. 6d.

This report provides an excellent appraisal of German powder metallurgy during the war years.

Powder metallurgy made two major contributions to the German war effort, the sintered iron driving band and hard metal for cutting-tools and armour-piercing shot. Powdered iron was also used to some extent in small arms components and bearings. Shortage of copper restricted development in this field. The work on the lightweight accumulators with plates of sintered nickel and cupro nickel, although not entirely successful, is of interest.

The report gives a good picture of the size of the industry and of its rapid growth during the war. In 1944, for instance, the Krupp output of tips for cutting tools alone reached 500 tons. Nearly 700 tons of armour-piercing shot was produced in 1940, although the severe tungsten shortage prevented this figure being approached in subsequent years. In contrast to this, shortage of copper stimulated the production of sintered iron driving bands, an output exceeding 3,000 tons per month being reached in 1944. Aluminium flake powder production amounted to 12,000 tons yearly, with an addition of 10,000 tons of atomized aluminium powder.

In mining applications considerable progress was made in Germany during the war on percussive rockdrills tipped with hard metal, although undoubtedly a contributory factor in their success lay in the use by the Germans of light, fast-hitting machines. In spite of the serious shortage of tungsten almost all chain coal-cutters were tipped with hard metal.

In general, German powder metallurgy during the war produced no outstanding innovations, with the possible exception of iron powder manufacture. The industry developed in size and importance mainly through persistent refinements in technique in individual processes, and in improvements in equipment.

The report concludes with an excellent index to the official published reports and other documents available in England on this branch of German industry.

D. H. SHUTE

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*Books (excluding works marked *) may be borrowed by members personally or by post from the Librarian, 424, Salisbury House, London, E.C.2*

Books and Pamphlets:

Bergbau-Archiv, Band 10. Essen: *Gluckauf*, 1949. 236 p., illus., diagrs., tabs.

Canadian Mining Manual, 1949. Gardenvale, P.Q.: National Business Publications Ltd., 1949. 349 p., illus., diagrs., tabs., flowsheets.

IMPERIAL CHEMICAL INDUSTRIES LTD., *Gold extraction for the small operator, 2nd ed.* London: I.C.I., 1949. 103 p., diagrs., tabs.

MARCOSSON, Isaac F. *Metal magic: the story of the American Smelting and Refining Company.* New York: Farrar, Straus and Company, 1949. 313 p., illus. $4.50.

* *Metal industry handbook and directory 1949.* London: Cassier, 1949. 476 p., illus., diagrs., tabs.

SPALDING, Jack. *Deep mining: an advanced textbook for graduates in mining and for practising mining engineers.* London: Mining Publications Ltd., 1949. 405 p., illus., diagr., tabs., glossary, 40s.

Government Publications:

NATIONAL COAL BOARD. *Quarterly and annual statistical statement of the costs of production, proceeds and profit or loss of collieries for the fourth quarter of 1948 and for the year 1948.* London: The Board, June 1949. 14 p., tabs. 6d.

NATIONAL COAL BOARD. *Report and accounts for 1948.* London: H.M.S.O., June 1949. 299 p., tabs. 6s. 6d.

VICTORIA, DEPT. OF MINES. *Annual report including gold and mineral statistics and boring records for the year 1948.* Melbourne: Govt. Printer, 1949. 51 p., tabs.

Proceedings and Reports:

AMERICAN INSTITUTE OF MINING AND METALLURGICAL ENGINEERS. *Transactions, vol. 179, petroleum development and technology, 1949.* N.Y.: The Institute, 1949. 395 p., illus., diagrs., tabs.

Maps:

Banffshire, Eastern Elginshire, Northern and Central Aberdeenshire, and parts of

East Inverness-shire and North Kincardineshire. Geological survey of Scotland. Sheet 9. Scale 4 ml. = 1 in. Southampton : Ordnance Survey Office, 1948. 3s.

Brampton. Geological survey of England and Wales. Sheet 18 (solid). Scale 1 in. = 1 ml. Southampton Ordnance Survey Office, 1949. 5s.

Brampton. Geological survey of England and Wales. Sheet 18 (drift). Scale 1 in. = 1 ml. Southampton : Ordnance Survey Office, 1949. 5s.

Carlisle. Geological survey of England and Wales. Sheet 17 (drift). Scale 1 in. = 1 ml. Southampton : Ordnance Survey Office, 1949. 5s.

Carp Lake, Cariboo district, British Columbia. Geological survey, map 979A. Scale 1 in. = 1 ml. Ottawa : Dep. of Mines and Resources, 1949.

Carp Lake, Cariboo district, British Columbia. Geological survey, map 980A. Scale 1 in. = 1 ml. Ottawa : Dep. of Mines and Resources, 1949.

Dudley. Geological survey of England and Wales. Sheet 167 (solid with drift). Scale 1 in. = 1 ml. Southampton : Ordnance Survey Office, 1949. 5s.

Dumfriesshire, Central and Eastern, Roxburghshire, Central and Southern portions, Selkirkshire, Central and Southern portions, Peeblesshire, S. part of Lanarkshire, portion of S.E. Geological survey of Scotland. Sheet 17. Scale 4 ml. = 1 in. Southampton : Ordnance Survey Office, 1948. 3s.

Fifeshire, Linlithgowshire, E. part Edinburghshire, Haddingtonshire, Berwickshire, Roxburghshire, N. part Selkirkshire, N. part Peeblesshire, North and Central Lanarkshire. Geological survey of Scotland. Sheet 15. Scale 4 ml. = 1 in. Southampton : Ordnance Survey Office, 1948. 3s.

Forfarshire, parts of Aberdeenshire, Kincardineshire, Perthshire and Fifeshire. Geological survey of Scotland. Sheet 12. Scale 4 ml. = 1 in. Southampton : Ordnance Survey Office, 1948. 3s.

Goldfields, Saskatchewan. Geological survey preliminary map 49-17. Scale 1 in. = ½ ml. Ottawa : Dept. of Mines and Resources, 1949. 2 sheets. (Preliminary account.)

Inverness-shire (S. part), Argyllshire (N.E. part), Cona Glen, Glen Tarbert, Loch Linnhe, Fort William, Ben Nevis, Ballachulish, Appin, Glen Etive. Geological survey of Scotland. Sheet 53 (solid with drift). Scale 1 in. = 1 ml. Southampton : Ordnance Survey Office, 1948. 3s. 9d.

Lizard. Geological survey of England and Wales. Sheet 359 (drift). Scale 1 in. = 1 ml. Southampton : Ordnance Survey Office, 1949. 5s.

Manchester. Geological survey of England and Wales. Sheet 85 (solid). Scale 1 in. = 1 ml. Southampton : Ordnance Survey Office, 1949. 5s.

Orkney Islands. Geological survey of Scotland. Sheet 3. Scale 4 ml. = 1 in. Southampton : Ordnance Survey Office, 1948. 3s.

Stirlingshire, Dumbartonshire, Renfrewshire, with parts of Lanarkshire, Ayrshire, Buteshire and Argyllshire. Geological survey of Scotland. Sheet 14. Scale 4 ml. = 1 in. Southampton : Ordnance Survey Office, 1948. 3s.

Sutherlandshire, Caithness (West), and Ross and Cromartyshire (N.W. part). Geological Survey of Scotland. Sheet 5. Scale 4 ml. = 1 in. Southampton : Ordnance Survey Office, 1948. 7s. 6d.

Wem. Geological survey of England and Wales. Sheet 138 (drift). Scale 1 in. = 1 ml. Southampton : Ordnance Survey Office, 1949. 5s.

Weymouth. Geological survey of England and Wales. Sheet 342 (drift). Scale 1 in. = 1 ml. Southampton : Ordnance Survey Office, 1949. 5s.

Wigtownshire, Kirkcudbrightshire, Ayrshire, S. half, Dumfriesshire, Western portions, Lanarkshire, portions of S. part. Geological survey of Scotland. Sheet 16. Scale 4 ml. = 1 in. Southampton : Ordnance Survey Office, 1948. 3s.

Wolverhampton. Geological survey of England and Wales. Sheet 153 (drift). Scale 1 in. = 1 ml. Southampton : Ordnance Survey Office, 1948. 5s.

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553(595)

The mineral resources of Malaya and other Far Eastern countries. Lewis Leigh Fermor.—*4th Emp. Min. Metall. Congr.*, Lond., Pap. AB 4, July 1949, 31 p., tabs. (Paper 2s. 6d.)

553(595.17)

The mineral resources of Malaya and other Far Eastern countries: Christmas Island. Lewis Leigh Fermor.—*4th Emp. Min. Metall. Congr.*, Lond., Pap. AB 4, July 1949, 31 p., tabs. (Paper 2s. 6d.)

553(64)

Derzeitiger Stand und Aussichten der Bergwirtschaft in französisch Marokko. (Present situation and prospects of mining in French Morocco—description of deposits and ores.) Friedrich Gloeckner.—*Gluckauf*, Essen, 85, August 27 1949, 653–6, map, tabs. 4 DM.

553(664)

The mineral resources of the British West African colonies: Sierra Leone. N. R. Junner.—*4th Emp. Min. Metall. Congr.*, Lond., Pap. AB 5, July 1949, 16 p., maps, tabs., biblio. (Paper 2s.)

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The mineral resources of the British West African colonies: Gold Coast. N. R. Junner.—*4th Emp. Min. Metall. Congr.*, Lond., Pap. AB 5, July 1949, 16 p., maps, tabs., biblio. (Paper 2s.)

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The mineral resources of the British West African colonies. N. R. Junner.—*4th Emp. Min. Metall. Congr.*, Lond., Pap. AB 5, July 1949, 16 p., maps, tabs., biblio. (Paper 2s.)

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The mineral resources of the East African colonies: Uganda. Edmund O. Teale.—*4th Emp. Min. Metall. Congr.*, Lond., Pap. AB 6, July 1949, 20 p., map, tabs., biblio. (Paper 2s.)

553(676.2/.9)

The mineral resources of the East African colonies: Kenya. Edmund O. Teale.—*4th Emp. Min. Metall. Congr.*, Lond., Pap. AB 6, July 1949, 20 p., map, tabs., biblio. (Paper 2s.)

553(678.2/.9)

The mineral resources of the East African colonies. Edmund O. Teale.—*4th Emp. Min. Metall. Congr.*, Lond., Pap. AB 6, July 1949, 20 p., map, tabs., biblio. (Paper 2s.)

553(68.01)

Some notes on the mineral resources (excluding coal) of the Union of South Africa. A. R. Mitchell. —*4th Emp. Min. Metall. Congr.*, Lond., Pap. AB 2, July 1949, 12 p., tabs. (Paper 1s. 6d.)

553(681)

The mineral resources of other Empire territories: Bechuanaland. E. H. Beard.—*4th Emp. Min. Metall. Congr.*, Lond., Pap. AB 9, July 1949, 30 p. tabs. 1s. (Typescript.)

553(682.8)

Notes on portion of the far western bushveld igneous complex, Marico district, and possibilities respecting nickel-copper chromitite and other minerals.—*Industr. Miner.*, Pretoria, Jan.–March 1949, 16–24, map, tab. 5s.

553(683.4)

The mineral resources of other Empire territories: Swaziland. E. H. Beard.—*4th Emp. Min. Metall. Congr.*, Lond., Pap. AB 9, July 1949, 30 p., tabs. 1s. (Typescript.)

553(689.1)

Significant changes in the known position of the mineral resources of Southern Rhodesia during the past 20 years. J. C. Ferguson.—*4th Emp. Min. Metall. Congr.*, Lond., Pap. AB 8, July 1949, 9 p., tabs. (Paper 1s.)

553(689.4)

The mineral resources of other Empire territories. (Northern Rhodesia, Trinidad, British Guiana, Cyprus, Jamaica, Swaziland, Nyasaland, and Bechuanaland.) E. H. Beard.—*4th Emp. Min. Metall. Congr.*, Lond., Pap. AB 9, July 1949, 30 p., tabs. (Typescript) 1s.

553(689.7)

The mineral resources of other Empire territories: Nyasaland. E. H. Beard.—*4th Emp. Min. Metall. Congr.*, Lond., Pap. AB 9, July 1949, 30 p., tabs. 1s. (Typescript.)

553(71)

Changes in Canada's mineral resources during the past ten years. G. C. Monture.—*4th Emp. Min. Metall. Congr.*, Lond., Pap. AB 3, July 1949, 18 p., map, tabs. (Paper 1s. 6d.)

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The mineral resources of other Empire territories: Jamaica. E. H. Beard.—*4th Emp. Min. Metall. Congr.*, Lond., Pap. AB 9, July 1949, 30 p., tabs. 1s. (Typescript.)

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The mineral resources of other Empire territories: Trinidad. D. H. Beard.—*4th Emp. Min. Metall. Congr.*, Lond., Pap. AB 9, July 1949, 30 p., tabs. 1s. (Typescript.)

553(881)

The mineral resources of other Empire territories: British Guiana. E. H. Beard.—*4th Emp. Min. Metall. Congr.*, Lond., Pap AB 9, July 1949, 30 p., tabs. 1s. (Typescript.)

553(911.1)

The mineral resources of Malaya and other Far Eastern countries: British Borneo. Lewis Leigh Fermor.—*4th Emp. Min. Metall. Congr.*, Lond., Pap. AB 4, July 1949, 31 p., tabs. (Paper 2s. 6d.)

553(94)

Mineral industry of Australia with particular reference to the past twenty years. P. B. Nye and others.—*4th Emp. Min. Metall. Congr.*, Lond., Pap. AB 1, July 1949, 18 p., map, tabs. (Paper 2s.)

553.1 Determination, properties, of economic minerals

553.1 : 53 ?·312

Determination of reflectivity of the ore minerals. R. E. Folinsbee.—*Econ. Geol.*, Lancaster, Pa., 44, August 1949, 425–36, illus., tabs., biblio. $1.

553.12 : 553.411

Utilization of waste slag in gold assaying. L. A. Goedbloed.—*Canad. Min. J.*, Gardenvale, P.Q., 70, August 1949, 81–2. $1.

553.2 Ore deposition

553.2 : 553.412.1

Deposition and resolution of native silver at Gowganda, Ontario. Edson S. Bastin.—*Econ. Geol.*, Lancaster, Pa., **44**, August 1949, 437–44, illus. $1.

553.3/.4 Metalliferous deposits

553.3/.9 : 667.6

Use of minerals in the paint industry.—*Indian Miner.*, Delhi, **2**, Jan. 1948, 31–9. 3s.

553.311.1(713)

Magnetite in eastern Ontario. (Comparison with the Adirondacks deposits suggests similar deposits exist in Canada.) W. M. Bonham.—*Canad. Min. J.*, Gardenvale, P.Q., **70**, August 1949, 57–60, maps, illus. $1.

553.32(541.1)

The manganese ore deposits of the Jamda-Koira valley. E. Spencer.—*Trans. Min. Geol. Inst. India*, Calcutta, **44**, Oct. 1948, 61–77, illus., map, tabs. Rs. 4s.

553.32(687)

Manganese. (Deposits in N.W. of Cape Province—analysis, production, etc.)—*Industr. Miner.*, Pretoria, Jan.–March 1949, 13–16, tabs. 5s.

553.411(713.7–Hard Rock)

The geology of Hard Rock Gold Mine's quartz stringer ore zones. Harold F. Morrow.—*Precambrian*, Winnipeg, **22**, Sept. 1949, 11–15; 39, diagrs.

553.412.1 : 553.2

Deposition and resolution of native silver at Gowganda, Ontario. Edson S. Bastin.—*Econ. Geol.*, Lancaster, Pa., **44**, August 1949, 437–44, illus. $1.

553.445(682.8)

Lead. (Deposits in Marico, Springs and Hay districts.)—*Industr. Miner.*, Pretoria, Jan.–March 1949, 10–13, tabs. 5s.

553.461(541.4)

The chromite deposits of Nuasahi, Keonjhar state, Eastern States Agency, India. S. K. Borooah.—*Trans. Min. Geol. Inst. India*, Calcutta, **44**, Oct. 1948, 79–90, map, biblio. Rs. 4s.

553.461(682.8)

Notes on portion of the far western bushveld igneous complex, Marico district, and possibilities respecting nickel-copper chromitite and other minerals.—*Industr. Miner.*, Pretoria, Jan.–March 1949, 16–24, map, tab. 5s.

553.463(786.32–Henderson Gulch)

Investigation of Henderson Gulch tungsten deposits, Granite county, Mont. Robert J. Hundhausen.—*U.S. Bur. Min. Rep. Invest.* 4513, July 1949, 8 p., maps, diagrs.

553.48(682.8)

Notes on portion of the far eastern bushveld igneous complex, Marico district, and possibilities respecting nickel-copper chromitite and other minerals.—*Industr. Miner.*, Pretoria, Jan.–March 1949, 16–24, map, tab. 5s.

553.492.1(546.1)

Bauxite deposits of Riasi, Kashmir state.—*Indian Miner.*, Delhi, **1**, July 1947, 186–7. 3s.

553.492.1(758)

Investigation of the Irwinton bauxite district, Wilkinson county, Ga. William A. Beck.—*U.S. Bur. Min. Rep. Invest.* 4495, Wash., D.C., July 1949 16 p., maps, tabs.

553.492.7(54)

Kyanite in India.—*Indian Miner.*, Delhi, **2**, Jan. 1948, 40–4, tabs. 3s.

553.551.1(797.41–Whitehorse)

Investigation of Whitehorse limestone deposits, Snohomish county, Wash. Constantine C. Popoff.—*U.S. Bur. Min. Rep. Invest.* 4510, Wash., D.C., July 1949, 9 p., map, tabs., diagrs.

553.6 Non-metallic deposits

553.62 : 66.02

Fused silica; its properties and some uses. A. E. Williams.—*Min. J.*, Lond., **233**, Sept. 24 1949, 884–6, illus., diagr. 8d.

553.641(68.01)

Phosphates. Resources in the Union of South Africa.—*Industr. Miner.*, Pretoria, Jan.–March 1949, 54–8. 5s.

553.641(68.01)

Union's phosphate-bearing rocks; survey of known occurrences.—*S. Afr. Min. Engng. J.*, J'burg, **60**, Pt. 2, Sept. 17 1949, 57; 65. 6d.

553.661.21(761.24)

Investigation of copper-bearing pyrite ores, Pyriton, Clay county, Ala. H. D. Pallister and J. R. Thoenen.—*U.S. Bur. Min. Rep. Invest.* 4494, Wash., D.C., July 1949, 15 p., maps, tabs., diagrs.

553.673(764)

Investigation of the Alfred Davis soapstone deposits, Gillespie county, Tex. W. D. McMillan.—*U.S. Bur. Min. Rep. Invest.* 4509, Wash., D.C., July 1949, 9 p., map, tabs., diagrs.

553.677(541.4)

Mica in the western states of Orissa, north of the Mahanadi river. H. Crookshank.—*Indian Miner.*, Delhi, **1**, July 1947, 169–73. 3s.

553.678.1(764)

Investigation and laboratory testing of vermiculite deposits, Llano county, Tex. W. D. McMillan and A. W. Gerhardt.—*U.S. Bur. Min. Rep. Invest.* 4486, Wash., D.C., July 1949, 42 p., illus., maps, tabs., diagrs.

553.678.1 : 621.74

The use of vermiculite in foundry work. B. Haigh.—*Min. Industr. Mag. S. Afr.*, J'burg, **39**, August 1949, 449. 1s.

553.9 Carbonaceous deposits

553.982(54)

Petroleum resources of India: Pt. 1. D. N. Wadia.—*Indian Miner.*, Delhi, **1**, Oct. 1947, 221–7, maps, diagrs.

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The toxic properties of beryllium. H. Manley.—*Mine & Quarry Engng.*, Lond., **15**, Oct. 1949, 318. 1s. 6d.

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How mining men are using electronics today: Pt. 3. J. A. Setter.—*Engng. Min. J.*, N.Y., **150**, Sept. 1949, 85–7, illus.

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›tions into cyclone dust collectors. A. J. —*J. Proc. Instn. Mech. Engrs.*, Lond., 1949, 233–45, illus., diagrs.

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Gold Concentration at the Amalgamated Banket Areas Reduction Plant*

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A paper published on 10th November, 1949, by the Institution of Mining and Metallurgy, to be submitted for discussion at the November General Meeting.

SYNOPSIS

The principal subject of the present paper is a description of mechanical methods which have been introduced in place of straking for the recovery of free gold at the reduction plant of Amalgamated Banket Areas, Ltd. Relevant to gold concentration reference is also made to the mineralogical characteristics of the banket ore; the general arrangement of plant; security aspects; overall performance of the plant when straking was practised and as now operating; the association of gold with ore minerals in relation to the necessity for selective regrinding of such minerals, and alternatives to concentration.

INTRODUCTION

THE property of Amalgamated Banket Areas, Ltd., is at Tarkwa, 39 miles by rail from the modern port of Takoradi in the Gold Coast, and includes two of the original (and in their day prominent) mines located and worked on the banket lode—Tarkwa and Abbontiakoon. Whereas the descriptive term 'banket' originated in the Transvaal and was later applied to the auriferous conglomerates of the Gold Coast, it is of interest that the latter were, in fact, the first discovered (in 1877) and developed, some seven to eight years before the discovery and exploitation of the more famous and extensive Witwatersrand field.

Two main classes of ore are being received at the Amalgamated Banket Areas plant for treatment—deep-mine unaltered banket, and lower-grade weathered ore from a flat anticlinal banket capping at Pepe, which is mined by opencast methods. The two are treated in parallel, but practically distinct, circuits. It is with the treatment of the underground ore that the present paper is mainly concerned.

THE BANKET ORE

The Gold Coast 'banket' has been described in memoirs of the Gold Coast Geological Survey. Some of the principal characteristics are briefly mentioned here in their relevance to the metallurgical problem.

The conglomerate is comprised substantially of ellipsoidal water-worn pebbles of quartz, some clear and glassy, others white and opaque. Occasional pebbles of amethystine quartz and of pink felsite are also present. The pebbles vary in size over an average range of one to three inches in length, although smaller and larger pebbles are encountered. Distributed through the pebble interstices are sand grains and small crystals of specular haematite, with minor amounts of ilmenite and magnetite. The whole has been cemented by silicification into an

* Paper received on 23rd June, 1949.

† Consulting metallurgist, West African Gold Corporation, Ltd.

extremely hard compact conglomerate superficially resembling the Witwatersrand banket.

The haematite, ranging up to 7 per cent by weight of the ore, is of primary origin and persists to the deepest levels (2,500 ft. at Taquah and Abosso Mines, Ltd.) yet worked. The gold, which is unevenly distributed through the secondary quartz matrix, is crystalline and in extremely fine grains, being rarely visible to the unaided eye. Sulphides are so seldom encountered as, for metallurgical purposes, to be considered absent. The quartz pebbles are substantially barren. There is microscopic evidence of occasional occurrence of gold in haematite itself, but our investigations have shown that this is of minor significance compared with the proportions that obtain for gold in the pyrite of the Witwatersrand banket.

The underground ore treated at the plant is comprised of banket, banded auriferous haematite-quartzite, and varying dilutions of quartzite wall rocks. As received the ore ranges from 4 to 6 dwt./ton*.

FORMER PRACTICE—STRAKING

The flow-sheet to which the plant originally operated is set out in Fig. 1 (Plate I), which includes descriptive data covering the principal items of equipment. The original layout, it will be noted, included straking for recovery of free gold, the installation being set in a strake house 315 ft. long by 22 ft. wide (the central bay of which is 70 ft. long by 30 ft. wide) and in this were accommodated the strake washing boxes.

The strakes were each 10 ft. by 1 ft. 6 in. wide, arranged in eight banks of sixteen strakes each. With four primary and four secondary 8-ft. diameter by 6-ft. ball-mills installed, one bank of strakes was available for the treatment of the discharge from each ball-mill.

As originally operated Pepe (weathered) ore was also straked, but there have been changes in Pepe ore treatment, one of which has been the discontinuation of straking. Recovery of gold from the main bulk of this weathered low-grade ore is now effected by cyanidation only.

One European and twelve Africans per shift were required for supervision and changing and washing of strakes. The installation operated efficiently to recover 52 per cent of the total gold from underground ore treated in the plant.

Originally the stored strake concentrates were reconcentrated on the day shift on the Wilfley table, from which a deep-cut concentrate was removed for working up in the amalgam barrels. Later amalgamation was discontinued and a novel procedure was adopted, accumulated Wilfley table concentrate being fed to two small flotation machines with additions of xanthate and pine oil. By this means a clean gold concentrate suitable for drying and direct smelting was floated in a froth of unique beauty, recoveries closely approximating those by amalgamation. Such a procedure was practicable due to the average fineness of the free gold and to the absence of sulphide minerals from the original ore.

Set out in Table I is a metallurgical balance compiled from recorded data for the nine months of operation immediately prior to the suspension

* All assays refer to the ton of 2,000 lb.

at this plant in March, 1943, under the Concentration Scheme, ne arrangement whereby, with the supplies of diesel oil and res restricted by the emergency and Government action, certain spended operation in order that the others might continue to t their normal rates of production.

TABLE I

ALLURGICAL BALANCE PRACTISING STRAKING AND AMALGAMATION

Product	Weight %	dwt. per ton	% of Total Gold
....	100·0	4·417	100·0
by straking and amalgamation ...	100·0	2·297	52·0
by cyanidation	100·0	1·853	42·0
ing	100·0	2·120	48·0
anidation	48·6	2·730	30·0
/anidation	51·4	1·543	18·0
ue—Total	48·6	0·329	3·6
lue—Total	51·4	0·209	2·4
IDUE	100·0	0·267	6·0
COVERY	100·0	4·150	94·0

GRADING ANALYSES (I. M. M.)

Product	%
ne : + 100	67·3
- 100 + 200	14·3
— 200	18·4
ue : + 100	5·5
- 100 + 200	11·1
— 200	83·4

riod of operation has been chosen for present purposes as, during months, the plant was milling underground ore only, treatment re having been suspended. It therefore affords data on straking lgamation comparable with current concentrating practice, being applied to the whole of the deep mine ore *plus* a proportion

(15 to 18 per cent of the total) of low-grade coarse ore from the Pepe workings.

THE SECURITY PROBLEM

Metallurgically the concentrating layout and operations briefly described yielded very good results. The snag was the temptation offered to gold thieves and the depressing prison-like conditions for the operating shift in the locked strake house.

As its name might imply the Gold Coast has an historical association with the winning and working of gold which dates back some centuries before the arrival on the scene of the European miner. There is in the country an ancient, widely-developed, and skilled guild of African goldsmiths working under Government licence. Gold ornament figures prominently among the insignia of chiefs and in the adornment of African women and children. International borders are not so very far distant from most centres of gold production and the dense bush and forest preclude close control. Minor gold occurrences are fairly frequent apart from the major occurrences being worked by European enterprise and Africans still win gold with the traditional calabash and more recently acquired prospector's pan, the adept panners being the women. Legislation regarding the possession of unwrought gold is nothing like so restrictive as in other comparable fields.

Thus it will be appreciated that there is probably no other field in Africa in which the local population in general is so gold conscious and in which the identity of illicit gold can be so simply and completely lost. Analyses of detected thefts of gold at a majority of the producing mines showed that over 90 per cent of the surface thefts and a considerably higher proportion of the value, were of gold in concentrate. Examination of the records summarized in Table I disclosed that under-recoveries of gold by straking and amalgamation were reported in seven of the nine months, two months only reflecting small over-recoveries. Based on the head sample assay of the ore entering the plant the indicated shortage on gold call over the whole plant for the period was 2·8 per cent of gold received in ore.

Since the war losses from plants due to theft have been reduced to minor proportions by the more thorough organization of security measures and the exercise of unremitting vigilance. Apart from these preventive measures the author has felt that the surer safeguard and the greater peace of mind would result from modifications of metallurgical practice by which high-grade intermediary products in process were kept out of sight and reach of the potential thief. The problem is not peculiar to this field, although the complexities which attend it possibly are.

Such then is the background against which a variation from the orthodox and metallurgically efficient but, in other respects, vulnerable and labour-consuming practice of straking and amalgamation is to be viewed.

PRESENT PRACTICE—MECHANICAL CONCENTRATION

The alternative adopted was to use continuous discharge Pan American jigs in the primary mill-classifier circuit and the comparatively uncommon and, it is to be feared, neglected Johnson concentrator, in the secondary mill-classifier circuit. Both are machines of low capital and operating cost. The savings on labour alone in one year of operation were sufficient to meet the cost of installation of jigs and concentrators. Further, the installation required has been comfortably accommodated with adequate room for extension in a portion of the central 70-ft. by 30-ft. bay of the strake house. For a similar layout installed *ab initio* a capital economy in building accommodation over that required for straking is thus implied.

Fig. 2 (Plate II) shows the original strake installation in plan and section, and in Fig. 3 (Plate III) the new Jig-Johnson concentrator installation is set out in plan and section. The unit illustrated in Fig. 3—i.e. two primary ball-mills, two jigs, one secondary ball-mill, and two Johnson concentrators is treating a feed of *minus* ¼-in. banket ore at the rate of 900 tons per 24 hours.

The flow-sheet to which the concentrating section is now operating is set out in Fig. 4.

The jig and Johnson concentrates discharge via pipes through the concrete floor to the concentrate storage set in the gold room below. The concentrates are out of sight and reach and are never handled. The jigs and Johnson concentrators operate behind a locked door, the machines being visited periodically during shifts by the European mill shift-bosses. The stored concentrates from both machines are reconcentrated on the morning shift by passage over a Wilfley table, from which a clean gold concentrate suitable for direct smelting is recovered. Wilfley tailings pass over two strakes each 5 ft. by 1 ft. 9 in. wide before returning to the milling circuit. The strake concentrate, very small in bulk, is stored in the strong room and twice a month is separately reconcentrated on the Wilfley table.

The installation operates smoothly with the minimum of attention. With the jigs operating alone and no concentration step in the secondary mill circuit the recovery of free gold into bullion averages 25 per cent of the gold in ore, the finer gold escaping to the cyanide plant. With the Johnson concentrators also operating in the secondary mill circuit gold recovery is averaging 47·8 per cent made up as follows :

	per cent
Recovery from jigs	25·0
Recovery from Johnson concentrators (30·4 per cent of 75 per cent)	22·8
	47·8

Typical results of plant performance with the new concentrating installation are set out in Table II, which is compiled from current operating records. *The results in this Table are more or less directly comparable with those set out in Table I. There are qualifications : First, the ore feed to the section was comprised of 83·5 per cent of underground ore averaging 4·554 dwt./ton and 16·5 per cent of weathered coarse ore from Pepe averaging 1·906 dwt./ton. The concentratable gold content of the weathered ore is low, being of the order 10 to 15 per cent. Further, only 83 per cent of the ore milled passed the jigs, the remaining 17 per cent

*February and March, 1949.

having been milled in a third primary ball-mill which is not at the moment in circuit with the jigs. The product from the secondary stage of grinding of the 17 per cent did pass to the Johnson concentrators.

There have been improvements in cyanidation practice which, while falling outside the scope of this paper, are to be borne in mind when comparing total recoveries in Tables I and II.

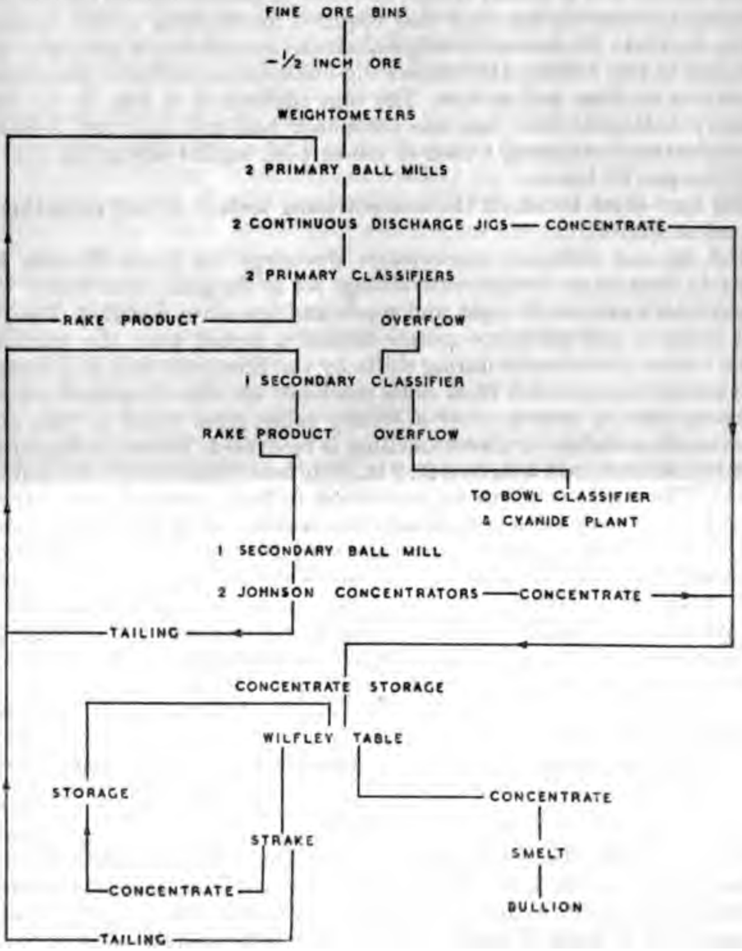


FIG. 4.—Ball-mill classifier and gold concentration circuit.

By taking a larger concentrate from the Wilfley table and working this up by amalgamation, gold recovery by concentration no doubt could be improved. Also by increasing the amount of concentrate drawn from the Johnson concentrators (see Table III), and running the Wilfley table for a

TABLE II

METALLURGICAL BALANCE PRACTISING CONCENTRATION

Product	Weight %	dwt. per ton	% of Total Gold
feed	100·0	4·117	100·0
very by concentration	100·0	1·967	47·8
very by cyanidation	100·0	2·041	49·6
entration tailing	100·0	2·150	52·2
l to cyanidation	44·4	2·849	30·7
e to cyanidation	55·6	1·592	21·5
l residue—Total	44·4	0·149	1·6
e residue—Total	55·6	0·077	1·0
l Residue	100·0	0·109	2·6
l Recovery	100·0	4·008	97·4

GRADING ANALYSES (I. M. M.)

Product	%
residue : + 100	61·9
— 100 + 200	22·6
— 200	15·5
e residue : + 100	3·9
— 100 + 200	8·3
— 200	87·8

er period, or by installing a second Wilfley table, recoveries by concen-ion could be raised. In view of the performance of the cyanide plant he current concentration tailing neither of these courses has been idered necessary. It is of interest to add that the gold call on the t is now being regularly and satisfactorily met.

JOHNSON CONCENTRATOR RECOVERIES

he Pan American jigs (36 in. by 36 in.) are standard equipment, iliar in dredge operation. The Johnson concentrator is probably less

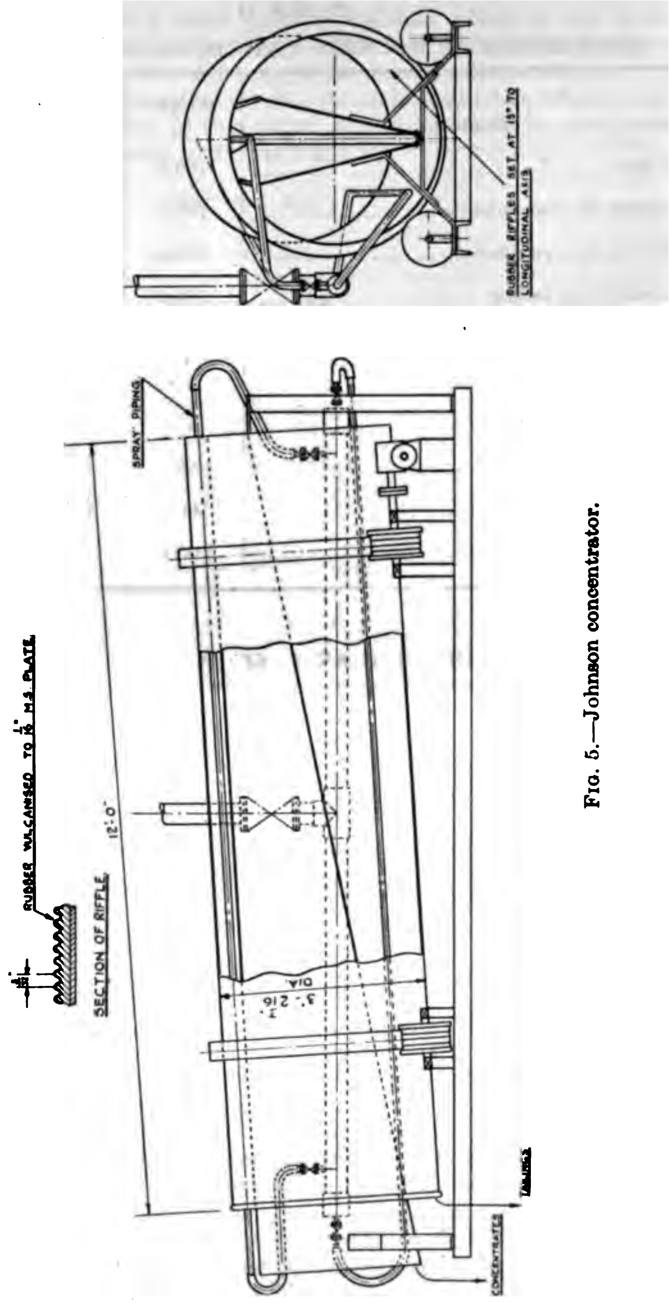


FIG. 5.—Johnson concentrator.

TABLE III
JOHNSON CONCENTRATION RECOVERIES UNDER VARYING CONDITIONS

Test no.	1	2	3	4	5	6
Drum, r.p.m.	7	5	7	5	5	7
% moisture in feed	35	33	35	30·5	40	34
Feed to concentrators						
*Tons per concentrator/24 hr.	283	324	250	448	303	332
dwt./ton	5·60	4·67	3·39	3·81	4·61	3·25
Concentrate						
Tons per 24 hr.	9·04	8·79	4·88	4·70	2·27	2·21
Weight—% of feed	3·20	2·72	1·95	1·05	0·75	0·67
dwt./ton	94·8	79·5	74·6	129·4	204·2	136·8
Recovery—dwt./ton of feed	3·03	2·16	1·45	1·36	1·53	0·92
Amalgamation of concentrate						
Tailing—dwt./ton	10·2	10·9	6·6	4·8	8·8	8·2
Tailing—dwt./ton of feed	0·33	0·30	0·13	0·05	0·07	0·05
Recovery—dwt./ton of feed	2·70	1·86	1·32	1·31	1·46	0·87
Concentrator tailing						
Weight—% of feed	96·8	97·28	98·05	98·95	99·25	99·33
dwt./ton	2·65	2·58	1·98	2·48	3·10	2·35
dwt./ton of feed	2·57	2·51	1·94	2·45	3·08	2·33
Recoveries						
(1) In Johnson concentrate: % gold in feed	54·1	46·2	42·9	35·6	33·2	28·0
(2) By amalgamation of concentrate: % gold in concentrate	89·2	86·3	91·1	96·3	95·7	94·0
(3) By amalgamation of concentrate: % gold in feed	48·2	39·8	39·1	34·3	31·7	26·3

*Includes circulating load.

familiar to many members. This machine was described* by that doyen of Rand metallurgists, Mr. E. H. Johnson, by whom the concentrator was developed. For convenient reference a drawing of the machine is included in Fig. 5.

In Table III the results from a series of controlled tests, in which conditions of operation of the Johnson concentrators were studied, are set out. Rotation was varied at speeds of 5 and 7 r.p.m. No appreciable difference in performance was noted. For normal operation the slower speed has been adopted. The percentage weight of concentrate was controlled by varying the amount of water used in the side sprays. Gold recovery was found to vary directly with the amount of concentrate made. A further variable possible with the machine operating on finer feeds is the inclination of the rotary drum. This has not yet been studied.

During the period of each of the tests recorded in Table III it is to be noted that Wilfley table tailings were not being returned to circuit. Each of the tests was of about three hours' duration. Concentrate recovered from each test was carefully riffled down to a sample of approximately 5,000 g., which was amalgamated without grinding. The concentrate values reported are built up from the gold recovered in amalgam *plus* gold found in the amalgamation tailing. It will be noted that an average of 90 per cent of the gold recovered in concentrate was amalgamable. Moreover, the assays of the tailings from amalgamation reflect but a minor lock-up of gold in haematite.

The grading analyses of products to and from the Johnson concentrator are set out in Table IV.

There is little doubt that the jigs could be excluded from the concentrating circuit and that with the Johnson concentrators operating alone on the secondary ball-mill discharge an equally satisfactory overall recovery of gold from the ore would be possible. There would almost certainly result a build-up in gold content of the rake product from the primary simplex classifiers which would almost as surely attract attention from the receivers' metallurgical scouts. It has been considered advisable to retain the jigs on their duty of recovering the coarser fraction of what is, on the average, very fine free gold.

The operation of the Johnson concentrator differs in several respects from South African practice. As described by E. H. Johnson, this machine operated in a single-stage tube-milling circuit, in which the size range of particles in the feed to the concentrator would be considerable. It was thus required to handle the total tonnage of ore under treatment including the circulating load. The operation of this machine on the reduced tonnage and finer sizings of a secondary milling circuit may be new. Its efficiency on these sizings has been impressive.

ENCASEMENT OF GOLD IN ORE MINERALS

On the Witwatersrand there is also an important association of gold enclosed in pyrite, and concentration of pyrite for separate selective

* Concentration and selective regrinding. *J. Chem. Soc. S. Afr.*, 27, April, 1927, 215.

TABLE IV

GRADING ANALYSES (TYLER)* OF PRODUCTS FROM JOHNSON CONCENTRATOR TESTS (TABLE III)

Test no.	1	2	3	4	5	6
	%	%	%	%	%	%
Feed						
+ 48 mesh ...	10·6	13·0	11·1	10·2	11·3	10·2
— 48 + 100 „ ...	30·2	34·7	29·3	31·0	31·5	31·0
— 100 + 200 „ ...	23·3	25·0	23·2	25·7	23·6	25·7
— 200 „ ...	35·9	27·3	36·4	33·1	33·6	33·1
Concentrate						
+ 48 mesh ...	2·1	2·3	3·3	2·8	2·7	2·9
— 48 + 100 „ ...	17·5	19·0	16·6	21·5	15·7	12·6
— 100 + 200 „ ...	45·6	45·7	42·9	43·6	46·5	41·8
— 200 „ ...	34·8	33·0	37·2	32·1	35·1	42·7
Tailing						
+ 48 mesh ...	10·4	12·7	10·9	10·0	11·8	10·0
— 48 + 100 „ ...	30·8	34·2	30·5	30·7	29·7	30·7
— 100 + 200 „ ...	23·2	25·6	23·0	26·1	23·4	26·1
— 200 „ ...	35·6	27·5	35·6	33·2	35·1	33·2

* An explanation of the appearance of a second grading series is necessary. Plant gradings have hitherto been conducted with I.M.M. sieves, but are shortly to be changed to the British Standard which closely approximates the Tyler series. At the West African Gold Corporation's laboratory the Tyler series is used.

regrinding and exposure of encased gold in preparation for cyanidation was an important function of the machine as described by E. H. Johnson. The author's investigations have not disclosed any such intimate association of gold with the haematite of the Gold Coast banket. At Amalgamated Banket Areas the first intention was to return Wilfley table tailings to the primary ball-mill feed, but in the light of experience it has been found sufficient for treatment requirements to return the Wilfley tailings to the secondary classifier so that haematite trapped in the concentrating section re-traverses the secondary stage of classification and grinding until size reduction has progressed to the point at which it escapes in the secondary classifier overflow.

The following examination of a representative sample of current Johnson concentrate illustrates the negligible encasement of gold in haematite :

Johnson concentrate—22·1 oz. Au per ton.
Grading Analysis (Tyler):

	+ 48 :	5·1 per cent
— 48	+ 100 :	16·4 per cent
—100	+ 200 :	28·2 per cent
	— 200 :	50·3 per cent

The concentrate was amalgamated without grinding and yielded:

By amalgamation : 21·9 oz. Au per ton concentrate.
Amalgamation tailing : 4·0 dwt. per ton.

The amalgamation tailing was ground in the laboratory ball-mill and re-amalgamated with the following result:

By amalgamation : 3·0 dwt. Au per ton concentrate.
Final amalgamation tailing : 1·0 dwt. per ton.

The final tailing from amalgamation was deslimed and the sand fraction was floated with Reagent 801 to effect a concentration of haematite. The products from flotation were a concentrate, middling and tailing.

In Table V are set out the percentage weights of the several products—their gradings, gold content, and partial analyses.

The magnetics were composed mainly of metallic iron. The concentration of haematite effected was not of a high order but sufficient to provide a comparison of the relationship between the Fe_2O_3 and gold contents of

TABLE V

Product		Slime	Flotation			
			Feed	Concentrate	Middling	Tailing
Weight—% of original		24·7	75·3	30·5	33·7	11·1
Weight—% of flotation feed		—	100·0	40·5	44·8	14·7
Gradings (Tyler)						
+ 48	%	—	0·1	—	—	0·7
— 48 + 100	%	0·1	2·7	0·9	1·6	11·0
— 100 + 200	%	1·9	29·7	16·6	28·7	68·9
— 200	%	98·0	67·5	82·5	69·7	19·4
Partial Analyses						
Au	dwt./ton	0·77	1·10	1·15	1·15	0·77
Fe_2O_3	%	10·3	30·1	43·5	25·2	5·7
Magnetics	%	Nil	4·0	5·4	5·2	4·0
Insoluble	%	85·0	63·3	47·8	67·1	89·1
Undetermined	%	4·7	2·6	3·3	2·5	1·2

everal products. The evidence is that the encasement of gold in atite is insignificant.

'ALL-SLIMING' AS AN ALTERNATIVE

ving described the new concentrating layout the author is still) that there are two schools of thought on the subject of concentra-)f gold from banket ores, the second of which poses the provocative · 'Why bother?'

ə first argues that by concentration a high proportion of the total is earlier in the bank, the cyanide plant is relieved of the duty of ution of all but the finest gold, treatment time in the cyanide plant be shortened and as a corollary less equipment capacity will be red, and residue washing will not be attended by the same risks of ved gold loss. These considerations assume greater importance in lant cyaniding a portion of the mill product as sand by leaching.

ə second school is confined to advocates of the one-pulp plant, in ı grinding is carried far enough to enable the whole of the ore to be ded by agitation and filtration or counter-current decantation. This l argues that with the mill-classifier circuit suitably arranged the ›ntering the cyanide plant will have been sufficiently reduced in size fective recovery in reasonable time by cyanidation alone—although o be remarked that this time is likely to be longer than if a concen- n step were included. Further, the result is a simplified flow-sheet which the processes and labour of working up a gold concentrate iminated. Also, in consequence, this type of plant does not present ımptation of or opportunity for gold theft that are present when the ing of a high-grade concentrate, often containing visible gold, is a

TABLE VI

CYANIDATION OF MANTRAIM ORE BY AGITATION

Ground to 65·6 per cent *minus* 200-mesh (Tyler)

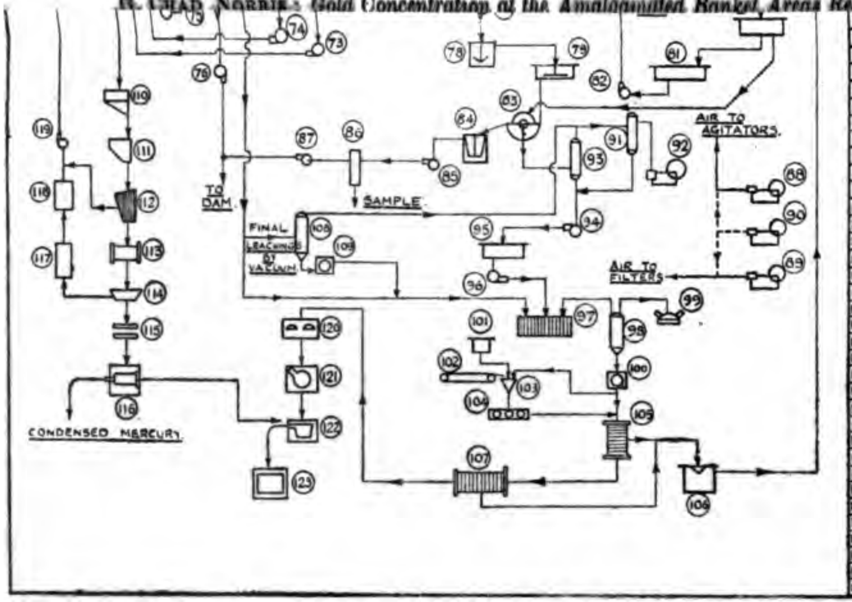
Cyanided	Directly			After Amalgamation		
yanidation—dwt./ton	4·87			1·08		
ion—hours	4	8	16	4	8	16
anided at NaCN%*...	·026	·026	·027	·030	·030	·030
ashed residue,dwt./ton	0·86	0·25	0·10	0·67	0·23	0·12
tal extraction, %......	82·3	94·9	97·9	86·2	95·3	97·5
ınided at NaCN %...	·047	·048	·047	·051	·051	·045
shed residue, dwt./ton	0·52	0·22	0·10	0·30	0·17	0·10
·al extraction, %	89·3	95·5	97·9	93·8	96·5	97·9

* At end of agitation period.

tion, Ltd., on whose recommendation the change in concentrati
practice was made and at whose laboratory research was conducted ;
the mechanical engineering department of New Consolidated Goldfiel
Ltd., for the preparation of the drawings, and to the general manager a
colleagues on the plant and in the Corporation's laboratory for co-ope
tion in establishing the new concentrating routine on a successful operati
basis.

*** *Extra copies of this paper may be obtained at a cost of 2s. 0d. ea at the office of the Institution, Salisbury House, Finsbury Circ London, E.C. 2.*

CROWTHER & GOODMAN (*Branch of The Argus Press Ltd.*), 2, Carmelite St., London, E.C.4.



89	2-COMPRESSORS FOR FILTER SERVICE.
90	1-COMPRESSOR STANDBY FOR 88 OR 89
91	2-MOISTURE TRAPS 34" DIA x 72"
92	3-DRY VACUUM PUMPS. (1-STANDBY.)
93	2-VACUUM RECEIVERS 34" DIA. x 72"
94	2-4" SUBMERGED EXTRACTION PUMPS.
95	UNCLARIFIED SOLUTION TANK 25' DIA x 10'.
96	2-5" HARLAND PUMPS. (1-STANDBY)
97	GOLD SOLUTION & CLARIFIER TANK 25' x 8'
98	MERRILL CROWE VAC. RECEIVER 72" x 12'-0"
99	2-VACUUM PUMPS (1-STANDBY.)
100	1-8" SUBMERGED EXTRACTION PUMP
101	3-REAGENT FEEDERS.
102	3-ZINC DUST BELT FEEDER.
103	3-MIXING CONES.
104	3-THREE THROW ZINC EMULSION, PUMPS.
105	3-36 CHAMBER 36" SQ. PRECIP. PRESSES.
106	LEA-RECORDER.
107	1-12 CHAMBER 25" SQ. CLEAN-UP. PRESS.
108	VACUUM RECEIVER 34" DIA. x 72".
109	1-4" SUBMERGED EXTRACTION PUMP.
110	3-WASHING BOXES FOR CORDUROY.
111	CONCENTRATES BIN.
112	WILFLEY TABLE.
113	2-AMALGAM BARRELS.
114	BATEA.
115	AMALGAM PRESS.
116	AMALGAM RETORT FURNACE.
117	COPPER PLATE.
118	GUARD STRAKE.
119	1-2" WILFLEY PUMP.
120	CALCINING FURNACE.
121	2-TILTING RETORT FURNACES.
122	BULLION FURNACE.
123	VAULT.

FIG. 1.—Original flow-sheet for treatment of 60,000 tons of ore per month (Amalgamated Banket Areas, Ltd.).

tion, Ltd., on whose recommendation the change in concentrating practice was made and at whose laboratory research was conducted; to the mechanical engineering department of New Consolidated Goldfields, Ltd., for the preparation of the drawings, and to the general manager and colleagues on the plant and in the Corporation's laboratory for co-operation in establishing the new concentrating routine on a successful operating basis.

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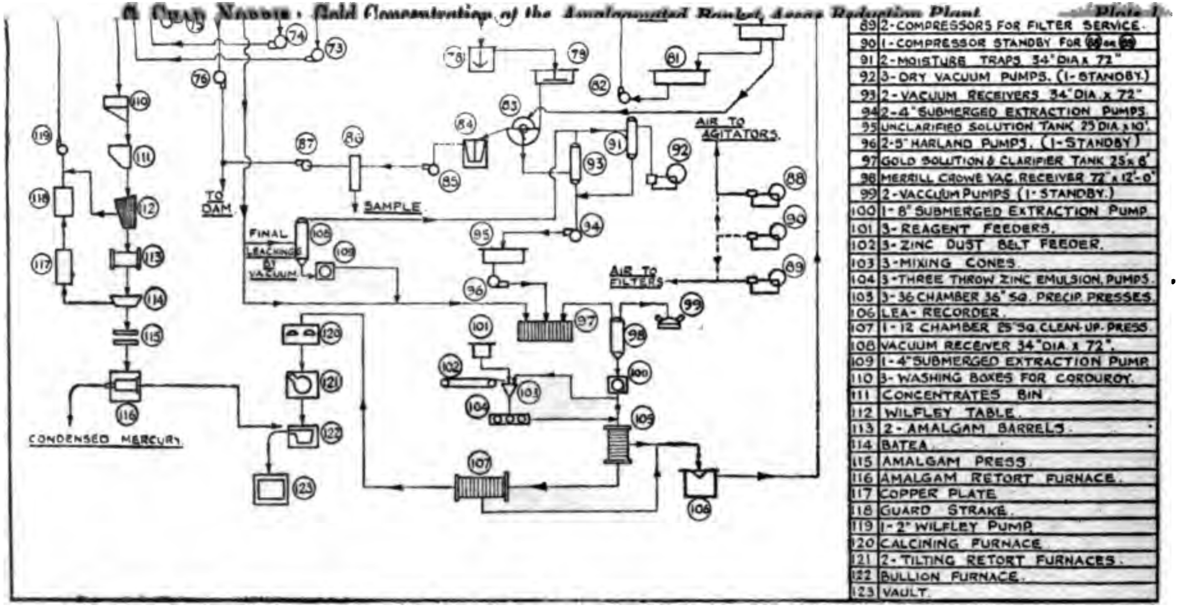

Fig. 1.—Original flow-sheet for treatment of 60,000 tons of ore per month (Amalgamated Banket Areas, Ltd.).

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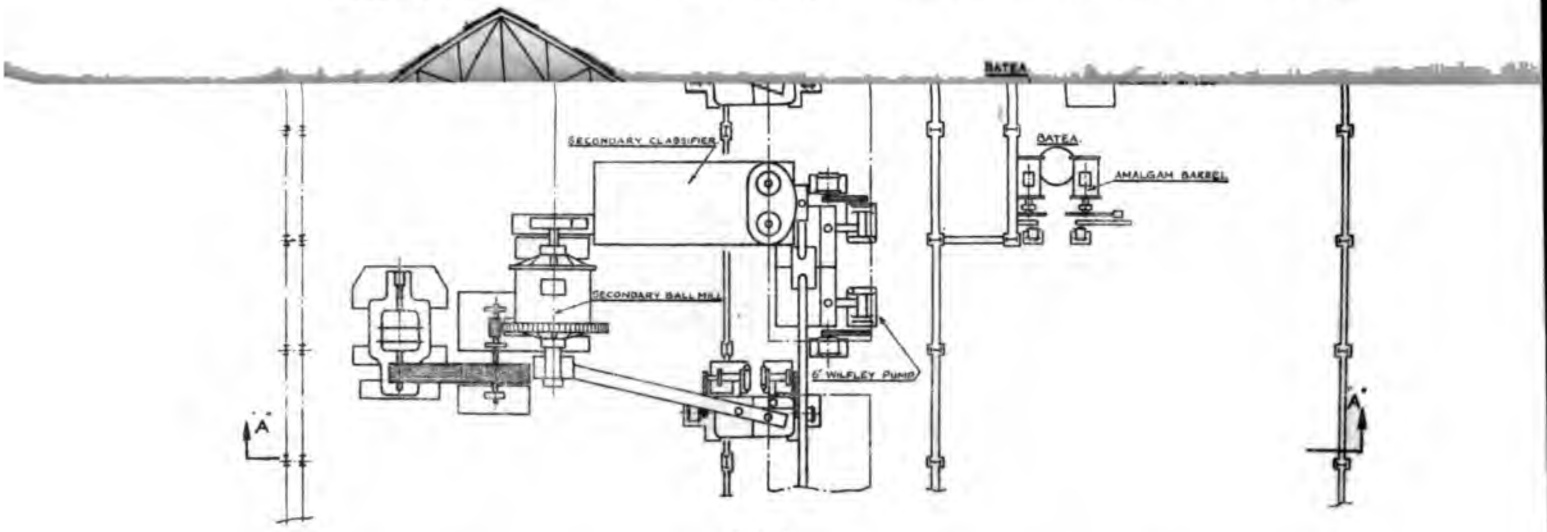
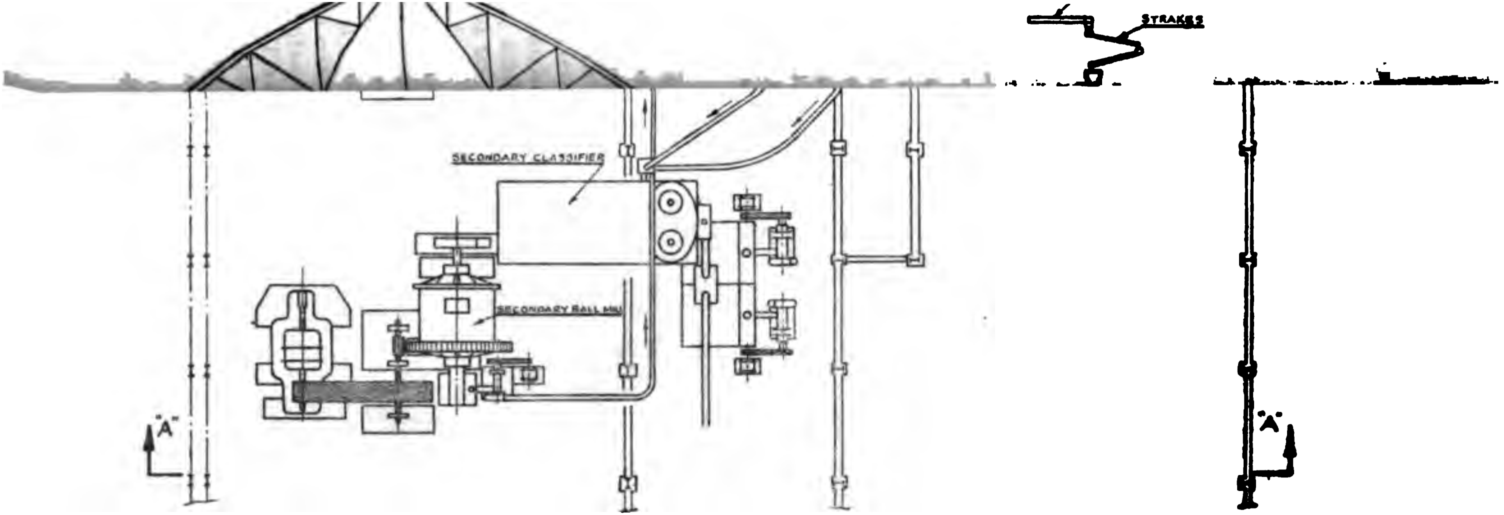


FIG. 2.—Original milling and strake installation—section and part plan (Amalgamated Banket Areas, Ltd.).



PLAN

FIG. 3.—New milling and concentrator installation—plan and section (Amalgamated Banket Areas, Ltd.).

Some Notes on a Mechanical Concentrator*

T. HADEN, B.E.M., ASSOCIATE MEMBER

A paper published on 10th November, 1949, by the Institution of Mining and Metallurgy, to be submitted for discussion at the November General Meeting

INTRODUCTION

For a very long time the amalgamating plate, as used to collect gold from an ore pulp, has been a speedy, reasonably efficient, and reliable device. Working with comparatively small tonnages it has been used in most countries and on most ores. Yet its characteristics are such that, with every increase in the monetary value of gold or in the tonnage milled, the trusty old amalgamated plate becomes more and more costly to operate. The plate holds up large, and frequently astonishingly large, weights of gold. Theft risk is always great and, no matter what precautions are taken, free mercury and amalgam will contrive to creep all over the circuit, to add to that risk and complicate other processes.

As a consequence of these defects the number and variety of gravity concentrating machines and methods which have been developed as substitutes illustrate both the ingenuity of the millman and his dissatisfaction with most of these devices. The purpose of the present notes is to describe yet another machine of this nature.

MECHANICAL CONCENTRATOR

The machine in question may perhaps be called a mechanized strake, but in its design and operation there is much of the vanner. As may be seen from Fig. 1 the side elevation resembles a vanner working at a steep slope. In the precise form of the concentrating surface itself probably lies the secret of the efficiency of the machine. This surface is a riffled rubber blanket, widely used in South Africa as a concentrating medium for gold, and in the present machine the separate blankets have been cemented to a thin wide rubber conveyor belt. The belt is endless, with a vulcanized joint. This belt, much as in vanner practice, moves slowly upstream with its load of concentrate, and at the top of the table the concentrate is subjected to a cleaning action under a light flow of water. The cleaned concentrate is carried round the upper pulley and is detached by gravity under the surface of water in a collecting tank. From this tank the concentrate is recovered in any convenient way—as by sluicing it intermittently through a valve for transport by hand, or by a stream of water. The washed belt returns under the machine to the lower pulley to repeat the cycle.

The mechanism of the machine is relatively simple. There is but one movement involved—the slow travel of the belt upstream. The frame of the machine consists of two main members, spaced by cross members

***Paper received on 21st June, 1949.**

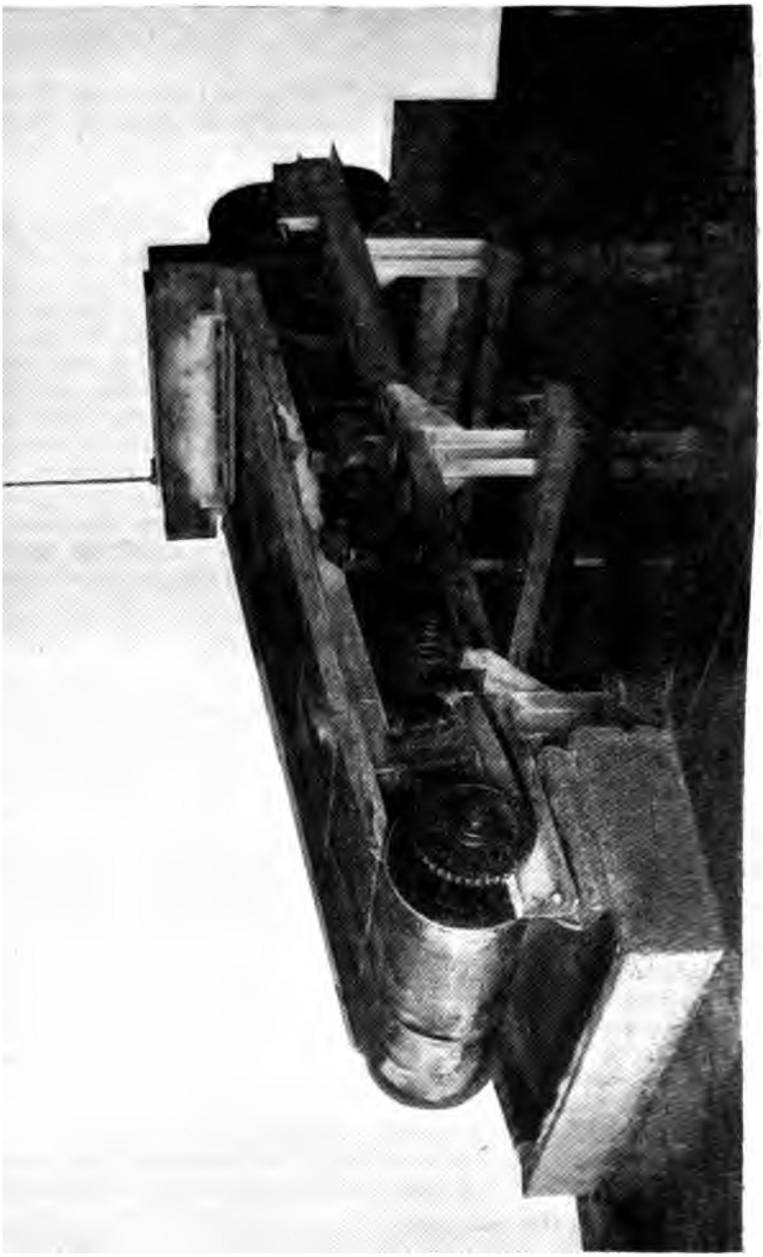


Fig. 1

welded in place, and the whole supported on legs to resemble an inclined table. At the top edge is the upper pulley. Centred about 2 ft. behind, and about 4 in. lower, is the snubbing pulley ; the feed is applied above this pulley. At the lower, tailing, end is the drive pulley. All these pulleys are 15 in. in diameter, 49 in. between flanges, and very lightly crowned ; they are carried on stiff shafts running in sealed ball bearings. Drive is applied at the lower end, on the discharge pulley. This end has been chosen for the drive in order to bring the mechanism away from the inevitable splashing of pulp and water at the feed end. The drive is by chain of 1-in. pitch, itself driven via a train of reduction gearing from a 1-h.p. motor. A ratchet wheel in this train offers the possibility of altering the speed of the belt if desired. On this design the belt speed is approximately 9 ft. per hour minimum, when working one tooth of the ratchet wheel, and this may be increased in steps of one tooth up to ten teeth—approximately 90 ft. per hour—if desired. The higher speeds are, however, merely possibilities inherent in the mechanism, and the author sees no practical use for them. The lowest speed, 9 ft. per hour, is equal to a complete and automatic change of the whole concentrating surface every 50 minutes, which is much better blanket changing than one could expect from any Bantu ' blanket boy ' on a cold night shift.

Both the upper and the lower pulleys on the machine are capable of adjustment. The lower pulley tensions the chain drive and the upper pulley tensions the belt. Angular adjustment of the lower pulley has the greatest influence on the ' training ' of the belt. The pulleys are lightly crowned to assist this training and are provided with flanges merely to keep pulp and water in the proper channels and prevent creep of pulp into the bearings. Obviously the pulleys must not be too highly crowned, as this would interfere with the flat concentrating surface of the belt.

Deck

The concentrating deck of the machine is approximately 8 ft. long by 3 ft. 9 in. wide, a total of approximately 30 sq. ft. between feed points and the curve of the discharge pulley. Over this area the deck is held to a constant slope in the direction of flow and as nearly as possible horizontal laterally. The optimum degree of slope on the deck is probably determined by pulp characteristics and with the pulp so far observed a slope of about 12 per cent seems to be suitable. If flatter than this, sand banks may build and, by covering the surface of the belt and blinding the riffles, reduce the area of deck in actual use. It is obvious that too steep a slope will lead to high pulp velocity, turbulence, and scour. When the deck is working at approximately optimum load and slope the surface of the pulp presents a pattern of horizontal lines in exact agreement with the riffles of the deck. With too great a load or too steep a slope the pattern is one of ' diamond ' waves on the surface of the pulp and a condition of scour in the riffles. By looking at one point on the deck and ignoring the movement of the actual pulp flow the patterns may be seen very easily.

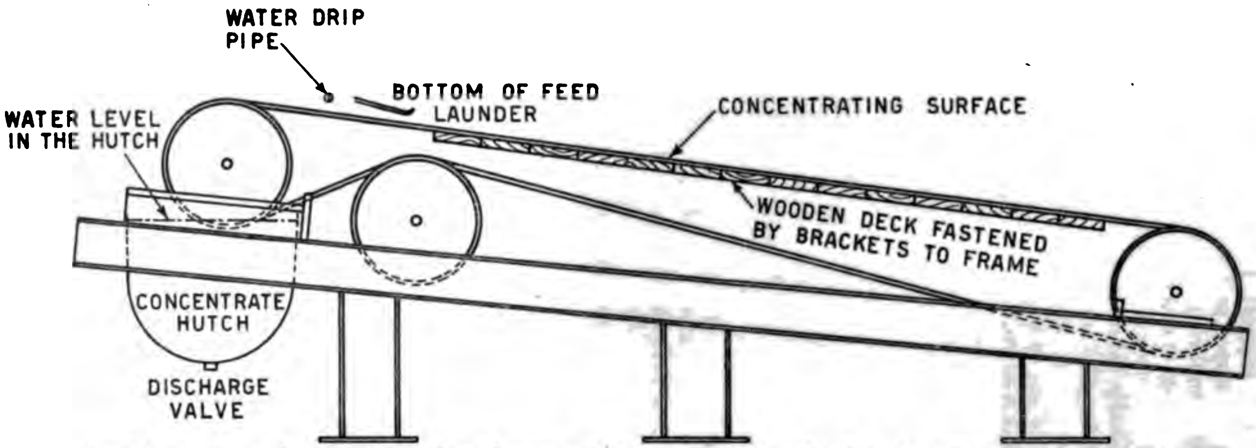


Fig. 2.—Side elevation of the machine, simplified by omission of all obvious detail work. (Scale approx. 1 in. = 2 ft.)

Fig. 3.—Approximate cross-section of the rubber concentrating surface.

Belt

The belt is held to this angle by running it over a wooden deck firmly fixed to the frame of the machine. This deck takes the place of the idler rollers which could be used for the same purpose at the risk of a corrugated surface and constant bearing trouble on the rollers themselves. The additional wear on the belt is thought to be negligible for there is very little weight load on the belt, and even under full load and the drag of the wooden deck the mechanism may be turned by one hand on the ratchet wheel, at much greater than normal working speed. This ability is useful when scrubbing the belt.

The belt itself is built on an endless rubber conveyor belt 48 in. wide, of 3-ply nylon duck fabric, with $\frac{1}{16}$-in. top and bottom covers. On the working side of this belt three flanges and two areas of rubber concentrating blanket have been cemented. The flanges should be made of solid soft rubber with no reinforcing of cord or fabric. They must stretch and contract over the pulleys, and cords would be troublesome. They are intended to serve two distinct purposes : (*a*) to act as the sides of the pulp channel on the deck, and (*b*) to hold important areas of the concentrating surface away from contact with the snubbing pulley on the return under the machine. This last is thought to be important, because the efficiency of the concentrating blanket depends on the profile of the riffles, and contact with the snubbing roller surface would ultimately deform these surfaces and edges.

Concentrating Surface

The concentrating surface is made of a rubber blanket which has won a wide popularity in South Africa. Its profile is approximately as shown in Fig. 3. It is purchased in units of approximately 20 in. by 40 in. These units were cut and joined to suit the spaces between the flanges and cemented in place. It has been found necessary to scrub out the riffles with dilute hydrochloric acid about once a week, because lime concretions form in scales on the surface of the rubber and interfere with the clearing of the concentrate under water in the hutch. When scrubbing, the feed pulp is stopped and the dilute acid applied at the feed point. The belt is scrubbed with a stiff brush parallel with the riffles, while a labourer turns the ratchet wheel by hand to drag the whole surface of the belt through the acid-and-scrubbing-brush area in about ten minutes. The machine is then hosed down to remove the acid from the steelwork of the hutch and snubbing pulley.

Feed to Surface

The feed method needs careful attention. There is room for more than one opinion on the subject of feeding such a machine and it is probable that differences of ore and mineralization will greatly affect the methods used. The author offers his opinions and outlines his reasons as those suitable for at least one ore.

The feed should be applied to the belt softly and with a minimum of splash, but at specific points. The feed launder should be quite free from pockets, baffles, traps, perforated plates and the like and should be capable of rapid and complete cleaning ; and the author believes that an ideal feed launder would be merely an open chute, bifurcate at the delivery, and terminating in spatulate areas about half an inch above the surface of the belt. The whole would be smoothly lined with rubber and set at an angle steep enough to keep itself clean. Whatever form of feed box is used the feed is applied at two areas on the width of the belt, at areas A–A of the sketch, Fig. 4. Within a few inches of

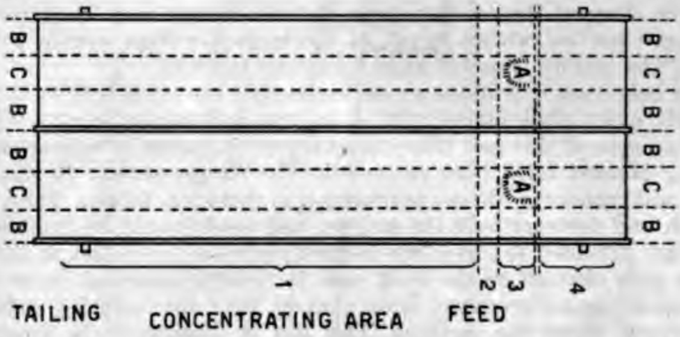


Fig. 4.—A diagram of the plan of the deck, indicating the zones and areas referred to in the text.

this area the feed pulp has spread over the full width of the belt and covers most of the areas B–1, C–1.

Concentration of gold takes place behind the riffles and this concentrate is dragged slowly upstream by the travel of the belt. Particles of concentrate in the zone B will be dragged past the turbulence of the feed areas without disturbance. Moreover, as they approach B–2 they will be overlaid and protected by sand banks in which are embedded coarser particles of concentrates, often with the coarsest fragments and coarse gold, right on the top. In effect this bank resembles 'reverse classification'.

Any particle of concentrate settling in the zone C–1 will be dragged back through the scour of the C–2 area into the turbulence of the A area, where the new feed falls to the belt. Here, depending upon the individual properties of the particle, it will sooner or later be scoured out of the riffle and washed downstream, and direct observation has shown that the only type of concentrate which can penetrate that curtain of turbulence is the rare case of the little gold nugget large enough to be picked up with the fingers. This displaced concentrate, however, has, in effect, joined the new feed and, spreading fanwise over the areas B–1 and C–1, again takes its chance of entering the B zones. Thus there is a small, but not serious, 'middlings' load constantly on the belt with one chance in two of joining the concentrate at each settlement.

The rough concentrate is dragged up past the feed areas in the form of sand banks almost free from slime, and some drainage and washing takes place in the areas B–3.

A water spray is set between zones 3 and 4, to deliver trickles of clean water. Very little is needed—merely a rapid succession of drops which cut into the sand banks and disturb them a little. This is sufficient to remove much of the valueless quartz sand at low velocity and leave on the belt a fairly clean sulphide concentrate, which, in this zone 4, can be distributed over the whole B–4—C–4 area in parallel ridges. No attempt is made to take a very clean concentrate. On the contrary it is thought better to take at least one ridge each side with a high quartz content. It is in this quartz-rich strip that the very fine free gold is trapped and entrained. Frequently the cleaner sulphide ridges carry coarser grains of gold right on the crest, a $\frac{1}{4}$ in. above the rubber deck. Close observation of the work of this belt in this zone leads to an obvious conclusion that the phenomena involved are simply those of reverse classification with the bed dilated by the slow movements of the belt under the light flow of water. This seems to explain the ability of the machine to trap and recover fine free gold from an unclassified, slimy, battery pulp when worked as outlined above.

Another idea for feeding the machine may be visualized, and has been tried, with the feed pulp laid down in a shallow even stream over the full width of the belt. It is clearly a practical impossibility to lay down a feed pulp in this way with literally *no* turbulence. Any turbulence at all leads to a condition resembling that of the C–2 area over the full width of the belt, with a light scour constantly cleaning out the finer concentrates from the riffles and permitting only the coarser concentrate to pass. The fine concentrate then builds up into a heavy middlings load on the belt and purges itself in irregular surgings either up or down the belt. This has been observed under these conditions. Moreover, as such a condition of shallow even feed can only be obtained by the use of baffles in the feed launder (and is very difficult to hold even then) gold concentrations are made in holes and corners and are very difficult to clean out.

The concentrate is carried over the upper pulley and, being wet, adheres to the belt until the surface makes contact with water in the hutch. This water, by simple contact, detaches the concentrate, which falls and collects in the bottom of the hutch to await routine removal. Separation is clean unless the belt needs scrubbing (a limed-up or greasy belt can carry gold through the water). The water level in the hutch is raised by the constant inflow of concentrate and is kept constant by overflowing via a goose-neck pipe—this to avoid loss of floating concentrate.

The daily recovery of concentrate is treated normally in the amalgam barrel for recovery of a clean gold amalgam.

TEST RESULTS

Some specimen results of the work of the strake might be given. In the following example the strake feed was a gold ore—a stamp battery pulp through ' 600-mesh '* screens at approximately 10 per cent solids. About 15 per cent of the ore was ' primary slimes '. A specimen screen analysis showed : *plus* 60, 21 per cent ; *minus* 60 *plus* 90, 27 per cent ;

*600 apertures per square inch approximates to a 25-mesh standard screen.

minus 90 *plus* 200, 26 per cent ; *minus* 200, 26 per cent. The average of nine assays of the feed was 8·63 dwt. and the average of nine assays of the tail, 2·15 dwt., the average extraction therefore being 75 per cent of the total gold in the feed.

Some of the gold in the strake tailing was chatted with the coarse sand, while in this ore some of the gold is occluded in pyrite. There are no figures to show just how much gold was in these two classes, but from study of old records it is possible to reach a conclusion that these two together may carry more than 25 per cent of the total gold. Therefore the strake extracted almost the whole of the free gold in the pulp, with the concentrate sand. After making due allowance for the tailing sand from the amalgam barrel, the actual gold recovery as amalgam from the strake was slightly higher than that from the old amalgamated plate system.

The weight of the concentrate varied between 1 per cent and 1·5 per cent of the feed ore weight. Probably less could have been taken. The weight of the feed was approximately 24 tons of ore per 24 hours run. Later work indicated that at that degree of dilution a probable maximum would have been about 30 tons of ore, or about one ton of ore per square foot of concentrating area per day, at about 10 per cent solids. The feed, being a battery pulp, carried all its original slime, water and tramp chips ; had it been practical to classify this feed and to adjust a slime-free pulp to optimum solid-water ratio, it is probable that much greater tonnage could be handled over the deck. The slime fraction would then have been a separate study. The additional equipment needed for this classification and elevation, however, would add greatly to first cost and labour charges. Direct concentration of the whole battery pulp at a loading of one ton per square foot per day seems to be a simpler and cheaper method for the smaller tonnages of gold ores. Ball-mill circulating pulp, normally deslimed in the classifier and capable of adjustment of the solid-water ratio, is also a separate study.

CONCLUSION

It cannot be claimed that there is very much that is original in this machine ; it is so clearly descended from both vanner and strake practice. The one described here was designed in the light of the author's own experience and prejudices and Messrs. F. Issels and Son, of Bulawayo, who constructed the steelwork, modified the original sketches to conform to engineering practice and to utilize available materials. Several other forms of this general type of machine can be visualized and the author would expect equally good work from them all. Such a machine could be of great help in a low-tonnage gold mill, particularly where much of the gold is free or coarse. It is not costly, draws only one horsepower to operate it and has a high capacity even when working a dilute stamp battery pulp. It does not hold up gold and can be comparatively easily guarded against theft. Once the belt has been trained almost the only labour should be that necessary to empty the hutch and transport concentrate.

*** *Extra copies of this paper may be obtained at a cost of* 1*s.* 0*d. each, at the office of the Institution, Salisbury House, Finsbury Circus, London, E.C.2.*

Recovery of Sulphur from Smelter Gases by the Orkla Process at Rio Tinto

H. R. POTTS, MEMBER, and E. G. LAWFORD, A.R.S.M., MEMBER

Further contributed remarks on paper published in Bulletin 509, *April* 1949

MR. L. U. SALKIELD : I have read the discussions on this paper with interest and would like to make a few comments, especially about pyrite and the rate of removal of the labile sulphur. Pyrite is far from a constant chemical compound, being seldom found in a pure state. Dr. David Williams, in an article in Thorpe's *Dictionary of Applied Chemistry*, says: 'Pyrite is notably non-stoichiometric, its composition ranging from $FeS_{1 \cdot 94}$ to $FeS_{2 \cdot 04}$, even at ordinary temperatures, with a corresponding variation in physical properties'. The work done by Allen and Lombard[1] was on a very pure pyrite, said to come from Colorado, and containing 46·71 per cent Fe and 53·29 per cent S; it is questionable whether there were any traces of impurities present.

Many investigators have determined the dissociation pressure of pyrite, but they have all worked in a temperature range between 575 and 689°C., just on, or slightly below, the transformation or transition point, where pyrite alters to pyrrhotite—Fe_nS_{n+1}. Thus, the dissociation pressure curves obtained are for the reaction—

$$2nFeS_2 + \text{heat} = 2Fe_nS_{n+1} + n-1S_2,$$

and not the reaction—

$$2FeS_2 + \text{heat} = 2FeS + S_2,$$

which has been commonly assumed and on which the thermodynamic properties of pyrite have been calculated[2]. Allen and Lombard[1] found 'A strong absorption of heat in pyrite between 665 and 685°C.', attributing this to the fact that the vapour pressure of the sulphur present had reached atmospheric pressure. Apparently they had not considered that this absorption might have been due to the formation of pyrrhotite, when the lattice changes from the cubic to the hexagonal structure.

Work carried out on a more impure pyrite (the analysis is given later) showed that this strong absorption of heat takes place between 694 and 696°C. This was the only transition point observed between the temperatures 240 and 946°C. Perhaps the best data on the dissociation pressure of pyrite are those given by D'Or[3] and shown in Fig. 7.

In the Orkla furnaces, where the sulphur content of the gases is about 200 g. per cu.m., the vapour pressure is just over 18 mm.Hg. (assuming that, of the total sulphur molecules present, 15 per cent are S_2, 61 per cent are S_6, and 24 per cent are S_8). No dissociation would be expected below the temperature of 588–590°C. This is above the temperature of vaporization of the high-arsenic sulphur mentioned by Mr. Rich. In other words, contrary to Mr. Rich's contention, this high-arsenic sulphur, which is

(1) See references at end of contribution, p 30.

deposited on the cold charge entering the furnace, is removed from its surface before labile sulphur would be liberated from the pyrite.

Mr. Rich, in his written contribution, is perfectly correct when he says that the kinetics of the calcination of pyrite are extremely important. It is to be regretted that he did not give the final weights of the calcined mineral, nor the original and final complete analyses. Without this information the usefulness of those data is much impaired, because it is impossible to calculate the amount of labile sulphur released.

The theory for the dissociation of a solid is that it starts at certain centres, usually on the surface of cracks and lattice imperfections, the

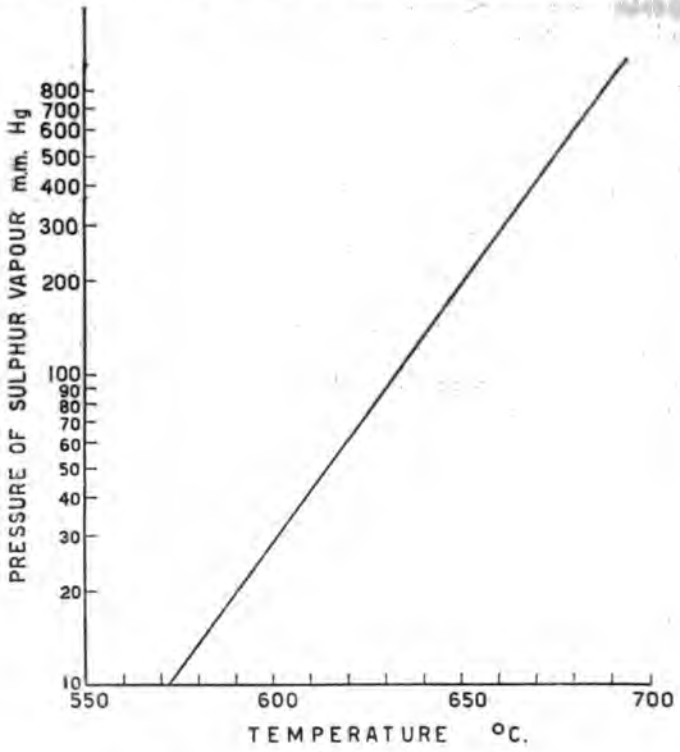


Fig. 7.

reaction then spreading along the surface and into the interior. When pyrite is heated in an inert atmosphere the surface becomes covered with cracks after a very short interval of time, the whole mass increasing in volume. This growth is strongly reminiscent of the opening of a rose, from bud to full bloom, because the pyrite forms what are to all intents and purposes hundreds of petals. It has been noticed by many investigators that the rate of dissociation of a solid increases with decrease of particle size up to a point, but below a certain size (about 150–200 mesh) the rate decreases. This statement only applies when the particles are lying quiescent and not suspended in a gas. In the case of pyrite, although the rate of dissociation must be somewhat retarded by the size of the mineral entering the blast furnace when compared with, say, a 1-mm. particle, nevertheless, owing to the cracks forming, the retardation is not so great as it would be in the case of, say, limestone.

In tests carried out at Rio Tinto the rate of dissociation of pyrite was determined as a function of temperature and time. The results are shown in Fig. 8 and are interesting as showing that the rate of dissociation increases rapidly to between 75 and 82 per cent removal of the labile sulphur, depending on time of calcination; after this the rate gradually decreases until 100 per cent of the labile sulphur has been removed.

As pyrite is by no means homogeneous, the mineral was crushed to *minus* 60-mesh to get as standard a sample as possible for each test. The pyrite was of the following percentage composition: Cu, 0·35; S, 51·00; Fe, 44·64; As, 0·85; Zn, 0·20; Pb, 0·11; Bi, 0·005; moisture, 0·23; rest

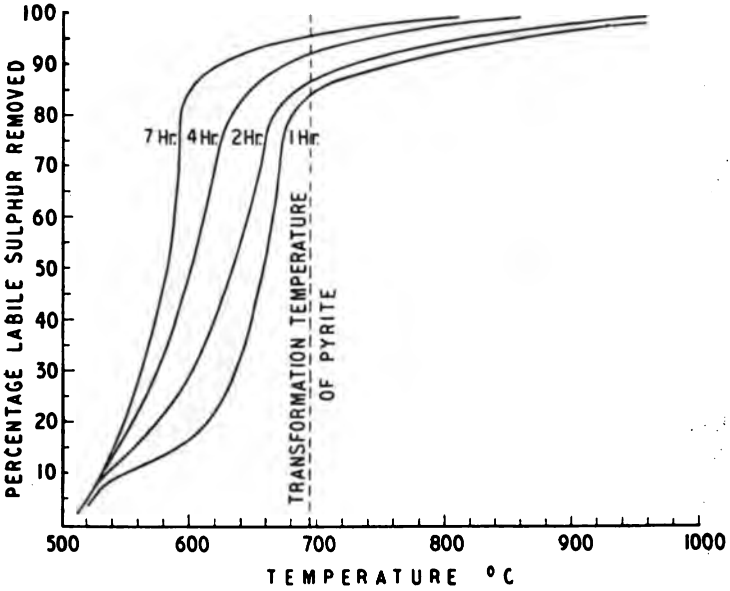


Fig. 8.

(SiO_2, CaO, etc.), 2·615. The labile sulphur was 24·53 per cent. In the tests samples each of 5 g. were put in a porcelain boat and placed in an electrically-heated $\frac{3}{4}$-in. silica tube, brought up to the required temperature before each test. An inert gas (carbon dioxide or nitrogen) was passed through the tube at the rate of 50 c.c. per min. (The nitrogen was freed from oxygen by being passed over heated charcoal.) On heating the mineral to a temperature of 490°C. for 7 hours, only 0·04 per cent of the labile sulphur was lost. On the other hand, when the mineral was heated to 810° for 7 hours, to 860° for 4 hours, and to 960°C. for 2 hours, all the labile sulphur was removed.

It will be noticed from the curves that at the transformation temperature there is no sudden change in the rate of elimination of sulphur. From these curves it is easy to appreciate why there is such disparity in the formula for pyrrhotite. As mentioned by Mr. Stanley Robson in the discussion, it appears to vary from Fe_3S_4 to $Fe_{12}S_{13}$, or in fact even higher. It seems to depend on the time of heating the mineral at the temperature of the transformation point.

Although the fact is not recognized in the published literature the dissociation pressure curve for pyrrhotite need not necessarily be the continuation of the curve for pyrite. It would be interesting to know if the dissociation pressure for pyrrhotite has been determined.

Clark and Spittle in their work on 'The manufacture of hydrogen sulphide'(4) give some interesting information on the rate of dissociation of the labile sulphur from pyrite. As the gas over the pyrite was hydrogen, combining readily with sulphur, the sulphur vapour pressure must have been very low and thus the rate of dissociation was extremely rapid. They based their calculations on the time required to remove 85 per cent of the labile sulphur. The analysis of the pyrite used contained 51·54 per cent S, of which 24·35 per cent was labile sulphur, and was very similar to the analysis already given, although the arsenic content was less. The mineral was screened between *minus* ½ and *plus* ¼ in. Some of the results given are shown in Table XVII (Cf. Fig. 8). For a full appreciation

TABLE XVII

Temperature °C	Time for removal of 85% of the labile sulphur
619	1 hr. 32 min.
620	58 ,,
623	48 ,,
694	36 ,,
697	39 ,,

of the data the original paper should be consulted; the space velocities of the gas passing over the pyrite are also given.

Dr. Levy states that he found that pyrite heated above 850°C. tended to fuse and slag with the silica tube. This is surprising, because no sulphide combines with silica. Is the explanation a trace of oxygen in the inert gas used? The melting point of ferrous sulphide is given as 1180°C., and certainly up to 960°C. the writer has under similar conditions never seen any sign of fusion. In fact, after nearly all the labile sulphur has been removed from lump pyrite it can be powdered by the slightest pressure. As the ferrous sulphide is so fragile it is always a mystery to the writer how it behaves in the lower zones of the blast furnace—i.e. before it becomes molten.

It has been found by Kohlmeyer(5) that at the boiling point many sulphides begin to lose sulphur. No doubt this is the reason for the loss of sulphur and the formation of carbon disulphide reported by Dr. Levy, but the writer is under the impression that the boiling point of ferrous sulphide is much higher than any temperature mentioned by Dr. Levy.

However, when a molten sulphide is mixed with carbon the temperature at which decomposition commences is said to be lowered, so perhaps the carbon crucible helped the decomposition of the ferrous sulphide.

It is doubtful if we are correct in assuming that in the blast furnace the final stage in the release of the labile sulphur before oxidation is Fe_7S_8, but it is a convenient end point, as we cannot expect much above 91 per cent recovery of the labile sulphur.

In the discussion, the suggestion made by Mr. Stanley Robson and Mr. Rich that the charge might be preheated with benefit before it enters the furnace certainly has some merit, if only to remove moisture. However, as the gases must leave the top of the furnaces at a temperature of between 400 and 450°C., then, if the charge enters at, say, 200°C., the rate of heat transfer between the hot gases and hot charge would be much less than when using a cold charge. This would mean, as Mr. Rich admits, a much higher furnace column and it is doubtful whether the extra cost of higher pressure blast would compensate for any further sulphur recovery. In fact, the writer believes that no better sulphur recovery would be obtained.

Mr. Stanley Robson asked whether a thermodynamic survey had been made for the probability of the chemical reactions in the furnace. This has been done and the work confirmed later by that of C. W. Siller in his paper 'Carbon disulfide from sulfur dioxide and anthracite'.[6]

All the equations given by the authors have standard free energy changes considerably less than 0. However, it is interesting to consider the reaction—

$$CS_2 + SO_2 = CO_2 + \tfrac{3}{2}S_2.$$

At 1000°F Δ F° is —23,700 (where Δ F° is the standard free energy change), and yet CS_2 is present in the exit gases. The reaction can only take place if the carbon disulphide and sulphur dioxide molecules are brought close together by a catalyst. (The concentrations of these two gases in the furnace gases are not high enough for any direct reaction.) Thus, although the thermodynamic calculations definitely show that the reaction can take place, it gives no guidance as to how it should be performed. Likewise, the equation—

$$SO_2 + 2H_2S = 2H_2O + \tfrac{3}{2}S_2$$

should take place at any temperature above 300°C., but on catalysing the gases the most effective results are obtained at temperatures of less than 200°C. This is because there is a number of different sulphur molecules which alter with temperature. If the reaction is

$$SO_2 + 2H_2S = 2H_2O + \tfrac{3}{2}S_2 \quad \text{(1)}$$

Δ F° at 200°C. is about + 1,060, but if the equation is

$$SO_2 + 2H_2S = 2H_2O = \tfrac{3}{8}S_8 \quad \text{(2)}$$

then Δ F° at 200°C. is about —15,000. As sulphur gas at 200°C. is chiefly S_8 the above reaction (2) will take place.

Mr. Stanley Robson suggests the recycling of the sulphur dioxide from the discharge gases after the cold Cottrells. If the sulphur dioxide could be removed from the exit gases and concentrated, by one of the many known processes, then recirculating the sulphur dioxide, by adding it to the blast, would certainly help sulphur recovery, provided the molten ferrous sulphide was at a temperature of over 1500°C., below which the velocity of the reaction $2FeS + SO_2 = 2FeO + \frac{3}{2}S_2$ is too slow. However, perhaps he was considering the beneficial effect of a higher concentration of sulphur dioxide in the gases at the reducing zone. Again, a higher concentration would most certainly give a better recovery. This is shown in Robert Lepsoe's excellent paper on the 'chemistry of sulfur dioxide reduction[7].'

Finally, Dr. Levy mentioned the rapid elimination of arsenic trisulphide when pyrite is calcined; this is also mentioned by Clark and Spittle[4]. They make the observation that if pyrite is first used for the production of H_2S, the residue remaining can easily be roasted and the sulphur dioxide gas is practically free from arsenic.

Not only is arsenic eliminated, but also bismuth, as well as lead and some zinc, depending on the temperature and time of calcination. Thus, the removal of arsenic is as shown in Table XVIII.

TABLE XVIII

Temperature °C.	Percentage of As left in calcines after heating 1 hour	2 hours	7 hours
470	0·35	0·20	0·11
550	0·15	0·10	0·07
680	0·02	0·01	nil
960	nil	nil	

The bismuth removed is as shown in Table XIX.

TABLE XIX

Temperature °C.	Percentage Bi left in calcines after 2 hours' calcination
470	0·005
550	0·004
680	0·002
725	0·001
960	Trace

On heating pyrite to 960°C. in an inert atmosphere, all the lead was removed and about 25 per cent of the zinc. The original analysis of the mineral was given earlier.

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(5) Kohlmeyer. *Metall. u. Erz*, **29**, (6), March 1932.
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Mr. N. E. Lenander: Before going closer into the subject I would like to express my satisfaction that the process has found interpreters so eminently capable, as well as my admiration for the work carried out at Rio Tinto. The subject matter has been arranged in a remarkably clear way so that even those not already earlier familiar with the process can thoroughly study its technology and understand without difficulty.

I have examined the figures given: (1) for the *gas quantity per ton of yrite*, founded on a *C-balance* (Table I), (2) for the *sulphur balance*, founded n the supposed gas quantity of 1,250 cu. m. per ton of pyrite, and (3) or the equilibria of the various sulphur compounds (Table III). I have no bjections either to the way of making the calculations or to the results btained.

I have read, and agree with, what the authors say about the enormous lifficulties involved in obtaining accurate samples and analyses of the urnace gases. My own experience has taught me how difficult it is to have sampling and analysis that gives the correct distribution of sulphur in he gases as between the various compounds of SO_2, COS, CS_2, etc. The esults specified in the paper, founded, as is confessed, on broad assumpions, must naturally not be regarded as absolutely fixed. The authors nention, for instance, that calculations have been made with a coke the nalysis of which is as follows (p. 24): volatile compounds, 1 per cent; shes, max., 8 per cent, and moisture, max., 3 per cent. It is stated that he specification aimed at for the coke for the furnace is that cited above. When examining the figures, it will be found that it is this analysis which has been employed; but then follows the statement:

'Actually great difficulty has been experienced in obtaining Spanish coke which even approaches this ideal; most of the coke, received from the North of Spain, contains not less than 15 per cent ash, and on occasions 26 per cent, while he moisture is rarely less than 10 per cent and sometimes reaches 15 per cent.'

Nothing is said about the volatile compounds. When comparing these conditions with those at San Domingos (and with those prevailing all over the world during recent years) it seems as if it would have been advisable to calculate with coke of a quality inferior to the one assumed by the authors.

For the sake of completeness it might have been suitable to give a calculation showing how the *moisture of the air* affects the equilibria and nfluences the balances. Also it might have been profitable when treating the section on 'the practice of the process' to have given greater attention to discussing the classification of the charge—i.e. the particle size of the coke, ore, quartz, and limestone. In addition I think that the most effective way of studying the various effects, when changing the composiion of the charge, would be to find out a really effective method and apparatus for measuring the quantity of air introduced at various intervals nto the furnace. It would then be much easier to follow the furnace operation, and one would have to rely less on the analyses which are obtained at a much later date. It would also be easier to follow how the concentration of the matte should be regulated in a proper way.

My remarks should be considered as constructive only. I am pleased once more to congratulate the authors on the excellent work carried out by the Orkla process, one to which I have to some extent contributed.

[otes on Mining Education and Postgraduate 'raining

A. S. RITSON, O.B.E., D.S.O., M.C., MEMBER

ırther contributed remarks on paper published in Bulletin 511, *June* 1949

MR. A. M. BENSUSAN: Two points are stressed in the discussion on :ofessor Ritson's paper: the present-day student's lack of mining ıckground; and the fact that the necessary experience is gotten, not ven.

It is understandable that an entrant into the profession who is ficient in such background may well be at a loss to know what experi-.ce he should set out to get, and with what real object he is getting it. e is thus more easily side-tracked into some *cul-de-sac*. Manual labour the start of his career is not a substitute for this deficiency; it only ifles any existing 'thirst to know and understand'. Rather should the aduate be shown the final aim of his endeavours at the outset, by close ntact with management; he cannot then fail to appreciate his own lack ability in that direction. He can then be sent to the next lower steps in e organization and learn to appreciate the difficulties there. Following e principle through subordinate positions the learner must before long anxious to experience the fundamental problems themselves— ck-breaking, maintenance, etc.—if only for the sake of his personal ide; he will know his aim.

If such a scheme (to serve as proxy for a mining engineer father) could put into practice, the present difficulty would seem to be largely 'ercome.

MR. D. V. STORRS: It has been very interesting to me, as a student, to ad the stimulating discussion which followed the introduction of :ofessor Ritson's notes on mining education and postgraduate training. s a member of one of the four interested parties, one likely to be affected any action taken from the final conclusions of this discussion, I thought at a few comments would be in order.

I would like first to explain that I am an ex-service student and, erefore, older by six or seven years than the average. At the Royal hool of Mines, however, there is such a strong bond between 'ex-service' ıd 'boy' students, that the gulf may appear to be wider than it actually

Contrary to some previous statements, this has been an extremely ntroversial discussion, the main problem having been that of integrat-g the theoretical and practical training of the embryo engineer. Broadly eaking, there are two schools of thought: those in favour of a sensibly gid system of postgraduate training given by the industry (designed to 'ovide the necessary experience) and those who would prefer to increase e period of practical training undergone while at the university and chnical school and then to give the student a definite job of work on graduation. Before returning to this difference of opinion I think it is important briefly to review the links in the education chain which lead up to it.

The first link is, of course, the school. School training is, in itself, a highly controversial subject. I will content myself by saying that its prime task is to train minds to think clearly and accurately, and, at the same time, to provide necessary incentives for thinking. Thought is for the young (and I hope for the young only) a somewhat painful business—at best a poor substitute for action, from which, instead of being complementary, it is often divorced. The training for clear thinking usually involves subjects in which accuracy is at a premium—such as classics and mathematics. As Mr. J. A'C. Bergne has testified, this necessary process is often extremely repugnant to the scholar and, therefore, not conducive to voluntary thought. To overcome this, and the natural resistance to thought, it is necessary to balance the syllabus by providing some subjects—such as English literature and history—which stimulate the imagination and which start the student thinking on the broadest possible lines.

The second link is the university or technical school. It is difficult for a student to view this training objectively—in fact it is not his job to try. It is worth mentioning, however, that students find that the further their course progresses, the greater the value of their vacation work. This is as it should be, but can all other technical students make this claim about their courses? At the same time, the remarks of Mr. L. C. Hill, referring to the necessity for constant review of training in relation to contemporary requirements, bear emphasis, not only from the point of view of making additions to the course, but also from that of cutting out the dead wood.

The third link is the practical training provided by the industry. It would appear that a young mining engineer needs between two and three years' practical experience before he can be considered a real asset. The divergence of opinion occurs as to how this should be obtained. The pros and cons of the two possibilities have been fully explored by Professor Ritson and subsequent speakers, and it seems that some change is due.

On graduation the student wants a responsible job of work. He wants to feel that he is a part—however small—of the team entrusted with the responsibility of obtaining ever improving results. He wants to be in a position in which he can use his ability, energy, and imagination to the full. I think these are legitimate desires.

It is not for a student, however, to suggest what is best for him when graduated; he is not in a position to know, although he is in a position to state his general desires. If the industry can contrive in some measure to satisfy these desires, very much better results will be obtained from the young graduate.

Seen in this light, the prospect of a set postgraduate course, of, say, two years, is not attractive. The alternative has been clearly set out by Mr. Lawford and Mr. L. C. Hill and is much more likely to commend itself to the student. It must not, however, be supposed that I am against postgraduate training; I contend only that it should be as short as possible and be mainly concerned with teaching local conditions and methods.

The bulk of the practical work should be done before graduation.

It remains to be examined, therefore, how the student may best obtain more practical experience before graduation. A year's work underground between school and college seems to be the answer. I suggest that when a student from school applies for a place in a college, the advantages of this scheme are clearly pointed out, and that an offer to make the necessary arrangements is made. With good liaison between college and industry this seems a practical proposition, and there would be further advantages arising from it, as pointed out by Mr. J. S. Sheppard.

Finally, there is the question of the rearrangement of college terms and vacations, in the sandwich system, in order to improve opportunities for vacation work whilst at college. This might be a practical proposition for independent colleges and bear close examination. The case of colleges attached to universities is somewhat different and there is at least one serious disadvantage. Breadth of outlook has been stressed as important. A university life helps to provide this through communal activities. This advantage would be much reduced if the mining departments worked different terms from those of the remainder of the college or university. But perhaps it would be possible to provide an extra month's long vacation by careful rearrangement, without sensibly altering the dates of the three terms?

DR. G. A. SCHNELLMANN: The mode of employing young graduates no doubt varies widely. My own (I hope unusual) experience was that, engaged by London office as an 'underground learner' at an appropriately low salary, I arrived overseas to be given immediate charge of a section of the mine, with the status and responsibilities, but not the salary, of a shift-boss.

It can, of course, be argued that this is one way of learning, but it is apt to be expensive for the company and can scarcely be described as ethical. Surely it is essential under present-day living conditions for a new graduate to be in a position to earn, which entails being worth, a substantial salary? Fewer and fewer parents are in a position to subsidize young men during the years of subsistence-level wages envisaged by many speakers. Moreover, when to the undergraduate years are added two years' military service and, say, two years at learner wages, a young man must necessarily be approaching 30 before he can anticipate the domesticity which is the normal human desire—and is, we are assured, essential to the perpetuation of the more intelligent sections of the community.

It appears to me that the solution of the problem lies partly at least in a more effective use of the vacation periods. Observation forces me into reluctant agreement with Messrs. Bergne and Symons. To a large proportion of students vacation work is merely an irksome putting in of the minimum underground time required by their college regulations. It would, incidentally, be interesting if Professor Ritson would tell us what proportion of his own students returns substantially more than this minimum period.

Futility is not, however, inherent in the vacation work system, which needs organization in place of its present apparent haphazard nature.

Cannot the Institution make effective use of its influence and prestige by canvassing mining companies with a view to establishing a definite scheme for vacation experience, bearing in mind that there are still mines in this country where a student mining engineer can acquire the fundamentals, if not the ultra-modern refinements, of making a hole in the ground safely and efficiently? This, after all, is the essence of his job now that the mining geologist on the one side and the mineral engineer on the other have taken over large sections of what was formerly his territory.

Many of the things listed by Professor Ritson could be taught effectively in well-organized vacation work, the student having to do the jobs himself and not being permitted to assume the rôle of a slightly-bored observer. Others could be taught in the mine, which should be an integral part of every mining school's teaching aids, while the remainder should be considerably pruned or entirely eliminated—e.g. twelve weeks' underground surveying and two weeks' assay office. These are the jobs which, as Professor Ritson himself says, a man can do efficiently immediately on graduating.

Probably the best postgraduate training is to 'keep moving' for the first few years, seeing and operating a variety of mining methods. By joining the typical group, be its postgraduate training scheme of the best, a graduate is in danger of over-specializing—e.g. he becomes a highly efficient banket miner, for instance, but 'hasn't a clue' how to tackle an irregular replacement deposit or even a fissure vein.

Experiments on the Removal of Selenium and Tellurium from Blister and Fire-refined Copper

W. A. BAKER, B.SC., F.I.M., and A. P. C. HALLOWES, B.SC., A.I.M.

Further contributed remarks on paper published in Bulletin 513, *August* 1949

MR. F. D. L. NOAKES: The authors are to be congratulated on the clear presentation of their report on this interesting series of experiments. It is to be hoped that their work will inspire some large-scale tests on the removal of these troublesome elements, either during fire-refining or at some earlier stage in the production of copper. The increased use of electrolytically refined copper in industry should not mask the fact that direct fire-refining is quicker and cheaper, and is therefore to be preferred unless precious metals are present in sufficient quantity to render their recovery an economic consideration.

The presence of bismuth, selenium and tellurium always gives rise to difficulties and many fire-refining troubles would be avoided if these elements could be excluded from the blister copper. The papers by Buch and by Stevens, to which the authors refer, have shown that bismuth can be and, at two smelters, is being removed on a commercial scale during the conversion of high-grade matte, even though the high temperatures involved cause heavy wear on the converter linings. H.R. Potts, in a recent paper,* has shown that bismuth and arsenic can be efficiently removed from a low-grade matte.

Previously, it has been generally agreed that the conditions that favour bismuth removal during conversion encourage retention of selenium and tellurium in the copper and, in fact, substantially all the selenium and tellurium delivered to the smelter usually passes into the blister copper. The authors, however, have shown that with partial conversion of high-grade mattes, leaving 10 to 20 per cent of the copper as white metal, the bulk of the selenium and tellurium does not enter the copper, and it seems reasonable to assume that this would also hold true for low-grade mattes. If a practical method could be evolved for separating the remaining high-selenium white metal—and this should not prove an insuperable problem—it should be possible to remove first arsenic and bismuth and then selenium and tellurium during the one converting operation. Up to 90 per cent of the blister copper produced should be suitable for direct fire-refining and only the remaining high-selenium blister from the conversion of the accumulated impure white metal would need electrolytic refining.

In view of the earlier remarks of the authors concerning the removal of selenium by volatilization, some experiments carried out by the writer in the laboratories of the Royal School of Mines may be of interest.

* Further notes on converter practice at Rio Tinto. *Bull. Instn. Min. Metall.*, No. 496, March, 1948.

In the course of certain investigations, some samples of pure oxygen-free high-conductivity copper were heated with varying amounts of spectroscopically-pure selenium shot to temperatures well above the melting point of copper. Samples of O.F.H.C. copper, weighing 100 g. and holding varying amounts of the selenium inserted into plugged holes, were placed in small individual salamander crucibles. The samples were covered with pure crushed graphite and six of these small crucibles were placed in one large salamander crucible, which was also filled with crushed graphite and was then sealed with a lid. The whole was heated in a gas injection furnace to 1250°C. and held at that temperature for 2 hours. Subsequent analysis gave the following results:

Sample	Selenium added *per cent*	Selenium found *per cent*
A.1	Nil	0·00107
A.2	0·01	0·0127
A.3	0·05	0·0541
A.4	0·10	0·106
A.5	0·50	0·488
A.6	1·0	0·857

The figures suggest that, under these reducing conditions, some selenium migrated from the high-selenium samples to the lower-selenium samples. Most of the selenium contents are considerably higher than would be found in any commercial copper.

In a subsequent experiment the conditions were similar except that the individual small crucibles were also covered with lids and that the temperature was first held at 1150°C. for 4 hours and then raised to 1300–1350°C. for 2 hours. The analyses showed:

Sample	Selenium added *per cent*	Selenium found *per cent*
B.1	Nil	Trace
B.2	0·02	0·0175
B.3	0·08	0·0706
B.4	0·25	0·241
B.5	0·75	0·687
B.6	2·00	1·68

These figures indicate some loss of selenium by volatilization. That the loss was not due merely to volatilization of elemental selenium before it became alloyed with the copper is shown by the following. After the first period of heating at 1150°C. for 4 hours the samples were examined to see if fusion had taken place. They were then replaced and reheated to 1350°C. and, during this second stage, a platinum/platinum-rhodium thermocouple, inserted through the lid of the large crucible, failed. Micro-examination of the fracture showed the presence of selenium as deep red spots on the platinum. The quantity of selenium volatilized was too small for this treatment to be considered as a possible method of removal, but these results appear to be in conformity with the authors' observations on hydrogen-blown synthetic copper melts.

No. 517 DECEMBER, 1949

BULLETIN OF THE INSTITUTION OF MINING AND METALLURGY

Principal Contents:

AN OUTLINE OF UNDERGROUND OPERATIONS AT MUFULIRA COPPER MINES, LTD.

By J. P. NORRIE *and* W. T. PETTIJOHN

Published monthly by The Institution of Mining and Metallurgy
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CONTENTS

NOTICE OF GENERAL MEETING

The Third Ordinary General Meeting of the Fifty-Ninth Session of the Institution of Mining and Metallurgy will be held, by kind permission, in the Apartments of the Geological Society of London, Burlington House, Piccadilly, London, W.1, on Thursday, 15th December, 1949, at 5 p.m.

The paper for discussion will be *Management in Industry*, Mr. F. G. Hill's Presidential Address last year to the Chemical, Metallurgical and Mining Society of South Africa, published in the July, 1949, issue of the *Bulletin*. Mr. Hill has kindly given his consent to this arrangement.

Light refreshments will be provided at 4.30 p.m. for members and visitors attending the Meeting.

The Council invite written contributions to the discussion of papers from members who may be unable to be present at the Meetings of the Institution. The Council reserve the right to edit and condense such contributions.

INSTITUTION NOTES

ELECTION OF OFFICERS AND OVERSEAS MEMBERS OF COUNCIL FOR THE SESSION 1950–51

The Council of the Institution have pleasure in announcing the election or re-election of the following Officers for the Session 1950–51:

President: Col. L. C. Hill, D.S.O., M.C.

Hon. Treasurer: Mr. Robert Annan.

Vice-Presidents: Mr. A. L. Butler, O.B.E., Mr. Tom Eastwood, Mr. Donald Gill, M.C., and Sir Arthur Smout, J.P.

The following have been nominated for election as Overseas Members of Council for 1950–51, and, as only one nomination was received for each territory and no ballot will therefore be required, will be declared to be elected at the next Annual General Meeting:

Canada, Mr. R. W. Diamond; *Australasia,* Mr. M. A. E. Mawby; *South Africa,* Mr. Kenneth Richardson; *India and Pakistan,* Mr. E. R. Dempster; *the U.S.A.,* Mr. C. O. Lindberg; *Malaya,* Mr. W. M. Warren; *Northern and Southern Rhodesia,* Mr. B. W. Durham; *East Africa,* Mr. Hugh Sandys; *West Africa,* Mr. A. T. Climas.

THE PRESIDENT-ELECT

Col. L. C. Hill was educated at St. Lawrence College and received his technical training at the Royal School of Mines, London. He graduated as B.Sc. (London) and A.R.S.M. (Mining) in 1912, being awarded the Brough Medal, and took up a post in Spain with the Rio Tinto Co., Ltd. The whole of his subsequent professional career has been with the same Company. By 1928 he had become Technical Deputy Manager, a post which he held until he joined the army in 1940. Since the end of the war, Col. Hill has been Technical Adviser to the Rio Tinto Company, with headquarters in London, and has made several visits to overseas mining fields in that capacity.

The President-Elect has a distinguished record of military service in the two World Wars. In 1914 he was commissioned in the Royal Engineers, and first served in 174 Tunnelling Coy., R.E. In 1916 he became Assistant Controller of Mines, Third Army, and in the following year took over the command of 177 Coy., with the rank of major. During World War I he was awarded the D.S.O. and M.C. In 1940 he was recommissioned and until 1942 commanded the First Tunnelling Engineers, R.E., with the rank of lieutenant-colonel. After service with that group in England and Malta, he was appointed Deputy Chief Engineer, Malta, in 1942, and Chief Engineer, Malta, in 1943, with the rank of colonel.

Col. Hill relinquished his commission under the age limit in November, 1944, and immediately joined the Admiralty Runner Service, where he served until the end of the war as a deck-hand on fleet tenders and other small craft.

Himself the son of a mining engineer —the late Alexander Hill, M.I.M.M., of Messrs. Alexander Hill and Stewart— his own son is at present a student at the Royal School of Mines.

OVERSEAS MEMBERS OF COUNCIL FOR RHODESIA AND INDIA FOR THE CURRENT SESSION

The Council announce the election, under Bye-law 30(i), of Mr. B. W. Durham and Mr. E. R. Dempster as Members of Council for Rhodesia and India, respectively, for the remainder of the Session 1949–50, to fill the vacancies caused by the death of Mr. Geoffrey Musgrave and the resignation from the Council of Mr. R. G. K. Morrison, who has taken up residence in Canada.

STUDENT APPLICATIONS

The Council have recently considered the case of those who wish to join the Institution as Students but are over the maximum age for admission laid down in the Bye-laws, owing to their having served in H.M. Forces or in other branches of national war service. To avoid injustice in such cases the Council have resolved that they will consider an application for admission to Studentship from any candidate over 28 years of age, provided (i) that his application is received before 1st January, 1953, and (ii) that he has been in war service for a period at least equal to the period by which his age exceeds 28 years.

POSIUM ON SHAFTS AND FT SINKING

e Chemical, Metallurgical and ıg Society of South Africa have shed in one volume the thirteen rs on design, sinking methods and ıtion of shafts which formed the osium arranged by the Society which were published in the *Journal* he Society for the months of mber, 1948, to February, 1949, May and June, 1949.

e Soci ty have kindly supplied the u ion with copies of the symposium me, for sal to memb rs at the ed price of £1 each. Members ring c pies should apply to the tary of the Inst tution, Salisbury k, London, E.C. 2.

'S AND VIEWS

e Council of the Institution wish gin a new section in the *Bulletin* 1 they hope will be used by mem- of all classes for contributing news, and observations on their work the industry. The Institution's cations do not at present provide for brief notes and correspond- and it is proposed to meet this by introducing the new feature.

e Council trust that the new section e welcomed by members and that will be a ready response to this ation for contributions. All com- cations for publication should be to the Secretary. It will be under- . that the responsibility of deciding ıblish communications and to edit must remain with the Council.

ARY SERVICE— NSLATIONS

e Council of the Institution wish rm a panel of translators from g members of the Institution for translation of technical articles red by members, the cost of 1 would be borne by the members hom the translations were made. pers willing to translate articles nvited to notify the Secretary of nguages and subjects they would take, and terms of remuneration.

is hoped that in this way a trans- service may be established.

NOVEMBER GENERAL MEETING

Three papers were submitted and discussed at the Second Ordinary General Meeting of the Session, held on Thursday, 17th November, at Burlington House. They were: *Factors affecting the rate of formation of zinc ferrite from zinc oxide and ferric oxide*, by Mr. D. W. Hopkins, which, in the absence of the author, was introduced by Mr. G. L. Evans, Lecturer in Metallurgy at the Royal School of Mines; *Gold concentration at the Amalgamated Banket Areas reduction plant*, by Mr. G. Chad Norris, *Member*, introduced by Mr. Robert Annan; and *Some notes on a mechanical concentrator*, by Mr. T. Haden, *Associate Member*, introduced by Mr. E. A. Knapp. The last two were discussed together; a report of the Meeting will be published in the January issue of the *Bulletin*.

REMAINING MEETINGS IN THE SESSION 1949–1950

The dates of the General Meetings, to be held on the third Thursday in each month, during the remainder of the Session are as follows:

19th January, 1950
16th February, 1950
16th March, 1950
20th April, 1950
18th May, 1950
15th June, 1950

JANUARY GENERAL MEETING

The paper entitled *An outline of underground operations at Mufulira Copper Mines, Ltd.*, by Messrs. J. P. Norrie and W. T. Pettijohn, which is published in this *Bulletin*, will be submitted for discussion at the General Meeting to be held next month.

MEMBERS FROM ABROAD

The Council are always anxious to meet members who come to England after a long absence abroad, and ask such members to make themselves known to the Secretary when attending General Meetings of the Institution at Burlington House.

CANDIDATES FOR ADMISSION

The Council welcome communications to assist them in deciding whether the qualifications of candidates for admission into the Institution fulfil the requirements of the Bye-laws. The application forms of candidates (other than those for Studentship) will be open for inspection at the office of the Institution for a period of at least two months from the date of the *Bulletin* in which their applications are announced.

The following have applied for transfer since 10th November, 1949:

To Membership

Bhag Singh Lamba (*New Delhi, India*).
Ian Webster Morley (*Brisbane, Australia*).
Geoffrey Charles Norman (*Obuasi, Gold Coast*).
Thomas Pickering (*London*).
William Pulfrey (*Nairobi, Kenya*).
Leslie Vivian Trewartha (*Turk Mine, Southern Rhodesia*).

To Associate Membership

Henry Lawrence Butlin (*Johannesburg, Transvaal*).
Cecil Frederick Hopkinson (*Edwinstowe, Nottinghamshire*).
William James Marshall (*Winsford, Cheshire*).
Esmé William David Pritchard-Davies (*Springs, Transvaal*).
Ronald Teale (*Coventry, Warwickshire*).
Desmond Evered Wright (*Morococha, Peru*).

The following have applied for admission since 10th November, 1949:

To Membership

Kenneth Arthur Davies (*Entebbe, Uganda*).
Frank Ludwig (*Springs, Transvaal*).
Adolphe Charles Georges Pienne (*Ipoh, Malaya*).

To Associate Membership

Edwin Bennetts (*Penzance, Cornwall*).
William Findlay (*Rasa, Malaya*).
Robert Beverley Harrison (*Prestea, Gold Coast*).
Olaf Prins Hazlitt (*Kuala Lumpur, Malaya*).
Ian Donald Mackay (*Johannesburg, Transvaal*).
Eli Margo (*Maraisburg, Transvaal*).
John Tunn'el Pullen (*Tarkwa, Gold Coast*).

To Affiliateship

José Pina de Aragao e Costa (*Lisbon, Portugal*).
Barry Hugh James Edmond (*Kuala Lumpur, Malaya*).
Archibald Graham (*Dunedin, New Zealand*).
John Neill Jordon (*Salisbury, Southern Rhodesia*).
Robert Neil Millar (*Johannesburg, Transvaal*).

To Studentship

Edward Leonard Averill (*London*).
William Alexander Brandum (*Barking, Essex*).
David Griffiths (*London*).
William Stanley Hickson (*Camborne, Cornwall*).
Terence William Hulme (*Chester, Cheshire*).
Bernard Walter Locke (*Wembley, Middlesex*).
John Patrick Merryweather (*Konongo, Gold Coast*).
Edric Arthur Millard (*Deal, Kent*).
Stephen Nwachukwu Ofomah (*Camborne, Cornwall*).

TRANSFERS AND ELECTIONS

The following were transferred (subject to confirmation in accordance with the conditions of the Bye-laws) on 10th November, 1949:

To Membership

William John Alborno (*Konongo, Gold Coast Colony*).
Ernest Thomas Edward Andrews (*Barberton, Transvaal*).
William James Bichan (*Regina, Saskatchewan, Canada*).
Gilbert Frederick Hatch (*Johannesburg, Transvaal*).

To Associate Membership

John Anthony Desmond Bell (*Prestea, Gold Coast Colony*).
Denzil Vincent Sydney Dunn (*Filabusi, Southern Rhodesia*).
Gerald Percy Hutton (*Dunedin, New Zealand*).
Peter Alexander Nicholls (*Tarkwa, Gold Coast Colony*).
William Peter Holford Parkinson (*Maryport, Cumberland*).
John Eric Rockingham (*Kaduna Junction, Northern Nigeria*).
Terence Albert Rodgers (*Vatukoula, Fiji*).

The following were elected (subject to confirmation in accordance with the conditions of the Bye-laws) on 10th November, 1949:

To Membership

Stacey George Ward (*Edgbaston, Birmingham*).

To Associate Membership

William Davies (*Sheffield, Yorkshire*).
John Raymond Fletcher (*Kuala Kubu, Malaya*).
P. N. Vijaya Raghavan (*Oorgaum, India*).
Struan James Cunynghame Robertson (*Kisumu, Kenya*).

To Affiliateship

George Furnace Brown (*Rio Tinto, Spain*).

To Studentship

John Anthony Eaton Allum (*London*).
Kenneth Ernest Sylvester Applin (*Camborne, Cornwall*).
John Lindon Ashford (*Chandlers Ford, Hampshire*).
Anthony René Barringer (*London*).
John Hunter Bennie (*Camborne, Cornwall*).
George H. Boyle (*Uppingham, Rutland*).
James Gerald Brading (*Purley, Surrey*).
Brian Buckley (*Doncaster, Yorkshire*).
Michael Henry Cleary (*Camborne, Cornwall*).
Patrick Denis Coakley (*Gwithian Towans, Cornwall*).
Harry Kemp Cole (*Bedford, Bedfordshire*).
John Hamilton Crawford (*Kalgoorlie, Western Australia*).
Michael Dore Cruikshanks (*Herne Bay, Kent*).
Andrew Hugh Philip Fabel (*Reading, Berkshire*).
Isaac Goldberg (*Que Que, Southern Rhodesia*).
Keith Francis Gosling (*Mt. Hawke, Cornwall*).
Thomas Albert Winfield Haddon (*Camborne, Cornwall*).
Frederick John Trevor Hancock (*Kidderminster, Worcestershire*).
Robert William Alan Hansel (*St. Ives, Cornwall*).
Peter Henry George Hayward (*London*).
Thomas Andrew Henderson (*Hexham, Northumberland*).
John Michael Wren Humphreys (*London*).
Bruce Jones (*N'kana, Northern Rhodesia*).
Leong Sing Lim (*Camborne, Cornwall*).
Tony Bernard Lock (*Kingston-on-Thames, Surrey*).
John Carlo Loretto (*Montreal, Canada*).
Keith Edward Mantell (*Hayle, Cornwall*).
Edmund Charles Henry Mardell (*Enfield, Middlesex*).
Thomas Oswald Martyn (*Wadebridge, Cornwall*).
John Walton Martyr (*Camborne, Cornwall*).
William Garth Barrington Phillips (*Framlingham, Suffolk*).
James Potter (*St. Ives, Cornwall*).
Frederick Raynes (*Camborne, Cornwall*).
Peter William Edward Richardson (*Camborne, Cornwall*).
Mohan Singh (*Camborne, Cornwall*).
Michael Wyndham Stephenson (*Camborne, Cornwall*).
Arthur Raymond Tron (*Gateshead, Co. Durham*).
Peter Lloyd Walker (*London*).

NEWS OF MEMBERS

Members, Associate Members, Affiliates, and Students are invited to supply the Secretary with personal news for publication under this heading

Erratum: In the announcement in the August *Bulletin* it was not stated that Mr. R. G. Head's new post at Mufulira was in the Mine Technical Department. The title of the post has recently been changed to 'senior engineer designer in the Mine Technical Department'.

Mr. B. Beringer, *Member*, has left Palmiet Chrome Mines, Ltd., Johannesburg.

Mr. R. Bowie, *Associate Member*, expects to leave Britain early next month for Tanganyika to take up a position with Geita Gold Mining Co., Ltd.

Mr. A. V. Bradshaw, *Associate Member*, has returned to England from Portugal.

Mr. L. Bristow, *Member*, has left England for South West Africa.

Mr. E. C. Bromwich, *Student*, is now working in Northern Rhodesia as learner mine official with Roan Antelope Copper Mines, Ltd.

Mr. P. M. Bussy, *Associate Member*, has returned to England from Sierra Leone.

Mr. A. W. Clark, *Associate Member*, has returned to South Africa.

Mr. J. Davey, *Member*, returned to Chile last month.

Mr. H. I. Dunstan, *Student*, has taken up an appointment at Mysore Mine, Marikuppam, South India.

Mr. P. P. Edwards, *Associate Member*, has left England on his return to India.

Mr. O. G. H. GALE, *Member*, has returned to England from a vacation in Cyprus.

Mr. L. GOUDIE, *Associate Member*, resigned his post with the North of Scotland Hydro-Electric Board a few months ago in order to take up an appointment with Naraguta Tin Mines, Ltd., Northern Nigeria.

Mr. J. H. JACKSON, *Associate Member*, has left England for Northern Rhodesia after leave here.

Mr. A. M. JANE, *Student*, is now employed at the Grootvlei Mine, Springs, Transvaal.

Mr. H. A. LAVERS, *Member*, is returning to England this month on resigning his position as reduction officer with the St. John del Rey Mining Co., Ltd., Brazil.

Mr. D. H. McCALL, *Associate Member*, has returned to the Kolar Gold Field, India, after leave in the United Kingdom.

Mr. J. E. MENZIES, *Associate Member*, has returned to the United Kingdom from the Gold Coast.

Mr. H. W. MILLETT, *Associate Member*, will leave England shortly for Saudi Arabia.

Mr. P. G. F. MONEY, *Associate Member*, mining engineer for Atlas Steels, Ltd., has left Canada for an extended tour of Mexico, Central and South America.

Mr. L. E. T. PARKER, *Associate Member*, has returned to Abadan, South Iran, from England.

Mr. T. I. PINER, *Student*, has joined the staff of Malayan Tin Dredging, Ltd., Batu Gajah, Perak.

Mr. THOMAS PRYOR, *Member*, has left England on a visit to India and expects to return at the end of January.

Mr. H. J. STOCKDEN, *Associate Member*, has been transferred from Western Reefs Exploration and Development Co., Ltd., to South African Land and Exploration Co., Ltd., Brakpan, as mill foreman.

Mr. P. L. VAUGHAN, *Student*, has taken up employment with Blyvooruitzicht Gold Mines, Ltd., Transvaal.

Mr. Jack WILLIAMS, *Student*, is leaving South West Africa for England early next month.

ADDRESSES WANTED

R. Milton Thomas	J. A. Cocking
K. A. Knight Hallowes	A. S. Rogers
D. S. Broadhurst	A. J. W. Walser

OBITUARY

CLARENCE VIVIAN PAULL died on 17th November, 1949, at the age of 51. He entered the School of Metalliferous Mining (Cornwall) in 1919, and on obtaining his Diploma in 1922 he accepted employment in South Africa at Van Ryn Deep, Ltd. There he worked for 2½ years as assistant sampler, then on machine developing and stoping and, finally, for fifteen months as shift boss. He returned home to take up, in January, 1925, the position of assistant manager of South Crofty mine, Carn Brea, Cornwall, of which his father was general manager. He succeeded his father, the late Captain Josiah Paull, J.P., M.I.M.M., in 1930, and became a director of South Crofty in 1945.

Mr. Paull, who was a member of the Board of Governors of the Camborne School of Mines, was elected to Associateship of the Institution in 1934 and was transferred to Membership in 1947.

BOOK REVIEW

Metals Reference Book. COLIN J. SMITHELLS, ed. London: Butterworths Scientific Publications, Ltd., 1949. 735 p., 500 figs. 60s., by post 61s. 3d.

There can be no doubt that this is one of the most important reference books on metals that has appeared in the English language. It contains something for every kind of metallurgist, but unfortunately the bias is strongly towards the physical side, so that its appeal to members of this Institution is likely to be limited, as may be judged from the following outline of the contents.

After fourteen pages of tables of general interest, such as conversion factors, there are about 200 pages of mathematical formulae and physical data, intended mainly for the metal physicist. This is followed by a few pages of generally interesting information relating to geochemistry, then nearly 200 pages for the physical metallurgist, including an admirably comprehensive collection of equilibrium diagrams of alloy systems.

The next 37 pages contain thermodynamical data, mainly of interest in connection with the physical chemistry of extraction metallurgy, and the next 40 are devoted to physical properties, e.g. thermal, electrical and magnetic.

The following 70 pages should appeal to engineers as well as to metallurgists, as they are packed with useful information relating to the mechanical properties of metals and alloys.

The remainder of the book, comprising about 140 pages, covers a variety of subjects of practical interest, including lubricants, fuels, refractories, foundry data, corrosion, electro-plating and welding.

Most of the information is given in tables and diagrams. It has been critically selected, and skilfully condensed so as to pack the maximum amount of data into the minimum of space. This arrangement is admirable so long as the reader has a good knowledge of the particular branch of the subject about which he is consulting the book, but if he has to turn to unfamiliar subjects he may have difficulty in understanding what he finds. In most of the sections there is a certain amount of descriptive and explanatory matter, concise, well-informed and clearly expressed. Most readers would probably appreciate a considerable increase in the amount of this written matter, even if it had to be made at the expense of some of the tabulated data or diagrams. Bearing in mind, however, the fact that this is strictly a reference book and is not intended to serve as a text-book, congratulations and thanks are due to the contributors, editors and publishers for an outstanding piece of work. The book should be in every metallurgical library.

M. S. Fisher.

ADDITIONS TO JOINT LIBRARY OF THE INSTITUTION AND THE INSTITUTION OF MINING ENGINEERS

Books (excluding works marked *) may be borrowed by members personally or by post from the Librarian, 424, Salisbury House, London, E.C.2

Books and Pamphlets :

Bergwerk und Probierbuchlein. A translation from the German of the 'Bergbüchlein' a sixteenth-century book on mining geology, by Anneliese Grünhaldt Sisco and of the 'Probierbüchlein' a sixteenth-century work on assaying, by Anneliese Grünhaldt Sisco and Cyril Stanley Smith, with technical annotations and historical notes. N.Y.: A.I.M.E., 1949. 196 p., illus. $5.

* *Canadian mines handbook 1949.* Toronto: Northern Miner Press, 1949. 360 p., tabs. $2.

* British Engineers' Association. *Classified handbook of members and their manufactures.* 1949. London, British Engineers' Association, 1949. 650 p.

Draney, Jack. *Diesel locomotives: mechanical equipment: a practical treatise on the operation and maintenance of railway diesel locomotives.* Chicago: American Technical Society, 1944. 472 p., illus., diagr. 31s. 6d.

* Kirkham, Nellie. *Derbyshire lead mining glossary,* Cave Research Group Publication, No. 2. Leamington Spa: Cave Research Group, 1949. 34 p., illus., biblio. 3s.

* *Oil and petroleum year book, 1949.* Skinner, Walter, E., comp. London: Skinner, 1949, 252 p., 15s.

Institute of Petroleum. *Reviews of petroleum technology, vol. 8 (covering 1946).* George Sell, ed. London: The Institute 1949. 445 p. tabs., biblio.

* *South African mining and engineering year book, 1949-50. Incorporating the Rhodesian mining year book.* S. R. Potter, ed. J'burg: South African Mining Journal Syndicate, 1949. 567 p., illus., tabs. 45s

Wood, William. ***Planning and the law: a guide to the Town and Country Planning Act,*** **1947. London: Percival Marshall, 1949. 288 p. 21s.**

Government Publications :

Egypt, Geological Survey Dept. ***Terrestrial theories: a digest of various views as to the origin and development of the earth and their bearing on the geology of Egypt,*** **by W. F. Hume. Cairo: Government Press, 1948. 160 p., illus., maps, tabs., diagrs., biblio. P.T. 250.**

Eilender, Walther, senior author. ***Ferrous metallurgy, vol. 2. (F.I.A.T. review of German science, 1939-1946.)*** **Germany: Military Govt., 1949. 373 p., biblio. (German text.)**

Gold Coast, Mines Dept. ***Report . . . for the year 1948-49.*** **Accra: Govt. Printer, 1949. 17 p., tabs. 1s.**

Great Britain, Imperial Institute. ***The mineral industry of the British Commonwealth and foreign countries. Statistical summary (production, imports and exports),*** 1941–1947. London: H.M.S.O., 1949. 341 p., tabs. 21s.

Nigeria, Geological Survey Dept. *Annual report for the year 1948.* Lagos: Govt. Printer, 1949. 55 p., maps, tabs, diagrs. 9d.

Ontario, Dept. of Mines. *General index to the reports of the Ontario Department of Mines, volumes 36 to 49 (1927–1940),* by E. A. Trevor: Toronto: King's Printer, 1949. 505 p.

South Australia, Dept. of Mines. *Mining review for the half-year ended 31st December, 1947, no. 87.* Adelaide. Govt. Printer, 1949. 214 p., maps, diagrs., tabs.

Proceedings and Reports :

American Institute of Mining and Metallurgical Engineers. *Transactions, coal division, vol. 177, 1948.* New York: The Institute, 1949. 594 p., illus., tabs., diagrs., biblio. $4.50.

Canadian Institute of Mining and Metallurgy. *Transactions, vol. 51. 1948.* Montreal: The Institute, 1949. 302 p., illus., diagrs., tabs., biblio.

The Chamber of Mines of Rhodesia. *10th annual report for the year 1948.* Bulawayo: The Rhodesian Technical Press, Ltd., 1949. 36 p., tabs.

International Geological Congress (18th Session). *Il giacimento piombo zincifero di Raibl in Friuli (Italia): memoria presentata alla 18a sessione del Congresso Internazionale di Geologia —Londra, 1948,* by Dino di Colbertaldo. Rome: Cura della S.A. Miniere cave di Predil, 1948. 147 p., illus., diagrs. maps, tabs., biblio.

North East Coast Insitution of Engineers and Shipbuilders. ***Transactions, sixty-fifth session, 1948-49 vol. 65.*** **Newcastle upon Tyne: The Institution, 1949. 458 p., + 168 p., illus., tabs.**

Transvaal Chamber of Mines. ***Proceedings at the Annual General Meeting, held in Johannesburg, Monday, 27th June, 1949.*** **J'burg: The Chamber, 1949. 21 p.**

Maps :

A La Peche map-area, West of Sixth Meridian Alberta. Geological survey, preliminary map 49–7A. Scale 1 in. = ½ ml. Ottawa: Dept. of Mines and Resources, 1949. (Summary account).

Carp Lakes, District of Mackenzie, Northwest Territories. Geological survey, preliminary map 49–8. Scale: 1 in. = 4 ml. Ottawa: Dept. of Mines and Resources, 1949.

Collins Point, West of principal meridian, Manitoba. Geological survey, preliminary map 49–9. Scale: 2 in. = 1 ml. Ottawa: Dept. of Mines and Resources, 1949.

Pitchblende occurrences between Beaverlodge and Hottah Lakes, Northwest Territories. **Geological survey, map 49–16. Scale: 1 in. = 500 ft. Ottawa: Dept. of Mines and Resources, 1949. 2 sheets. (Paper, 49–16.)**

EX OF RECENT ARTICLES

of world literature on economic geology, mining (excluding coal), mineral dressing, extraction metallurgy, subjects, classified according to the Universal Decimal Classification. All articles indexed are available at Library but the current issues of journals are not available for loan.

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An Outline of Underground Operations at Mufulira Copper Mines, Ltd.*

J. P. NORRIE, B.SC.† AND W. T. PETTIJOHN‡ 622.343 (689.4)

A paper published on 8th December, 1949, *by the Institution of Mining and Metallurgy, to be submitted for discussion at the January,* 1950, *General Meeting.*

SYNOPSIS

The paper describes the geology and physical characteristics of the orebodies and country rocks at Mufulira and traces the development of mining methods from the commencement of operations up to the present time.

The problems encountered and the methods adopted for overcoming them are discussed. Current mining methods—shrinkage stoping, sub-level stoping, and block caving—are described in detail, as are the latest developments in blast-hole drilling. The paper also covers transport, underground crushing, hoisting, drainage, ventilation, and the organization of labour, and concludes with some data on mining costs.

INTRODUCTION

THE MUFULIRA COPPER MINE lies in latitude 12° 32′ S., longitude 28° 15′ E. in Northern Rhodesia, ten miles from the Belgian Congo border, and is one of the group of four operating mines comprising the Northern Rhodesian Copperbelt. The mine is connected to the outside world by road, rail and air, the airport being at Ndola, 45 miles to the south. The climate is sub-tropical, with definite wet and dry seasons of six months' duration, an annual rainfall of about 50 in. between November and April, and temperatures varying from 34° to 95°F. The elevation is 4,100 ft. above sea level. The surrounding country is gently undulating and well covered with hardwood timber.

The Rhodesian copperfields were prospected in the late 1920's and the mines came into production in the early 1930's. By the time the recent war had started the district was a major copper producer. The importance of the Mufulira mine in the field is best indicated by the amount of copper produced during the six years ended June, 1945; this was 487,982 long tons of blister copper, an average of 76,330 tons per annum, the highest annual production being 85,523 long tons.

GEOLOGY

The ore deposits at Mufulira are on the limb of a large syncline striking NW.–SE. and dipping NE. at about 45° towards the Congo border. The host rocks are well-consolidated, highly-silicified, felspathic quartzites of the lower part of the pre-Cambrian ' Séries des Mines ' of the Katanga. The ore rocks are underlain, unconformably, by the older ' Muva Series ', in this case hard, green, metamorphosed quartzites.

There are three superimposed orebodies, the first, or upper, orebody averaging 40 ft. true thickness with a strike length of about 5,000 ft., the second, or middle, orebody of about 50 ft. true thickness with a

*** Paper received on 4th July, 1949.**

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‡ Formerly mine superintendent, Mufulira Copper Mines, Ltd.

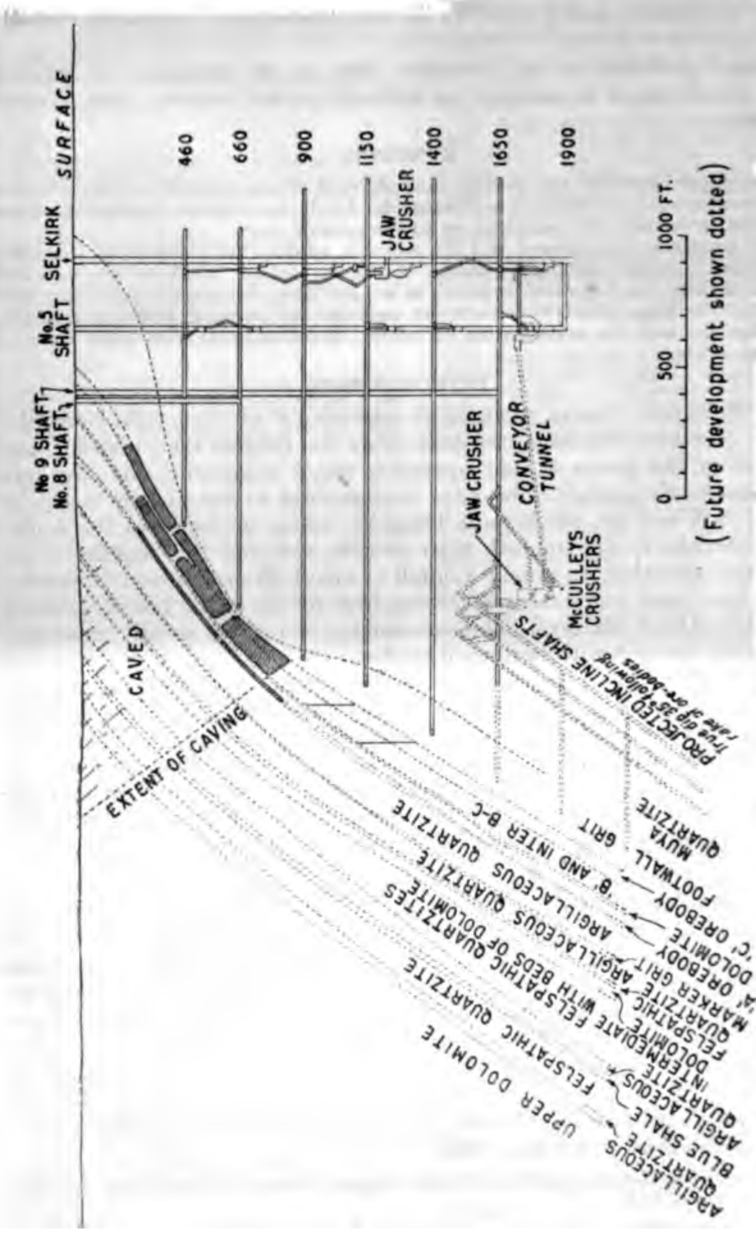

Fig. 1.—Section through main shafts showing orebodies and hanging-wall beds.

strike length of about 6,000 ft., and the third or lower orebody of around 60 ft. true thickness with a strike length of 8,000 ft. The orebodies are separated by varying widths of waste, or very low-grade ore, from a few feet up to 50 ft. The middle and lower orebodies are so close together for a strike length of 3,000 ft. that they are mined as one orebody. In all, the mineralized zone is about 200 ft. in true thickness.

Mineral distribution is remarkably even, both across the width of the orebodies and along the strike. There is, however, a falling-off in width and amount of copper mineralization towards the fringes, particularly to the west; so much so that when the mineralized width has dropped to about ten feet the copper content is not commercial.

Mineralization, occurring as bornite, chalcopyrite, and chalcocite (in order of importance), consists of disseminated particles with occasional stringers and veinlets cutting across the bedding or in joints and other lines of weakness. Malachite, cuprite, azurite, chrysocolla, and native copper are found, but not in quantity. Down to about 900 ft. below surface there is considerable secondary enrichment, generally chalcocite.

The orebodies, as such, do not outcrop. Commercial ore ceases at from 150 ft. to 350 ft. vertically below surface. There is a pronounced rake to the north. Diamond drilling has outlined the ore down to 3,000 ft. depth and the proved ore reserves now stand at 86,233,000 tons of 4·05 per cent total copper and 0·07 per cent oxide copper; the actual reserves are probably much greater.

The immediate hanging-wall of the mineralized zone is a 200-ft. thick band of argillaceous quartzite overlain by dolomite shales and quartzites, all heavily water-bearing (see 'Drainage'). The immediate foot-wall of the lower orebody is a cross-bedded sandstone, or grit, often poorly consolidated, sometimes water-bearing, but generally improving in depth. It is in this grit that haulages, ore-passes, and other development for ore-gathering must be excavated. The underlying Muva quartzite, already mentioned, is in such a position as to be the rock generally selected for shaft sinking, pump stations, crusher stations and other big excavations, and for this purpose it is ideal.

The origin of the ore is still controversial, but, at present, the 'syngeneticists' can produce more and better evidence in support of their theories than can the 'epigeneticists'.

DEVELOPMENT

The mine was originally opened up by a series of inclined shafts at intervals along the strike, sunk either in the ore horizons or in the foot-wall, but these shafts have now been abandoned, except in a few instances where they have been retained for access and ventilation.

The main ore handling and servicing system for the mine consists of two vertical rectangular shafts, Selkirk and No. 5, which extend to the 1,900-ft. level and which are connected to the orebodies by main cross-cuts on the 460-ft., 660-ft., 900-ft., 1,150-ft., 1,400-ft., and 1,650-ft. levels. Below the 1,650-ft. level ore handling and servicing will be through 35° sub-inclined shafts, following the rake of the orebodies, using the 1,650-ft. level as a service and waste transfer level. Ore transfer from the sub-inclined shafts to the vertical shafts will be by a 48-in. conveyor belt

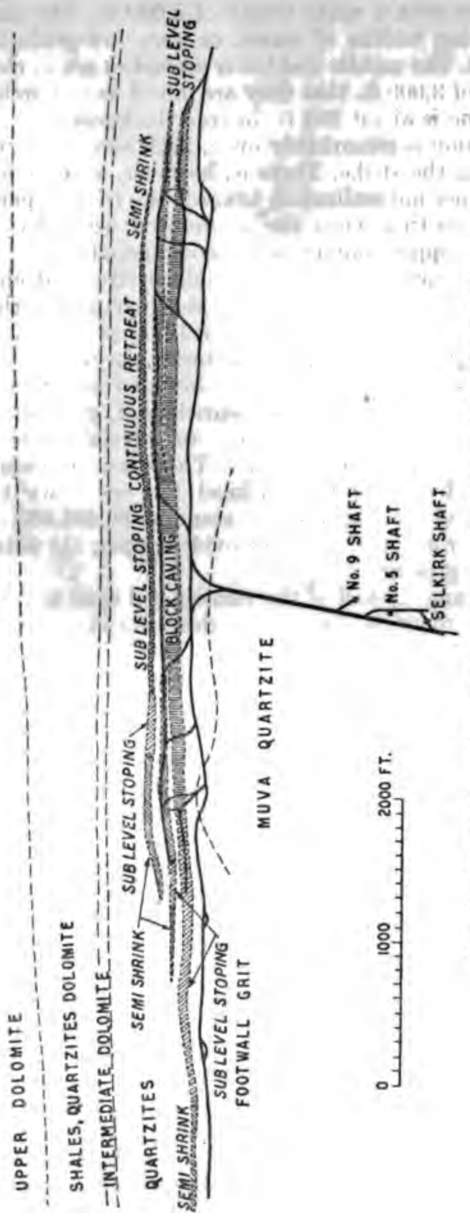

FIG. 2.—Plan of 1,150-ft. level showing orebodies and mining methods.

1,100 ft. long. The ore will be crushed before reaching the belt.

There are two main ventilation shafts, one upcast and one downcast, each 22 ft. in diameter, vertical, and smooth-lined with concrete. The downcast shaft extends to the 1,650-ft. level and is connected to each main cross-cut which is working. The upcast shaft is connected to each main cross-cut above which the ore has been mined out—at present the 460-ft. and 660-ft. main cross-cuts. All main cross-cuts below the 900-ft. level are of 22 ft. by 11 ft. section, to allow of double-track haulage and to accommodate the large quantities of air required for ventilation.

From the end of the main cross-cut the haulages turn off to follow the foot-wall of the orebody and at varying distances below it. This distance below the orebody is determined by the mining system adopted and has varied from 20 ft. to 90 ft. On all levels down to and including the 1,400-ft. level haulages are also provided in the orebodies, following the strike, in order to handle rock from separately-mined orebodies. On the 1,650 ft. level the orebody haulages are to be replaced by cross-cuts from the foot-wall haulage into the orebodies, so that tramming on this latter haulage will not be delayed by runaways from ore-passes.

All haulages are provided with ditches and, in the case of all levels from the 1,150-ft. level down, a separate travelling way is run parallel to the main foot-wall haulage, about 20 ft. from it and from 2 ft. to 3 ft. lower in elevation. This travelling way accommodates all main air, machine, water, and drainage pipe-lines, as well as keeping foot traffic off the haulage and dealing with excess drainage water which might otherwise delay the tramming.

Haulages are generally of 10 ft. by 11 ft. section. All chute raises, ladder-way cross-cuts, and other openings, if anticipated, are collared as the haulage advances. When the haulage has progressed to a point determined by the orebody being mined and the mining method to be used, secondary development is started—that is, the running of ore-passes, mining raises, ladderways, and sub-levels. Since all mining now tends towards a retreat system, this secondary development is concentrated on the initial mining faces and then follows a regular retreat pattern ahead of the mining.

The position of haulages and secondary development is generally determined by diamond drilling from the level above. Drainage development—that is, running cross-cuts into the hanging-wall strata—is an important one and is given precedence on any new level.

In driving haulages four 3½-in. drifter machines are used, mounted on standard bar gear. A drag round with 32 holes gives an advance of 8 ft. per shift. Rounds are cleaned out by 30-h.p. scraper slides, electrically operated.

Sub-level headings of 6 ft. by 6 ft. cross-section are advanced 4 ft. per shift, using a 3½-in. drifter machine and spoil is generally removed by shovel and wheelbarrow. Use is also made of 10-h.p. air-driven double-drum hoists in scraping out headings such as 'scram drives' where conditions are favourable. In raises 3-in. or 3½-in. stoper machines are used.

The steel used is 1-in. quarter-octagon, with conventional 4-point forged

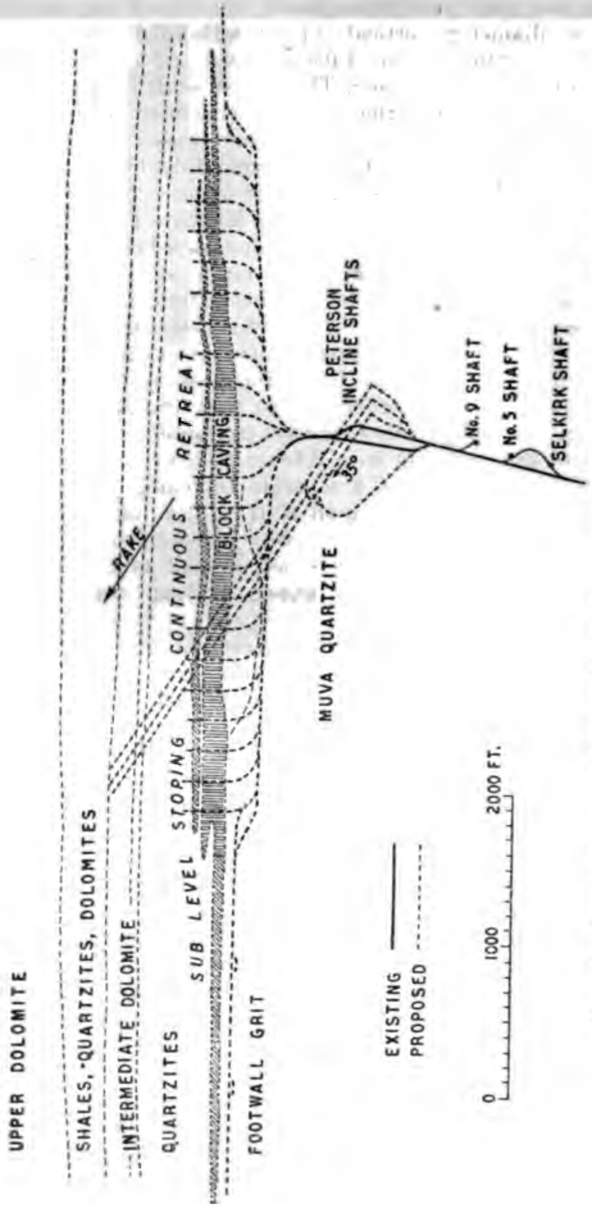

FIG. 3.—Plan of proposed 1,650-ft. level, showing orebodies and mining methods.

bit. With the exception of that used for jackhammers and for main haulages, all steel is straight-shanked. Detachable bits are used in sections of the mine involving awkward transport.

HISTORY OF MINING METHODS

The original diamond drilling from surface outlined the orebodies fairly completely and indicated that while the various bodies were well separated by leaching action above about the 460-ft. level, much ore could be mass-mined below that level.

During the underground exploration of 1930 and 1931 it was realized that a major drainage problem would have to be solved before any mining method involving movement of the hanging-wall beds could be entertained. Inrushes of water from a leached-out dolomite between the top and middle orebodies caused temporary abandonment of sections of the mine and nearly resulted in the loss of the pumps on several occasions. This drainage problem coloured all discussions on the early mine planning and this is still true to-day, since, with increasing depth and pressure, no method of roof support is practicable and drainage of the hanging-wall strata must precede mining.

The first stopes planned in 1931 were to have been above the 360-ft. level and extending to the top of the ore at about 260 ft. from surface. These stopes were to have been sub-level stopes about 150 ft. on the dip and 150 ft. along the strike, leaving island pillars as required. A considerable amount of development had been done, but no stoping actually commenced, when, owing to low copper prices, the mine was closed down in December, 1931, and subsequently allowed to flood.

Unwatering operations were started in July, 1933, and by the end of the year the mine was in a position to produce a small tonnage. Development was continued for a period on the pre-shutdown basis and several short-back stopes were mined from the 360-ft. level. It was soon found, however, that the width along the strike had to be reduced below the planned 150 ft. with island pillars—particularly in the second (middle) orebody, with its overlying manganese-wad and decomposed dolomite.

During this period the Roan Antelope and Nkana mines had come into production, both using sub-level mining methods in ore thickness up to about 40 ft. The success of these comparatively high lift, short-strike width stopes influenced Mufulira to abandon the 360-ft. level as a haulage and adopt the 460-ft. level for the purpose. This entailed stopes of about 300 ft. to 350 ft. on the dip. After much experimenting it was found that a stope width of 60 ft. along the strike was generally satisfactory, although this had to be reduced to 40 ft. in some cases.

Stopes were, of necessity, prepared in the first place in known high-grade areas and practically all development work was confined to ore. After the trial and error period all stopes were arranged so that the pillars in the three orebodies should be superimposed, the pattern emerging being 60-ft. stopes, with 15-ft. pillars, which were robbed as the ground dictated, there being six stopes to a block of 455 ft. on the strike with a ladderway in the orebody in the centre. Depending on the thickness of the orebody, these were sub-level 'bench and trail' stopes or semi-shrinkage 'back' stopes.

The ore was scraped down the foot-wall of the stope into the chute raises and thence by gravity to the haulages.

At first no method of roof support was used. The middle orebody stopes generally collapsed quietly within a month or so of completion, whereas in the first (upper) orebody, with its strong argillaceous quartzite hanging-wall, as much as 1,000 ft. with pillars was opened up along the strike before it collapsed. The hazard to life caused by the air blast occasioned by the sudden collapse of such an excavation, apart from flooding dangers, called for some method of controlling, if not entirely preventing, roof movement. With this in mind experiments with sand-filling were started in 1935, but it was some time before it reached a stage at which it became a routine part of the mining operation.

In 1936 the first attempts at mining the middle and lower orebodies as one were made. The usual bench and trail methods were applied, but the number of sub-level entrances to the stope faces was increased; the dimension of 60 ft. along the strike was retained. The true thickness varied from 100 ft. to 120 ft., or up to 180 ft. on the horizontal.

These stopes were between the 660-ft. and 460-ft. levels, with a back length of from 250 to 300 ft. At first no method of backfill was used and the hanging-wall was allowed to collapse, prior drainage having been effected. So long as the hanging-wall was weathered and possessed no great strength this caving caused little trouble as regards airblasts, but it was obvious that such would not be the case where weathering did not occur, so that sandfilling was developed to the stage where systematic filling of stopes could take place as soon as the mining had been completed.

One of the chief difficulties with these outsize stopes was in disposing of the ore broken. Foot-wall drawpoints (Fig. 6) were adopted, the ore being either scraped along foot-wall scram drives to a central ore-pass by three 100-h.p. scraper hoists, or passed through grizzlies, at each drawpoint, to a branching ore-pass system to the haulages. In this case, the gravity system was found to be better suited to the conditions and more productive of tonnage than the scraper system.

During the period 1936 to 1940 several variations of the sub-level open stope method were tried. Influenced by developments elsewhere ring drilling with rockdrills was used, both in sub-levels retreating along the strike and in raises retreating up dip. After large-scale tests extending over two years ring drilling with rockdrills was discontinued, because of slow and costly drilling in the hard quartzites, excessive rod and bit breakages, loss of gauge (requiring large starting bits up to 3½ in. in diameter), and, principally, because the thickness and attitude of the orebodies did not offer any appreciable reduction in development footage with the length of holes it was found possible to drill. It is possible, however, that the tungsten-carbide bit may alter views on past experience with long holes.

Diamond blast-hole drilling was introduced in 1939 and large-scale tests were carried out in stopes varying from 40 ft. to 120 ft. in true thickness. Holes varied in size from 1⅛-in. diameter to 1½-in. diameter (E special), the most successful crowns being coring, thin wall, with crushed boart in pressed and sintered matrices. Here again the hard and often variable rocks presented a drilling problem which is not yet solved. In the narrow stopes—say of 40 ft. to 50 ft. in true thickness—the flat dip, together with the accumulation of large rock chunks, involved difficult scraping, while in the thick orebodies it was found that the heavy blasting caused damage to the narrow pillars left between stopes as sand-fill retainers. Owing to this, as well as to the fact that facilities for developing a suitable bit were not available in war time, large-scale diamond blast-hole drilling was discontinued. More recently research on diamond crowns has given a new impetus to blast-hole drilling and considerable use of the diamond drill is now being made in pillar blasting.

The mining pattern evolved at this time—that is, about 1939-1940—was one of superimposed stopes of 60-ft. strike width with nominal 15-ft. pillars, in blocks of six stopes, retreating to a central ladderway. The mining and sandfilling sequence was: Mine two lower orebody stopes and sandfill; mine the overlying middle orebody stopes and sandfill; mine the overlying upper orebody stopes and sandfill. In the meantime the next adjacent lower orebody stope would be mined, and so on. When the combined middle and lower orebody were mined together the sequence was the same. Sandfilling was one of the most important operations. Practically all the tonnage mined during the war period was obtained in this manner.

As knowledge of the drainage problem and its solution developed and because of the possibility of obtaining better recovery than in stope and pillar methods, attention was again directed to mass-mining methods. At the same time, it was becoming apparent that sandfilling, while serving a very useful purpose down to moderate depths of, say, 1,000 ft., could not continue to provide adequate roof support or—and this was more important—prevent crushing of pillars. Means had to be devised for dealing with pressure on remnants.

For the combined middle and lower orebodies 'Shrinkage and Pillar Caving' was introduced. This consisted of alternate shrinkage stopes and pillars, each 30–60 ft. along the strike and extending between main levels. The pillars were undercut and wrecked by coyote blasts between shrinkage stopes and the broken ground drawn off from a system of foot-wall draw-points as a panel along the strike. An ore to waste draw plane of 45° to the horizontal was maintained, retreating from the ends of the combined middle and lower orebodies towards the centre.

Several sections were prepared and mined in this manner and the panel draw started. While the shrinkage stopes provided easy mining, undercutting of pillars between stopes was difficult, in that small pillars, fatal to satisfactory caving, were sometimes left and attempts had to be made to remove them by diamond blast-holes drilled from the foot-wall workings. The coyote blasts were none too satisfactory, although persistence in using them would probably have improved results.

The pillar-undercutting difficulty led to the suggestion of continuous undercutting, in blocks 25 ft. square, the dip being around 40°. At the same time it was proposed to weaken the mass of the ore by drilling from cross-cuts at 25-ft. centres on a vertical plane, thus instituting sub-level caving.

At this time it was considered essential that the orebody should be weakened, as experience on caving hard bedded deposits was not extensive. Weakening from cross-cuts was, at first, intended to be done by diamond blast-holes from chambers to be drilled around the cross-cuts, using the latter as 'cuts'. This weakening method was never actually applied; it was discarded in favour of a series of V-cuts, using 12-ft. holes, along the cross-cuts, the result being a series of zig-zag pillars, with little strength, across the thickness of the orebody. The V-cuts were later formed by using sprung holes—that is, holes 8 ft. deep drilled in pairs 8 in. apart, sprung with small charges, and finally blasted with 20-25 lb. of 60 per cent ammon gelignite or dynamite. These sprung holes were satisfactory for their purpose. Undercutting was made continuous along the strike, retreating level by level, from the top of the panel downwards. Undercutting became, in effect, a series of small rill stopes of around 30 ft. back length on the dip. As a result of local experience in sub-level caving and of a study of methods used in the United States and in Chile, block caving was introduced, at first using the same mass-weakening measures as in the sub-level caving. The change was made to circumvent the hazards of continuous undercutting and to allow of better accounting for the ore, as well as to provide a greater production rate for a given area. The foot-wall draw system, a pattern of drawpoints at 25 ft. centres when projected on a horizontal plane, with scram drives covering 200 ft. along the strike or 100 ft. on either side of a central ore-pass system, remained practically unchanged.

It was found that prior weakening of the mass of the ore was not necessary and that the stresses set up by regular undercutting were sufficient to break up the ore, despite its great strength. This is a comparatively recent development, but full block caving is at present standard practice in the sections of combined middle and lower orebodies.

In dealing with the narrower orebodies, of up to about 40-ft. thickness, where the relative amount of foot-wall development in waste rock would make caving methods expensive, the tendency has been to think along 'open stope' lines. The first ideas were to carry on mining with open sub-level stopes, of shorter dip length, and blast out the remaining rib and crown pillars, allowing, or, if necessary, inducing the hanging-wall to cave behind the working faces. Two mines in the Copperbelt use such methods successfully. Owing to the comparatively flat dip and to the strength of the hanging-wall quartzites it was decided to omit the rib pillars and to have a series of crown pillars, three per main haulage interval, to be blasted in sections as the face retreated. The initial caving of the hanging-wall beds was to be induced by slots cut in them across the dip and along the strike.

This method—'Continuous Retreat Open Stoping'—has been applied to the upper orebody, which must now be mined out in advance of the middle and lower orebody caving blocks, and will be applied to the sections of lower orebody on the fringes where there is no overlying orebody. This continuous retreat method has not been used for a sufficiently long period to lay many plans, but there seems no reason why it should not be successful down to moderate depths of, perhaps, 2,000 ft. from surface.

CURRENT MINING METHODS

The current mining methods are: (1) semi-shrink or back stoping; (2) sub-level stoping (conventional); (3) sub-level stoping (continuous retreat), and (4) block-caving.

Semi-Shrink or Back Stoping

This method is used in the single orebodies, where the ore thickness is less than 25 ft.

Ladderway raises are run from one haulage level to the next above at 455-ft. intervals along the strike, with mining raises between at 75-ft. intervals, giving the stope a dimension of 60 ft. on the strike by 300 ft. or over on the dip by the ore thickness of 25 ft. or less (Fig. 4). Where

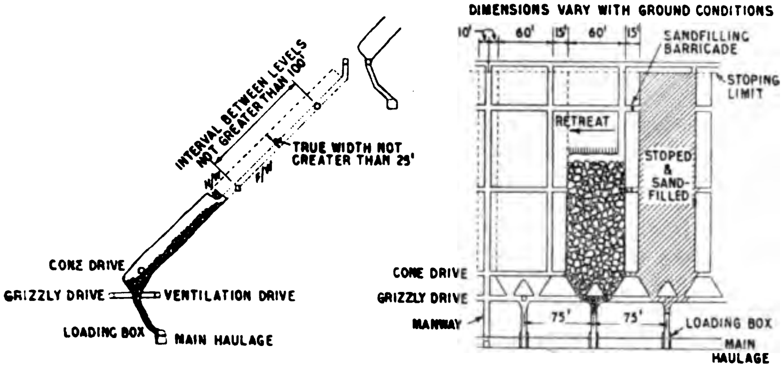

FIG. 4.—Semi-shrink or back stoping.

ground conditions require it the strike width of the stope may be reduced to 40 ft. or 30 ft. and the dip length shortened. Pillars of 12 to 15 ft. are left between stopes.

Ore chutes on the haulage are placed at 75-ft. intervals and connect to a grizzly drive 40 ft. above, where cross-cuts are run into a grizzly or bull-dozing chamber below each stope. A double-ended three-bar grizzly, with a 24-in. opening, is fed by ore drawn from two cones serving the 60-ft. stope. The next level up the dip is the cone drive, about 25 ft. higher vertically than the grizzly level, and this is used for opening up the stope face. A parallel drive nearer the hanging-wall is used for scraping when dips are flat and the ore will not run. Other sub-levels, run for convenience in mining and for ventilation purposes, connect the mining raises to the ladderways at intervals of from 60 to 90 ft. on the dip.

When development is well advanced, and after ladderways have been equipped and grizzlies installed, stoping operations commence with the cutting of the cones between the grizzly level and the cone drive. This is done by stoper machines shrinking up on the ground and retreating from the cone raises. As soon as the cones are complete, a square face across the complete 60-ft. strike width of the stope is established and carried up dip using stopers drilling off the broken ground which has to be drawn daily to take care of the swell and mucked back from the face because of the flat dip. As most hanging-wall rocks are strong and ore thicknesses are not over 25 ft., little difficulty is encountered in maintaining a safe mining face.

Access to the stope, while mining, is obtained through the various sub-levels from the main ladderway raise, and a ladder is also maintained in the mining raise from the stope face to the sub-level above.

This class of stope normally requires one European miner, who supervises a gang of about seventeen Africans—five machine crews, a boss boy, and a helper. Production varies with stope widths and conditions, but 6,000 tons per month is a fair average.

Good ventilation is obtained by bringing fresh air from the main ladderway, through the sub-level in use, to the end of the stope where it can sweep the whole face, blowing air into the mining raise which connects directly to a return airway at the top of the stope.

Sub-Level Stoping (*Conventional*)

This method was first introduced to this mining district at the Roan Antelope mine in 1931 and is a method that was used in all the mines of the Copperbelt and is, therefore, described here as conventional.

The main haulage level, as shown in Fig. 5, is run in the foot-wall about 40 ft. from the orebody. Then, 40 ft. in elevation above the main haulage, two drives are run, one in ore for entrance to the grizzly cross-cuts and one in foot-wall waste for exhausting blasting fumes. Grizzly cross-cuts are situated at 75-ft. intervals along the strike, corresponding to the chute raises from the main haulage. Next, 25 ft. in elevation above the grizzly drives, there are two sub-levels, one run over the centres of the cones, or drawpoints, and one run farther back in the hanging-wall. From this second drive cross-cuts are run through to the cone drives at the centre of the drawpoints and scrapers are set up here when the dip is too flat for the ore to run down the foot-wall of the stope. Above the cone level sub-level drives are spaced in the centre of the ore at 28-ft. vertical intervals for convenience in mining. Mining raises on hanging-wall and foot-wall are run at 75-ft. intervals and 'manway-service' raises are run at 455-ft. intervals from one main haulage level to the next. As in the semi-shrunk method the stope dimensions are 60 ft. on the strike, by 300 ft. or more on the dip, by the ore thickness, which may vary from 25 ft. to 60 ft.

A 15-ft. pillar is left between stopes and so arranged that the pillars in the three orebodies are superimposed one on another. The best sequence of mining found by experience is to mine the lower orebody first, then the middle orebody, and then the upper orebody, with sandfilling introduced following the same sequence.

The stope is opened by belling out the cone from the grizzly level to the cone drive, using stoper machines. As soon as the first cone is completed, a slot is cut at the end of the stope, removing the rock between the hanging-wall and foot-wall mining raises. This is usually done by stoper machines, shrinking upwards on the broken ground from the cone level right to the top sub-level of the stope. This operation is a slow one

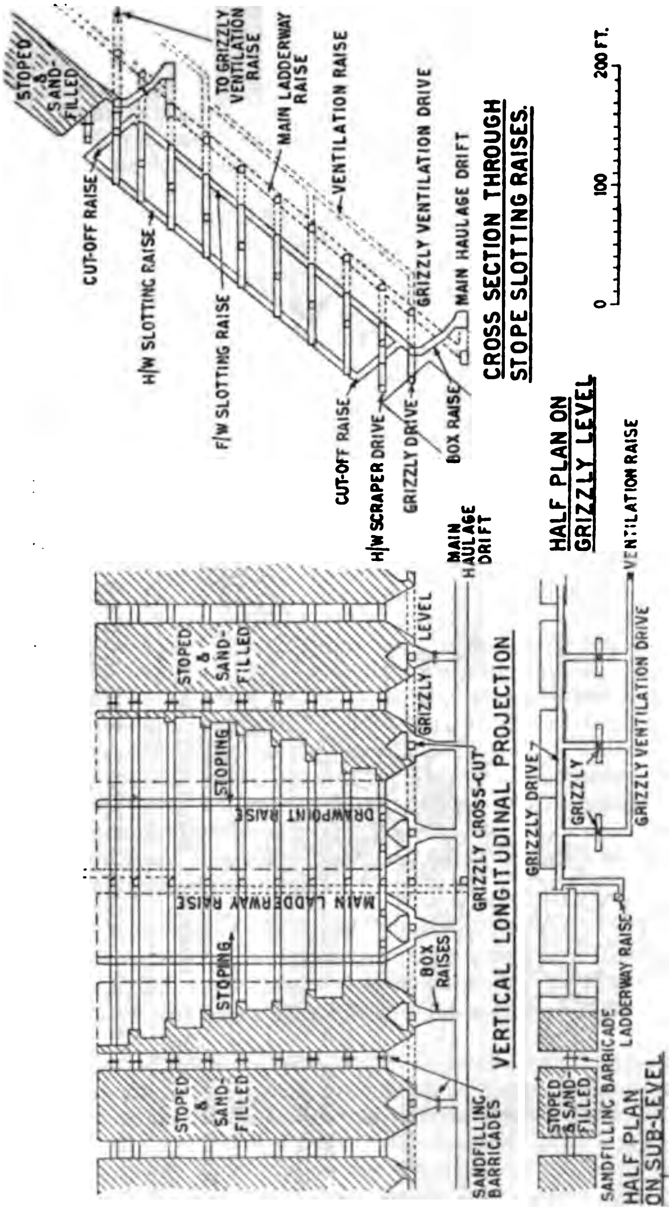

Fig. 5.—Sub-level stoping, single orebody.

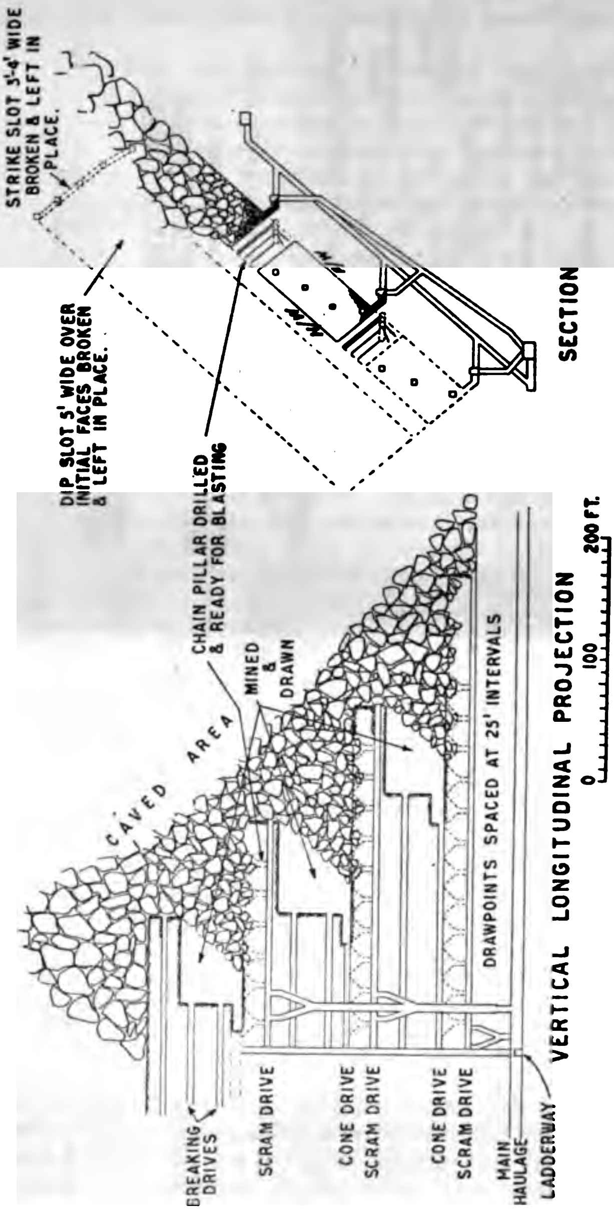

FIG. 6.—Sub-level stoping (continuous retreat).

and the tonnage gained from shrink-swell is about 2,000 tons per month. After shrinking is completed, trails are run to the hanging-wall and foot-wall on the various sub-levels and benches prepared for blasting. The trailing work is done by $3\frac{1}{2}$-in. drifter machines, the drilling of down holes on the benches by 3-in. jackhammers, and the up holes by $3\frac{1}{8}$-in. stoper machines. The average depth of hole drilled in stopes is about 8 ft. This operation of benching and trailing continues towards the ladderway until the stope face reaches the 15-ft. pillar, which is left between all sub-level stopes.

Each stope is run by a European miner, supervising a crew of thirteen Africans running five rock drills. The average monthly tonnage obtained from this type of stope is about 6,000, including the slotting period.

Ventilation is maintained, as in the semi-shrink method, by introducing fresh air, obtained from the ladderway intake, by means of a 14-in. ventilation pipe, with a fan blowing on to the stope bench. The mining raises are connected to a return airway in the crown pillar at the top of the stope, where the vitiated air carries on through the return air raises to the main exhaust fans and is not used again. As already mentioned, a grizzly ventilation drive, run below the foot-wall of the stope, is used to exhaust secondary blasting fumes and this opening also connects with return air raises and the exhaust fans and is the return airway for stopes coming from the next lower haulage level.

Sandfilling is still used in conjunction with sub-level stoping and, in the lower orebody, not more than two stopes are allowed to stand open without fill. In the upper and middle orebodies not more than four stopes are allowed to stand open without fill. Unless this practice is followed fairly closely stopes cave without much warning, causing air blasts, and also tearing into adjacent unmined stopes, causing losses in extraction. The cost per ton of sand placed as fill is at present about sixpence. The sandfilling operation has been well described by A. C. Turton.*

Sub-level stoping costs per ton, excluding development, vary from about 4s. 9d. to 3s. 6d., according to the amount of tonnage mined, the width of ore, and other local conditions.

Extraction from this class of mining varies from 60 to 80 per cent, depending on ground conditions.

Sub-Level Stoping (Continuous Retreat)

This system of mining is used in the upper orebody, in the sections over the 'Block Caving' areas, where it is desirable, in any case, to have the caving initiated and where sandfilling would be more of a hindrance than a help.

In this method there is a series of three open sub-level stopes with back lengths of about 85 ft. between main haulage levels. These stopes retreat continuously along the strike, leaving crown pillars of about 30 ft., which are blasted out in sections of approximately 70 ft. along the strike.

To start the caving of the strong hanging-wall rocks in this mining area a dip slot of about 300 ft. back length by 3 or 4 ft. wide, extending

* Sandfilling at Mufulira. *Bull. Instn. Min. Metall., No.* 478, May, 1946. *Trans. Instn. Min. Metall.,* 56, 1946-47, 79.

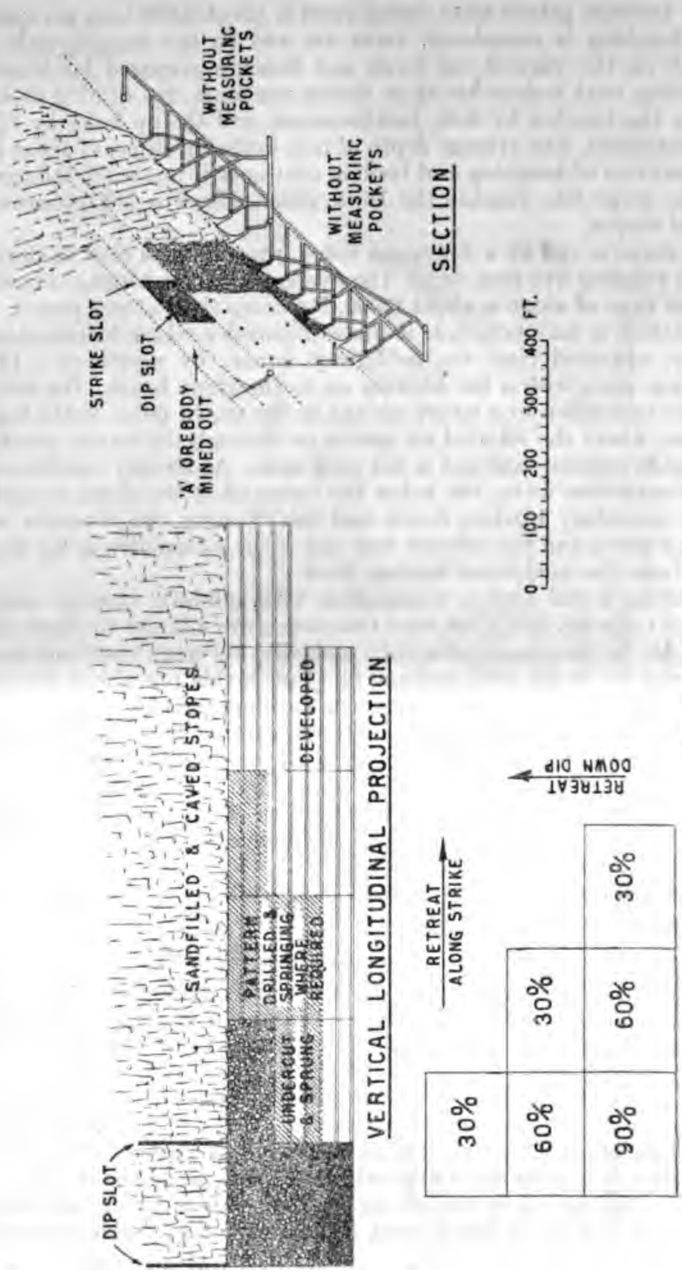

Fig. 7.—Block caving.

about 90 ft. into the hanging-wall, and a strike slot of similar dimensions, at right angles to the dip, and also in the hanging-wall, are cut. It is thought that the strike length of this latter slot should be several hundred feet, but this point has not yet been settled.

Draw from the stopes is through foot-wall openings to scraper drifts located below each crown pillar, where the ore is scraped to ore-passes spaced at 200-ft. intervals.

The rock-breaking operation carried on in the individual sub-level open stopes is exactly the same as that for conventional sub-level stoping.

Pillar drilling is done by high-speed diamond drills, core-drilling $1\frac{7}{16}$-in. diameter (Ex) holes varying in depth from 17 ft. to 50 ft. These holes are drilled from cross-cuts, in patterns as near as possible parallel to the strike, in order to blast the rock down to the drawpoints before caving waste rock runs in, in the interests of better extraction.

The anticipated extraction from this method of mining is about 85 per cent and it is expected that it will supplant sub-level stoping (conventional) methods in the other separate orebodies after more experience has been gained. Mining costs per ton for this class of work are expected to be less than those obtained in other sub-level methods because the slow and costly slotting process is partly eliminated.

Ventilation arrangements for continuous retreat are similar to those for the conventional sub-level stoping.

Block Caving

This method of mining is applied to the middle and lower orebodies (Figs. 2 and 3) where they run together over a strike length of about 3,000 ft. and the true thickness varies from 100 ft. to 120 ft. Two retreat faces are in the process of being established, one in the west, between the 660-ft. level and the 1,400-ft. level, and one in the east, between the 900-ft. level and the 1,400-ft. level. Blocks are approximately 200 ft. by 200 ft. in plan, and the sequence of mining is on an offset checker board pattern.

An ore-pass and ladderway system is run in the foot-wall waste rock from one main haulage level to the next at 200-ft. intervals along the strike, in the centre of each block. Scraper levels are established at 25-ft. intervals vertically. About 15–20 ft. under the orebody foot-wall scram drives are run along the strike. Branch ore-passes connect each scram drive to the main ore-pass, which transfers ore to the main haulage level below. On either side of the ore-pass, in each scram drive, four drawpoint raises are cut at 25-ft. intervals into the foot-wall of the ore. This development scheme thus forms a 25 ft. by 25 ft. horizontal pattern of drawpoints, with a 100-ft. maximum length of scrape to the ore-pass, which is bridged with a heavy steel 24-in. opening grizzly, carrying a 30- or 50-h.p. double-drum scraper hoist.

After waste development is completed the development in ore is carried out, first running the cone or undercutting drives, which follow the foot-wall contours, connecting up drawpoint raises. Boundary-weakening cross-cuts are then run from foot-wall to hanging-wall at the strike ends of the block. Weakening drives are also run along the down dip side or north

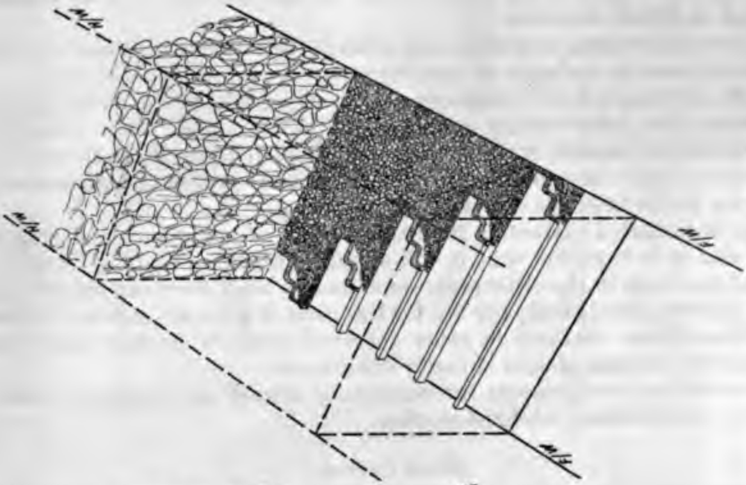

Fig. 8a.—Block caving—undercutting sequence,

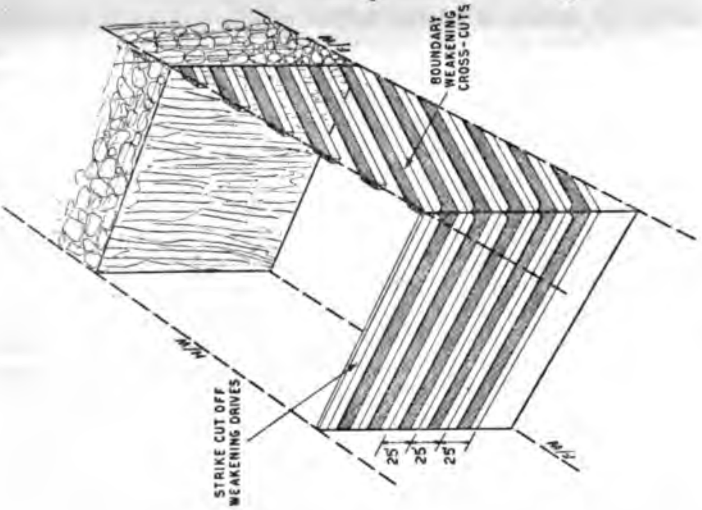

Fig. 8.—Block caving—boundary weakening.

ındary of the block, 25 ft. vertically over one another. More recently ı ore work has been accomplished by entering this development area n the upper orebody in advance of any waste work done from the foot-l side. This defines the block limits and establishes an accurate foot-l before drawpoints are laid out. Once a block is established the 'elopment continues along the strike at a rate determined by the duction required from the particular face.

nitial blocks require extra boundary development extending into the ıte between the middle and upper orebodies. A dip slot is cut on one ɜ extending from the hanging-wall of the middle orebody to the foot-.l of the upper orebody, and a strike slot is cut at the top end of the

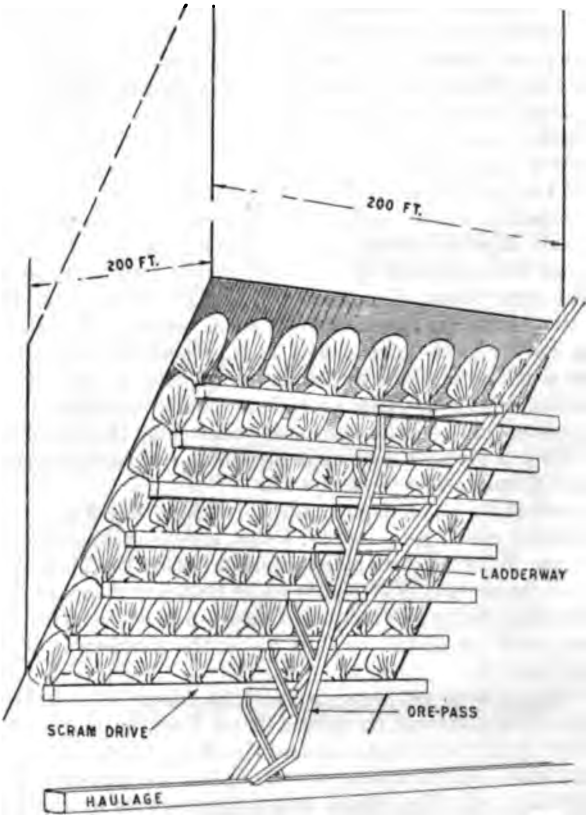

Fig. 9.—Block caving—foot-wall gathering system.

block covering the same vertical distance. This work ensures that the arch will break as undercutting progresses.

Undercutting is started at the top corner of the block, working down dip and retreating along the strike from the cave to the solid in each case. Stoper machines are used carrying inclined shrink stopes of 6 to 8 ft. in width from one cone level to the next, forming a checker board pattern. As undercutting nears completion at any particular horizon a certain amount of additional breaking is done in the boundary cross-cuts and drives, the backs being ripped down for 10 ft. and the 'solid' corner cut through from level to level.

The results of cantilever action breaking up the block can be observed as the undercut progresses. Bedding planes tear apart and joint planes open up. When natural caving forces come into action the apparently massive quartzites break up with much conchoidal fracturing, the cracks caused by undercutting appearing almost vertically above the working face in the various sub-levels as the mining progresses down dip. Heretofore, at this stage, the blasting of sprung holes, in order to induce the cave, was carried out, but recent observations show that no more than undercutting and boundary weakening is generally required.

After the completion of the undercut work, a slow and carefully-controlled draw is started, favouring the down dip or solid side of the block slightly. Drawing at this stage is at the rate of about 3 in. per day and this continues until the block is about 30 per cent drawn. The rate of draw is then stepped up to about 4 in., or a maximum of 6 in. per day, until drawn to completion. The differential in draw allowed between adjacent drawpoints is 5 per cent, between adjacent scram drives 10 per cent, and between adjacent blocks 30 per cent. Control of the draw is very important and this work is under the supervision of engineers in the Mine Technical Department, who give the draw orders which are passed on to the operating officials. Data from draw reports is recorded daily. From these reports profiles of the various sections through the block are kept up to date. In addition to this measuring of the ore from each draw-point, regular grab sampling is done and geological observations made to see that the ore is being drawn properly. When a block is nearing completion, low assays determine when a drawpoint should cease to be drawn.

Various attempts have been made to remove the human factor from draw control as much as possible. Draw sheets calling for a variety of scoopfuls from different drawpoints were not easy to check. It was found better to call for certain drawpoints to be pulled and record the scoopfuls of ore, there then being less chance of paper-perfect reports. Adjustments to draw are made on a shift-to-shift basis; the number of cars filled acts as an overall check.

Certain blocks were prepared with short raises or measuring pockets between the drawpoint and the scram drive. Each measuring pocket has a round-timber grizzly on top to control the flow of rock. The pockets were filled on one shift, the drawpoint closed off, and the measured rock scraped on the following shift. That these measuring pockets were not the success that had been anticipated is attributed to the very variable rock column above the drawpoint. Some drawpoints drawing, for example, from the ıter-orebody rocks would run like water and flood the scram drive, while thers in graywacke would fill the measuring pocket with large chunks. ŧecourse had then to be made to scoop counting. Control devices might e installed at the lower end of the measuring pockets, but with a maxiıum of 7,000 tons to be drawn from a drawpoint there is a limit to

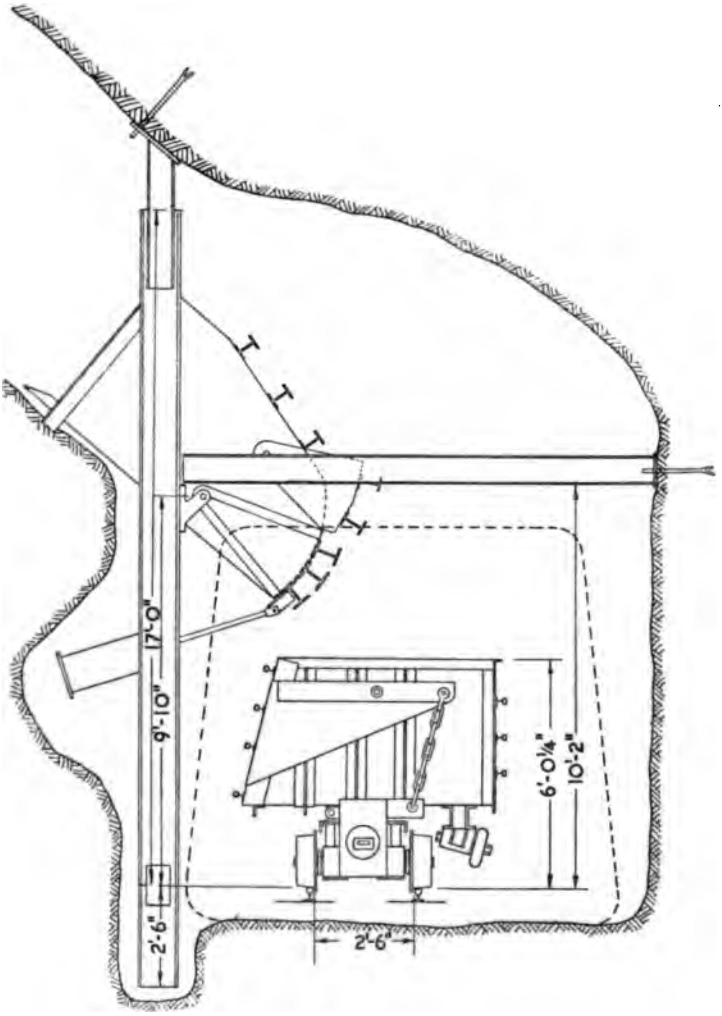

Fig. 10.—General arrangement of heavy type stope chute and 180-cu. ft. Granby car.

expenditure. Draw control is a problem still awaiting a satisfactory solution.

A series of experiments on caving and drawing was run, using a glass-sided model to a scale of 1 to 100. The rock was crushed to the approximate scale size and fritted together by means of a gypsum-base cement. The experiments with simulated sandfilled stopes above the draw area were particularly useful. Visual inspection of the progress of these experiments was of much value in instructing the operating officials.

The average monthly production from a block is 20,000 to 25,000 tons and this is obtained on two scraping shifts. A scraper crew for one block consists of eight Africans, with one European supervising two crews. Secondary blasting powder consumption is at the rate of about 1 lb. per 3 tons of ore.

Block caving costs per ton have varied widely in recent years, owing to the fact that a broken reserve was being prepared and that various schemes to induce caving were being tried, but last year's (1948) cost, excluding development, was 1·63s. per ton. It is expected that a cost of 1·5s. per ton can be obtained.

Throughout the development of the block caving method close contact was maintained with the Climax Molybdenum Co., and exchange visits between engineers of the two properties took place. Such items as mapping of caveability and the study of rock-behaviour by means of scale tests were copied directly, if with some modification, from Climax.

The Climax engineers deserve full credit for their share in developing a mining method that appears to meet the case.

TRANSPORT

The haulage of ore on the various main levels is done in 180-cu. ft. Granby-type cars, in trains of eight cars pulled by 8-ton General Electric Co. trolley locos, over 30-in. gauge track on 60-lb./yd. rails. Most of the waste hauling is done in 92-cu. ft. Granby cars, using either an 8-ton G.E.C. trolley loco or 6-ton Goodman or G.E.C. battery loco. Main haulage development work is serviced by a battery loco or a trolley loco with a gathering reel. Powder cars, drill-steel cars, and heavy material cars are all constructed on the Granby-type chassis.

All locos and all types of cars can be run directly into the man cage of the Selkirk shaft and can be transferred easily from level to level. Routine maintenance of rolling stock is done underground, at the various car repair shops, but any major overhaul is done in the surface shops.

At present the bulk of the ore trammed is gathered from stope chutes in the various foot-wall haulages on the 900-ft. and 1,150-ft. levels and hauled to a main ore-pass system at the Selkirk shaft, where the ore gravitates to the 1,200-ft. level crusher station. After crushing, the ore is hoisted to surface from the 1,350-ft. level loading pocket.

A small amount of stope ore is trammed on the 660-ft. level along a foot-wall haulage to an auxiliary ore-pass and then re-trammed on the 900-ft. level. Some development ore is trammed on the 1,400-ft. level and hauled to the main ore-pass at Selkirk shaft, where the ore gravitates to

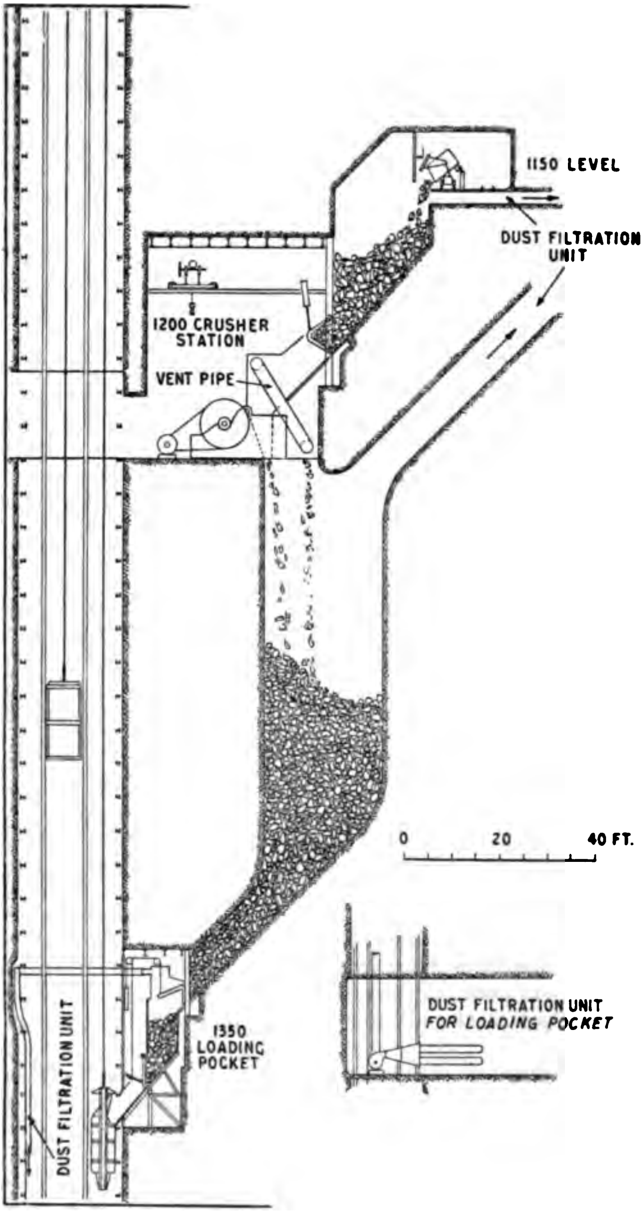

FIG. 11.—General arrangement of 1,200-ft. crusher station and 1,350-ft. ading pocket at Selkirk shaft.

the 1,850-ft. loading pocket and is then hoisted to surface.

Five trains are employed on each of the two hauling levels described, working on three shifts for six days a week.

Stope chutes are constructed of steel and are of the undercut arc gate type operated by an air cylinder, and have a gravity spillage door fitted above the air-operated door.

The average tonnage of ore being trammed at present is about 260,000 tons per month and, including waste, the total amount hauled is 290,000 tons per month. The cost per ton of ore and waste trammed is about 0·68s.

Waste hauling is done on all levels below 900 ft. and tipped into waste passes at No. 5 shaft.

UNDERGROUND CRUSHING

All ore trammed from stopes is passed through a 56-in. by 72-in. jaw crusher (Birdsboro Buchanan) on the 1,200-ft. level and is reduced to *minus* 12 in. before being dropped into a 1,500-ton storage pocket for hoisting. Control of ore feeding to the crusher is by means of large air-operated overcut claw-type fingers, which replaced the original Ross feeders, but it is now considered that the best type of feed control for Mufulira ore is a pan feeder, which will be used in future installations.

Ore trains tip directly into an open hole on the 1,150-ft. main haulage, where no grizzly is required because of the large crusher. A considerable amount of secondary blasting has been eliminated by the underground crusher. The most notable savings obtained by the use of the underground crusher are found in the maintenance of the shaft-loading pocket and in the even and regular loading of skips—very necessary for fast hoisting in the shaft.

The ventilation of this crusher station is maintained in good order by means of a 40-in. 20-h.p. axial-flow type fan delivering 30,000 cu. ft. per min. at 2·25-in. s.w.g. into flannel bags on the 1,150-ft. level. The dust produced is collected at the throat of the crusher and exhausted up a raise to the filtration unit; the whole station is under positive pressure, fresh air coming directly from the Selkirk shaft.

The crushing cost per ton of ore hoisted, based on present production rates, is about 0·08s.

HOISTING

At the Selkirk shaft two hoists are in use, one hoisting ore in balanced skips and the other operating the man cage, balanced by a counterweight. Both hoists are products of Messrs. Metropolitan Vickers and Vickers Armstrong, Ltd.

The *ore hoist* has two 1,600-h.p. D.C. motors, with Ward-Leonard control, and the maximum rope speed is 2,500 f.p.m. This hoist is semi-automatic, with push-button control at the loading pockets. With the present skip factor varying between 10 and 11 dry tons of ore, the peak capacity of the shaft hoisting from the 1,350-ft. level loading pocket is about 600 tons per hour. The present hoisting rate is 10,000 tons per day and it is planned to increase this to 11,000 tons per day, or 285,000 tons per month.

The *man hoist* is fitted with one 1,600-h.p. D.C. motor, with Ward-Leonard control, and has a maximum rope speed of 1,800 f.p.m. The cage used at present has two decks and carries 136 persons, but it is planned to install a new four-deck cage of lighter construction and an additional motor to take double the load. This will be necessary when all men and material are lowered to the 1,650-ft. level, where they will have to transfer to the future sub-inclined shafts to reach the future lower working levels.

No. 5 shaft is equipped with a smaller hoist, using a 350-h.p. A.C. motor, hauling skips of three-ton load in balance at a maximum speed of 800 f.p.m. At present waste only is hoisted in this shaft from the various levels, and the total tonnage per month is about 25,000 to 30,000 tons. This waste is hoisted from pockets below the 1,150-ft., 1,400-ft., and 1,650-ft. levels and tipped in a 750-ton bin on surface, where a six-ton battery loco and three (Hudson) 80-cu. ft. side-tip cars haul the waste to a dump adjacent to the caved areas.

No. 5 shaft is to be completely re-equipped, in order to have assurance of hoisting capacity while on peak production. This shaft has compartments large enough to accommodate the same size of skips as the Selkirk shaft and will be used for both ore and waste hoisting.

The hoisting cost, which includes maintaining and running both No. 5 and Selkirk shafts is, on present production rates, about 0·50s. per short ton of ore.

DRAINAGE

As has already been mentioned drainage must precede mining. Because of the amount of water involved and the difficulty in assessing results with any degree of accuracy, drainage presents the greatest single problem in the mine. While water may be found in any of the rocks, the three main waterbearing horizons are :

(1) The 'Lower or Inter-Orebody Dolomite,' about 15 ft. thick, which forms the hanging-wall of the middle orebody;

(2) The 'Intermediate Dolomite,' about 140 ft. thick and 200 ft. above the hanging-wall of the upper orebody. The actual 'dolomite' is about 60 ft. thick, the remainder being calcareous sandstones and shales.

(3) The 'Upper Dolomite,' upwards of 2,000 ft. thick and about 250 ft. above the Intermediate Dolomite. It consists of dolomites, shales, sandstones and quartzites, and is known to be weathered to great depths.

The Lower Dolomite is weathered to a depth of from 600 ft. to 900 ft. from surface and in that area is replaced by an iron and manganese wad. Drainage of this horizon interfered considerably with early mining. It was, however, found possible to drain it effectively by means of NX ($2\frac{15}{16}$ in. dia.) diamond-drill bleeder holes, which developed flows of from 500 to 1,000 g.p.m.

Drainage of the talcose Intermediate Dolomite by diamond drill-holes, was not effective. Various sizes of holes up to 7 in. in diameter were used, but it was found that, despite high pressures and good flows being struck, it was not possible to keep the holes from choking. Recourse was had to cross-cutting into the water-bearing horizon, water-tight doors being installed and spiling adopted where necessary. All cross-cuts are piloted by diamond drill-holes, careful logs being kept of cores, and particularly of core losses, which generally indicate soft horizons. The work is necessarily slow, but has been rewarded by flows of as much as 8,000 g.p.m. Many different methods were tried in traversing wet and broken ground, including cementation and the use of a steel shield, but timber spiling was found to be the best.

The *water-tight doors* are of local design, are made of mechanite and are of domed construction and ribbed. These doors are designed to stand a static head from surface according to their elevation underground. The door installation is in a by-pass tunnel, the main cross-cut being provided with a concrete plug and three 8-in. or 12-in. valves, air and water lines, and electric cables. The door is at a higher elevation than the plug. These arrangements are made so that, in case of a flood, there will be less likelihood of the door becoming jammed with floating timber or sand.

All cross-cuts into the Intermediate Dolomite have given good flows of water. When this happens the door is closed and the water piped directly to a pump station. When the flow drops the cross-cut is sometimes reopened and advanced to the next water horizon.

Pressures on water-tight doors and, when possible, flows of water, are recorded weekly. While a cross-cut is being run a daily check is made on the water passing over a V-notch weir.

Only one cross-cut (at the 860-ft. level) has, as yet, reached the Upper Dolomite. A considerable amount of drainage must, however, have taken place through fissures and cracks between the various water-bearing horizons. Subsidence of the surface, not connected with mining, has taken place over wide areas and this must be due to drainage only.

On surface, a network of churn-drill holes provides a partial correlation between underground water pressures and surface water levels and also helps to measure the progress of the drainage. The final test, however, is whether or not caving on a new level brings in more water.

Between 1933 and 1948 it is estimated that nearly 71 billion Imperial gallons of water were pumped, or 13·6 tons of water for every ton of ore hoisted. The highest recorded pumping rate, over a year, was 21,385,000 gal. per day. The present rate is around 20,000,000 gal. per day from an average depth of 1,100 ft.

There are three pumping stations on the 900-ft. and 930-ft. levels and two on the 1,400-ft. level, which relay water to the 900-ft. level. The pumps are moved around as conditions dictate. A new high-lift station, of 12,000-g.p.m. capacity, is now under construction on the 1,400-ft. level. As far as possible, all water is piped to the stations, thus ensuring an ample supply of clean water for industrial and domestic purposes and, at the same time, reducing the problem of de-sludging such large quantities of water. All pumps, with the exception of sludge pumps, are of the centrifugal type, in units of 1,000, 1,800, and 2,000 g.p.m.

Drainage costs, including the development of drainage cross-cuts, water-tight doors, and pumping, are about 1·48s. per ton of ore hoisted.

VENTILATION

A main exhaust ventilation system, with auxiliary blowing, is employed. Air enters the mine through No. 9 downcast shaft (22 ft. in. diameter, smoothlined) extending to the lowest level at 1,650 ft. The air is split off on the active main cross-cuts and travels along the main haulages to the various ladderways; it is then blown by auxiliary fans into the working places—such as development headings, stope benches, and undercuts. Scraper drives, grizzly chambers, and other places where secondary blasting is done, are all provided with direct connections to the return airways. As far as possible, the air is used once and then exhausted to the returns.

The return airways form a network of permanent openings in the foot-wall, which connect to disused haulages and main cross-cuts on the upper levels and thence to the main upcast shaft, No. 8, which is also 22 ft. in diameter and smooth-lined. The main fans on No. 8 shaft are of the axial-flow type and handle, at present, 500,000 c.f.m. at 3-in. water gauge. Other fans on surface bring the total air exhausted from the mine to 750,000 c.f.m.

All main tipping points, shaft-loading pockets, and the crusher station are equipped with flannel bag filtration units. Coke filters are used only when the air to be filtered may contain dense blasting fumes.

All rockdrills are of the approved front-head vented type.

The overlap, blowing, and exhaust system is used on all main haulage development and other big excavations—such as pump stations. Blowing only is used on sub-level development.

The fans used for auxiliary ventilation are air-driven units for the smallest sizes and electrically-driven units for sizes of about 6,000 c.f.m. at 4-in. water gauge or over. In the interests of maintenance, a heavy item where many small units are employed, the preference is for the centrifugal type of fan running at about 1,460 r.p.m. The pipes used for distributing the air from the fans are 22 in. and 14 in. in diameter. Lids are provided for covering the ends of pipes when a heading is idle. Rubber sleeves are used at pipe joints to minimize leaks.

The Devers konimeter is used for dust counting and the ignition-immersion-ignition technique is employed. A staff of seven, including a microscopist, is required for ventilation work.

During 1948, some 9,514 ft. of development work and 220 ft. of shaft sinking was done for ventilation purposes.

ORGANIZATION AND LABOUR

The Mining Department organization consists of a mine superintendent, with an underground manager and a mine engineer as sub-heads of the Department. Roughly, the responsibility is divided so that the underground manager controls all rock breaking, handling, and hoisting operations, the safety department, and so on, while the mine engineer deals with technical matters—planning and laying out of new work, draw control, ventilation, geology, surveying, sampling, and diamond drilling. Close liaison is maintained between the operating and technical sides and to this end meetings are arranged, as thought necessary, for discussion on new projects, current operations, safety, drainage, and other vital subjects.

The underground manager gives his orders to the mine captains and these, in turn, to the shift bosses and European 'gangers', who are, in effect, supervisors of gangs of from five to fifty Africans. The total strength is approximately as follows: European staff, 132; European 'daily paid', 340, and Africans, 3,850—a total of 4,322.

The production rate is, at present, around 260,000 tons of ore per month, with 30,000 tons of waste, and 13,000 ft. of development carried out.

The daily-paid Europeans have a union, run on the closed-shop principle, while the staff have an association.

Africans are recruited locally and are housed, with their families, in well-built compounds. The wearing of safety hats and other safety equipment—such as shin guards and hand pads—is compulsory. These items are supplied free by the company and boots are supplied at less than cost price.

In the Safety Division instruction is given to new European staff in the use of explosives and in safety practices, but the main effort is directed towards the instruction of the African and, particularly, the 'boss boy.' These boss boys have considerable responsibility, holding blasting licences and first aid certificates as well as having to possess a good knowledge of mining practice generally. Selection of suitable types for training as boss boys is a matter of some care.

First aid stations, manned by Africans, are maintained at strategic points throughout the mine and these Africans, along with the others, are given regular refresher courses on their duties.

Two Proto rescue teams are kept in training, with quarterly courses at the Central Rescue Station for the Copperbelt, monthly local practices, and weekly rescue-set tests.

MINING COSTS

For the financial years 1946–47 and 1947–48, the cost per short ton of ore crushed to *minus* 12 in. and delivered to the mill, has been 12·50s. This figure is almost double that obtained in 1940.

Fuel shortages have led to low production, and wood burning, to supplement coal, has led to high power costs. With the high and, to a large extent, fixed charges for such items as pumping, low production leads to high costs. During the period mentioned, much preparation work was done for block caving.

The operating cost per effective African shift has doubled in ten years and is now around 4·8s. per ton.

In November and December, 1948, with an average production rate of 258,574 short tons of ore per month, representing 6,200 long tons of blister copper from the smelter, the mining cost per short ton was 9·59s. The tonnage from block caving had not then reached its maximum, nor had the continuous retreat stoping progressed to the stage of routine mining. The figure of 9·59s. per ton, however, does give a clearer idea of present-day mining costs at a reasonable production rate than do the costs for the past two financial years.

TABLE I

ANALYSIS OF COSTS

	Year 1947/48 (Average: 175,269 short tons per month)		November and December 1948 (Average: 258,574 short tons per month)	
	Shill./Ton	*% of total*	*Shill./Ton*	*% of total*
European wages and salaries......	3·22	25·7	2·62	27·3
African wages......................	2·36	18·8	1·83	19·1
General supplies....................	2·51	20·0	2·09	21·8
Explosives..........................	1·65	13·1	1·30	13·5
Power (including compressed air)	2·19	17·5	1·18	12·3
Workshops and miscellaneous...	0·62	4·9	0·57	6·0
	12·55s.	100%	9·59s.	100%

CONCLUSION

There is not space within the limits of the present paper to describe many of the processes in detail. There are still many problems awaiting solution and opportunities for cost reduction. As labour becomes scarcer and more expensive, more mechanization will be required, always remembering that the African must operate the machine, which should, therefore, be robust and simple.

The use of conveyor belts will replace tramming to a considerable extent on this mine in the near future.

The crushing of all ore underground, by gyratory as well as the existing jaw crushers, to *minus* 4-in. size, will be adopted.

Tungsten carbide rockdrill bits, already applied experimentally, will probably play a large part in future rockdrilling.

Diamond blast-hole drilling at Mufulira has lagged behind the other mines of the Copperbelt, principally because of the hardness of the ore rocks. Tungsten carbide matrices for diamond crowns may provide the answer and research on these lines is taking place.

There is a healthy rivalry in technical matters between all the mines, and it is seldom long before a good idea developed at one property is being tried out at the others.

Acknowledgements

The writers wish to thank the various members of the Mining Department at Mufulira for their assistance in helping to compile data, and the management and directors for their encouragement and for permission to publish the paper.

[illegible]

J. P. Norrie and W. T. Pettijohn : *An Outline of Underground Operations at Mufulira Copper Mines, Ltd.*

Fig. 12.—Model illustrating sub-level stoping (continuous retreat) mining method.

Plate II.

J. P. Norrie and W. T. Pettijohn: *An Outline of Underground Operations at Mufulira Copper Mines, Ltd.*

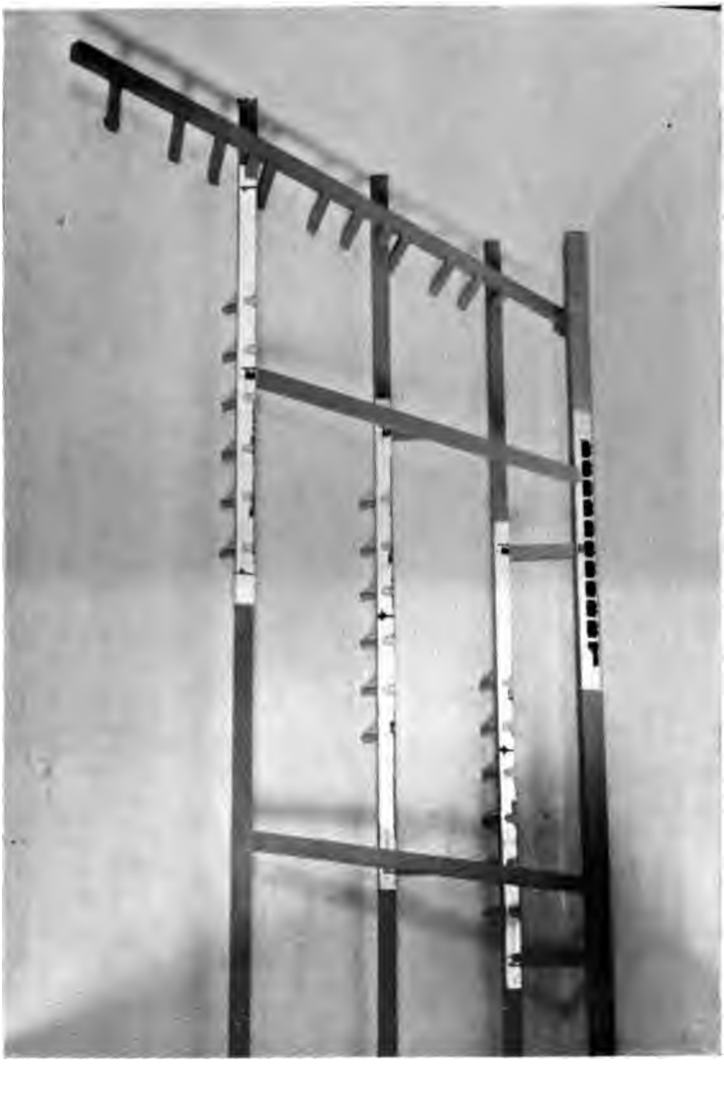

Fig. 13.—Model illustrating sub-level stoping (continuous retreat) mining method, showing foot-wall work.

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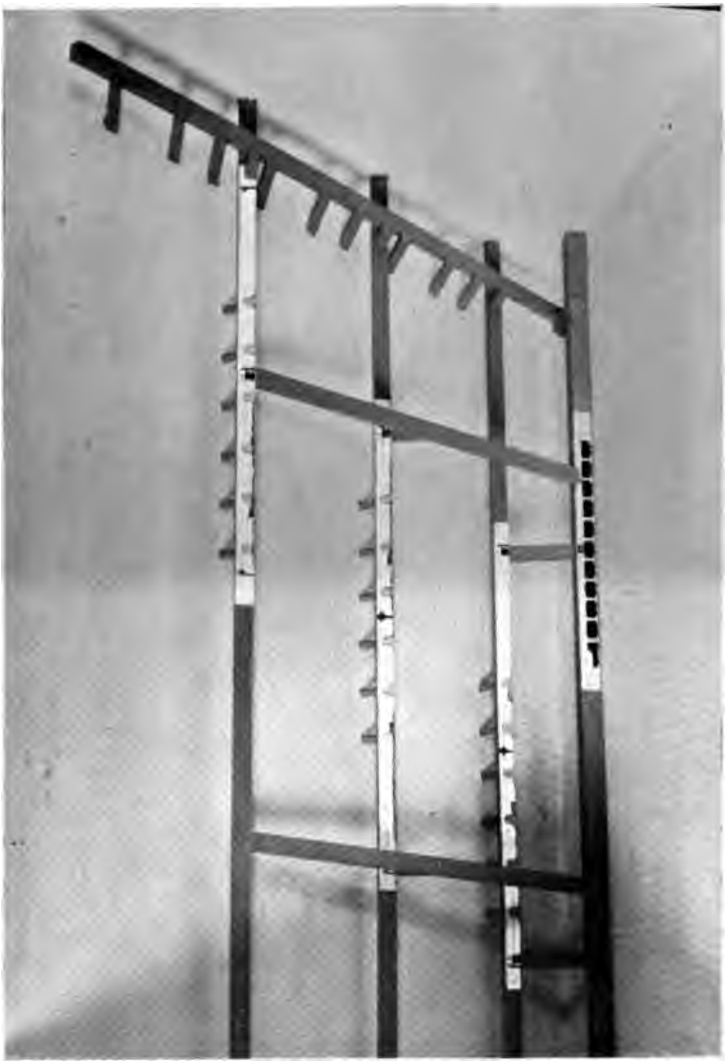


FIG. 13.—Model illustrating sub-level stoping (continuous retreat) mining method, showing foot-wall work.

Plate III.

J. P. NORRIE and W. T. PETTIJOHN: *An Outline of Underground Operations at Mufulira Copper Mines, Ltd.*

FIG. 14.—Model illustrating block caving mining sequence.

Plate IV.

J. P. NORRIE and W. T. PETTIJOHN: *An Outline of Underground Operations at Mufulira Copper Mines, Ltd.*

FIG. 15.—Drainage from bleeder holes, 1,150-ft. level.

Plate V.

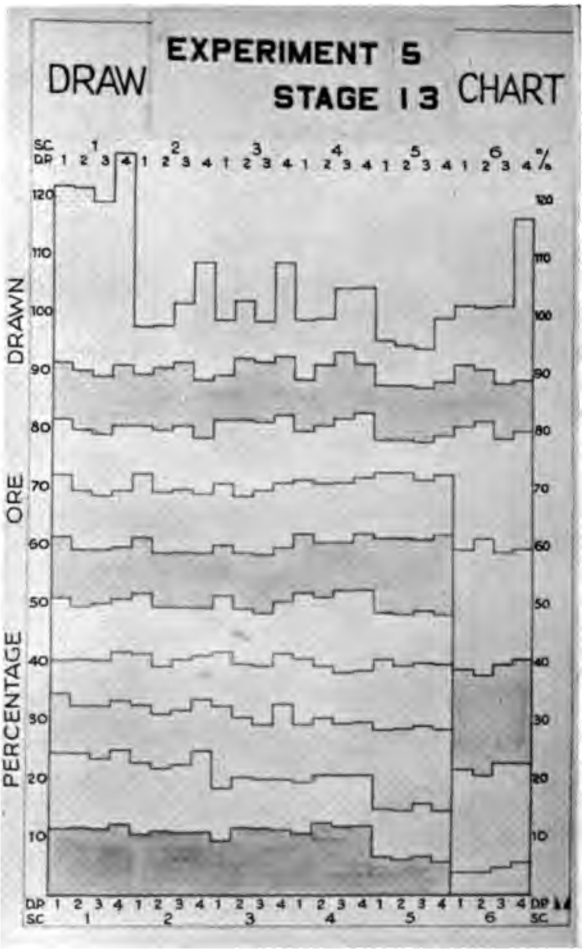


Plate I

The Use of the Wet Kata Thermometer on the Witwatersrand

P. H. KITTO

Report of discussion at October, 1949, *General Meeting* (*Chairman: Mr. W. A. C. Newman, President*). *Paper published in* Bulletin 514, *September,* 1949.

THE PRESIDENT said that there were two papers before the meeting, one by Mr. Kitto on *The use of the wet kata thermometer on the Witwatersrand*, which he would take first, and the second on *Investigations on the production of electrolytic cobalt from a copper-cobalt flotation concentrate*, by Mr. H. L. Talbot and Mr. H. N. Hepker.

The question of ventilation and dust in mines, especially those reaching to deep levels, was one which was extremely important and in which the Institution was becoming increasingly interested. At the Empire Congress that summer there was a session at which the interest on those matters was very great and it was hoped that further steps to consider and elucidate the serious problems arising would be stimulated.

Unfortunately the authors of the papers were unable to be present, but, in the case of Mr. Kitto's paper, one of his colleagues from the Transvaal Chamber of Mines, Mr. D. G. Beadle, would introduce it on his behalf.

MR. D. G. BEADLE said that he felt particularly honoured to be asked to attend the meeting. The members of his Department had in the past presented papers to the Institution and taken part in discussions, but he thought he was correct in saying that the present occasion was the first time that any member of their staff had actually been present when a paper by one of their colleagues had been presented. He hoped that in future it would occur that when one of the Chamber of Mines staff presented a paper he would be able to introduce it himself.

He confessed that the subject of heat was not his own speciality; he knew very little about it, but he had seen Mr. Kitto working with his kata thermometers and had seen some of his experiments. He could only give very general impressions of the use to which the kata thermometer was being put in the gold mines of the Witwatersrand. It was still regarded as a useful and important instrument and seemed to be becoming more so, because of the increasingly high temperatures and humidities under which mining was carried out to-day. In many parts of the world the kata seemed to have fallen from grace; it had been attacked by various people and there were all sorts of reasons put forward for not using it. In his paper Mr. Kitto had attempted to show that some at any rate of those criticisms were not justified and had described how he had removed some of the objections.

It was sometimes found that when two wet kata thermometers were used side by side in a working place of a mine they would give quite different results, or the ventilation officer would take a reading and report certain conditions and then somebody else would take a reading and report quite differently. To clear up some of those anomalies Mr. Kitto's first step was to standardize the instruments, and he had described some of that work in his paper. The speaker thought that the most important single factor was the introduction of a standard sleeve for the kata. It was found that the makers, particularly during and immediately after the war, were supplying all kinds of different material for covering the thermometer and that resulted in quite different results being obtained, so that the first thing which Mr. Kitto did was to introduce a standard sleeve, which was thinner than the one usually used. He thought it might be of interest to people in Britain to have a supply of those sleeves and he would leave some with the Secretary.

Mr. Kitto also described a method of calibrating wet kata thermometers. It was understood that the manufacturers in Britain calibrated the thermometers for use as a dry thermometer and it was found that that calibration was often incorrect when it was used as a wet kata thermometer. It was also calibrated by the manufacturers for a still atmosphere and as it was usually used in South Africa in moving air that also made a difference. All kata thermometers in use in Witwatersrand gold mines were sent to the Transvaal Chamber of Mines for re-checking; sometimes they were correct and sometimes a new factor had to be determined.

Those two standardization techniques had resulted in a great improvement in comparisons between different thermometers, and by laying down rather precise rules as to how the thermometers were to be used—e.g. how they were to be heated up, and other details of their use—there had been a great improvement in the consistency of the actual results obtained on the mines.

The merits of the kata would be argued again as a result of the paper; there were many objections to it, but, in spite of them, it had remained the standard instrument on the Witwatersrand for assessing comfort conditions and determining whether places were suitable for men to work in or not. It had been kept in that position because they had not yet satisfied themselves that any other method of measuring comfort was really better. Certainly no single instrument had been found to replace it. To use the 'corrected effective temperature scale' a man needed to take extra measuring instruments with him and any such procedure was not popular. The wet kata did indicate when working places were bad and the rather drastic reading of 4 was used as the unofficial limit of tolerance for active work. Observers in South Africa did not like such readings and tried to get a higher figure if they could, but if readings of less than 4 were obtained then something was invariably done to improve conditions. Men ought to be acclimatized before being sent to some working places and the kata was being used to determine whether a man should go straight to a working place or whether he should be acclimatized first.

The actual physical interpretation of kata readings had not worried them very much; they were aware that two quite different sets of physical conditions of temperature, humidity and air velocity could both give wet kata readings of, say, 6, but were also aware that the two conditions might be quite different from the comfort point of view. It was on such rounds that the use of the instrument could probably be criticized.

He was not sure what lines future work might take. His own feeling vas that there was a very great need for experiments on the actual vorking performance of Bantu underground labourers under carefully neasured physical conditions, to determine how much their efficiency ell off as such conditions got worse. At present there was no real informa-ion as to how working efficiency changed if the wet-bulb temperature vent up from, say, 85° F. to 90° F., or air velocity went down from, ay, 200 f.p.m. to 50 f.p.m.; and, in such cases, what percentage of the vork of which men were capable was lost. The South African Council for cientific and Industrial Research had recently appointed a physiologist, vho was to study the effect of environment on men's ability to work, and e hoped that as one of his items of research he would tackle the question f underground Bantu labourers. It was only after such precise measure-nents had been made that it would be possible to predict how far it was vorth while incurring the expense of, for example, installing an air onditioning plant.

MR. S. E. TAYLOR said that the importance of the effects of high tem-eratures on underground workers was brought into prominence at the ecent Empire Mining and Metallurgical Congress at Oxford and it was ppropriate that the kata thermometer should be the subject of the first aper of the new session, because it kept that important subject to the ore. The paper dealt with the measurement of the cooling power of the ir. The satisfactory and reliable measurement of the factors affecting the roblems of heat was essential to a study of the problem, and it was qually essential when it came to judging the effect of practical methods vhich might be taken to ensure satisfactory conditions.

The author said that a wet kata reading gave a very good idea of the tmospheric conditions and indicated whether acclimatization was ecessary; also that it could be used to compare the cooling powers of lifferent air conditions on a man. There was in the paper, however, a very rank acknowledgement of the discrepancies encountered, as well as of the imitations of the instrument and the unsatisfactory results obtained from ome accepted formulae. Indeed, the experiments described in the paper vere undertaken with the object of trying to improve matters. The paper ather gave the impression that the procedure for taking a wet kata reading vas complicated and that the results were unreliable unless very great are were taken. That impression did not quite seem to the speaker to do ustice to an instrument which, as they had just heard, was widely used n the mines in the Witwatersrand for obtaining routine measurements f conditions underground. It was refreshing to find systematic research eing applied to such a well-accepted instrument in order to try to improve he results obtained by its use.

Although he had nothing whatever against the kata thermometer, he ad not been converted to its use, and still preferred the less direct method f determining atmospheric conditions by measurement of wet-bulb and lry-bulb temperature and velocity. From those readings the humidity ould be calculated; it was, however, the wet-bulb temperature which was the most important.

It was interesting to notice that the author, when referring on p. 22 to the conditions likely to be found underground, mentioned a wet-bulb temperature of about 85° F. and a relative humidity of 95 per cent. Doubtless that terminology was for the benefit of those who were not counted in the wet-kata flock.

The author made an interesting reference on p. 33 to hot and dry conditions. He pointed out that the thin standard kata sleeve used in certain experiments started to dry out and consequently had a marked effect oupon the kata reading. He thought the author was probably correct in assuming that the heating effect on the human body would be just as marked if sweat were evaporating faster than it could be produced. That was only another way of saying that the cooling power of dry air lost its effect on the human body if the rate of evaporation exceeded the sweat rate. That might well happen if the velocity of the air were too high, and being in a strong hot dry air current when the skin was dry, for the reason that it could not produce sweat quickly enough, produced a very marked burning sensation.

He hoped that the author would be spurred on by criticisms of the paper to continue his efforts to perfect the use of the wet kata thermometer. It would be of great practical value if the wet kata reading were universally accepted as the measure of cooling power.

In connection with the problem of trying to correlate the cooling power of the air with human efficiency he pointed out that the author said on p. 41 that although his experiments indicated that the wet kata thermometer could be used as a means of comparing the cooling powers of different air conditions on a man, it did not follow that it could therefore be used as a direct measure of the efficiency of a man working in those conditions. That was a serious flaw in the practical application of the wet kata and it called for further research to try to find a remedy.

When considering matters of that sort the object aimed at had to be kept in mind; it was not merely a question of 'what were the limiting conditions for human endurance' or even 'when was acclimization necessary', but rather 'what were the best conditions for efficient work'. It was becoming clear that optimum air conditions, combined with an effective incentive, could be the means of increasing production and efficiency to an extent that not only was the cost of providing those conditions covered but an overall saving in cost was possible.

He was particularly interested in Mr. Beadle's references to an unofficial view that a wet kata reading of 4 represented the limit of conditions and it would be interesting if he could give a corresponding unofficial figure of the best conditions in which to work. Further, he was very pleased to hear that research was going to be undertaken on that question of efficiency when working in different conditions. He felt sure that research would yield valuable results and that it would be possible to state what were the best conditions for efficient work. He felt that the cost of providing those air-conditioning measures, whatever they might be, would be thoroughly justified and he looked forward to the time when they would no longer have to consider that a man had to be acclimatized to endure the worst conditions or that there were certain places where he must not work because the conditions were so bad—a time when they would be in a position to produce the best conditions which could be devised for efficient work.

He was particularly interested to see the author's reference on p. 41 to the scope for more work to be done on the correlation between the wet kata readings and the work output. That, along with the research mentioned by Mr. Beadle, would lead to further very interesting conclusions. He had no doubt that more would be heard of that research in due course.

THE SECRETARY then read communications received from Dr. Caplan and Dr. Bedford, as follows:

DR. ANTHONY CAPLAN*: There remains much to be learned concerning the most accurate and practical method of assessing comfort conditions at the working face. The subject is extremely important and in the future will become even more so. Every contribution which helps in advancing knowledge is therefore to be welcomed.

The wet kata thermometer has in the past decade gone out of favour in most mining fields. The great objection to its use in dry mines is that it is unduly influenced by high air velocities. However, on the Rand it has retained its original popularity and there can be no doubt it has provided much useful information.

The author refers to some of the deficiencies of the wet kata thermometer—such as the difficulties and necessity of careful calibration of each instrument and the discrepancies found in the readings of different observers. He also states that the wet kata is less efficient at wet-bulb temperatures above 90° F.—surely just the range where accurate information is needed. The author suggests how some of these deficiencies may be overcome.

He concludes that although his experiments indicate that the wet kata thermometer can be used as a means of comparing the cooling powers of different air conditions on a man, it does not follow that it can therefore be used as a *direct* measure of the efficiency of a man working in those conditions. To those of us interested in the subject the important question arises as to its value as an *indirect* measure of working efficiency. Working efficiency is obviously intimately connected with the degree of comfort or discomfort at the working place. When conditions are comfortable, other factors being equal, efficiency is unimpaired. When conditions become uncomfortable there is a progressive falling off in output and the ultimate loss of human efficiency is collapse. Does the improved but relatively complicated kata thermometer give more accurate information concerning the ability of men to work under varying conditions than the comparatively simple wet-bulb thermometer ? Is it not sufficient to know that the wet-bulb temperature in a certain section of a mine is rising

* Ministry of National Insurance Pneumoconiosis Medical Panel, Cardiff; formerly Senior Assistant Medical Officer, Kolar Gold Fields Hospital.

above 90° F. and is a sufficient indication for necessary action to be taken? Or is it possible to envisage on the Rand two working places with wet-bulb temperatures of 91° F., one with a wet kata index of about 4 and a low work output and the other with a wet kata index of about 7 with a relatively high output?

I agree with the author that there is scope for more work to be done on the comparison between kata readings and work output. I would add that the correlation should include other measurements of comfort conditions, with the object of comparing the relative merits of various indices.

DR. THOMAS BEDFORD*: I have read Mr. Kitto's paper on the wet kata thermometer with much interest.

My colleagues and I still make much use of the kata thermometer, but we use it only as a dry-bulb instrument (generally with the bulb silvered), and only as an anemometer. Mr. Kitto's experiments were undertaken to improve the accuracy of readings and to gain more knowledge of the significance of these readings in relation to the cooling power of the air on a man. The emphasis is thus on the value of the wet kata cooling power as an index of the thermal load on a man.

Mr. Kitto mentions the criticism that because of its small size the kata thermometer is more sensitive to air movement than is an object the size of a human body. Then he describes observations on the cooling rates of thermometers with bulbs of various sizes, and he concludes from his results that so far as size is concerned the kata thermometer can be compared, without very great error, with a human being. This conclusion rests on measurements made at various velocities, but at the temperatures prevailing in the laboratory which 'altered slightly from time to time'. Neglecting radiation, the cooling power of the kata thermometer depends on the air speed and on the wet-bulb temperature. In these observations one may infer that the wet-bulb temperature was approximately constant. Now since the convective and evaporative heat losses from these bulbs vary approximately as the square root of the air speed it was to be expected that with a constant wet-bulb temperature such results as Mr. Kitto plots in his Figs. 8 and 9 would be obtained. Likewise, if he had kept the air speed constant and varied the wet-bulb temperature one would have expected similar agreement between the different thermometers. It is when one varies both temperature and velocity in the same series of experiments that the effects of the size of the object show clearly, and unfortunately Mr. Kitto does not give data of such experiments.

If the cooling power of the kata thermometer is to be accepted as a reliable index of environmental warmth it should correlate well with human subjective sensations of warmth or with some acceptable measure of physiological stress. By these criteria the kata thermometer is not satisfactory.

In relatively cool surroundings (say 50°–75° F.) when sweating is not a factor, the dry kata cooling power might be expected to be a valuable index of warmth. Yet, in an extensive study of the thermal comfort of

* *Medical Research Council Industrial Health Research Board, London.*

factory workers, I found that as such an index it was distinctly less reliable than even the simple dry-bulb temperature. A dry kata cooling power of 8 millicalories per sq. cm. per sec. is given by a room temperature of 67·7° F. and an air speed of 100 f.p.m., and also by a temperature of 47·3° F. when the air is moving at only 10 f.p.m. Wearing normal indoor clothing most people would not be very cool in the first condition, but they would be badly chilled by the second.

One can compute from recent physiological data evidence which shows the unreliability of the wet kata thermometer at high temperatures, During the war extensive physiological studies of the effects of hot and humid environments on man were carried out on behalf of the Royal

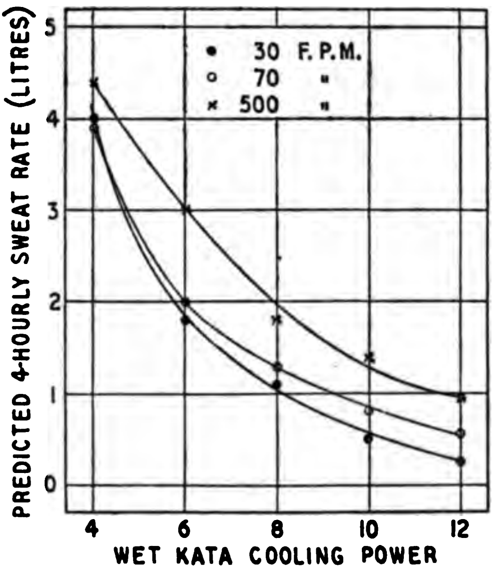


Fig. 11.—Predicted rates of sweating of men wearing shorts and working (metabolic rate 100 k.cal. per sq. m. per hr.) in relation to wet kata cooling power.

Naval Personnel Research Committee by Dr. B. McArdle and his colleagues at the National Hospital, Queen Square. As the measure of the physiological stress imposed by the environment, the amount of sweat lost during four hours was used. The experimental subjects were well acclimatized to heat.

Using Mr. Kitto's formula, I have calculated the wet-bulb temperatures which, with air velocities of 30, 70, and 500 f.p.m., would produce various wet kata cooling powers. Then, from the data of McArdle and his colleagues I have computed the predicted four-hourly sweat rates for persons wearing only shorts, and with a metabolic rate of 100 kilocalories per sq. metre per hr. In these calculations I have assumed the dry-bulb temperature to be 2° F. higher than the wet-bulb. If the wet kata cooling power were a good index of warmth one would expect that the sweat rates for a given cooling power would be the same, whatever the air speeds. Fig. 11 shows that that is not so. The higher the velocity the higher was the sweat rate—clear evidence that the kata thermometer makes too much allowance for the cooling effect of wind.

One of Dr. McArdle's colleagues, Dr. J. S. Weiner, has expressed predicted sweat rates in terms of what he has called total stress temperature. This takes into account rate of energy production and clothing, and the stress temperature is that temperature of still and saturated air in which a resting man clad in shorts would produce sweat at the same rate as a man under the conditions specified. In the left-hand half of Fig. 12 the wet kata cooling powers used in my calculations have been plotted against the corresponding stress temperatures. With a given cooling power the stress temperature is distinctly higher at the higher velocities, and at the higher temperatures these differences are significant. At the same wet kata cooling power the sweat rate may vary roughly as 2 to 1.

The well-known American scale of effective temperature has its drawbacks, the most important being that at very high temperatures excessive

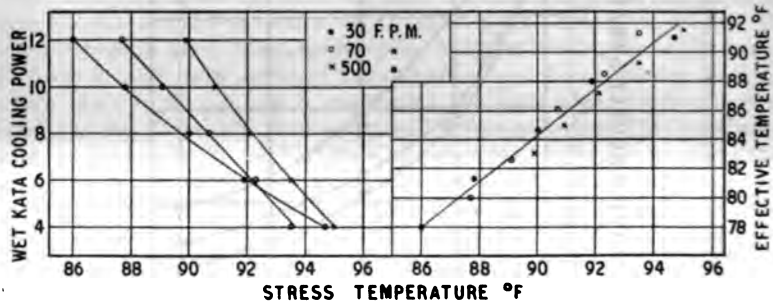


Fig. 12.—Wet kata cooling power and effective temperature compared with stress temperature.

allowance is made for dry-bulb temperature. Since in the mines of the Witwatersrand the air is generally very moist this defect of effective temperature is not likely to lead to difficulty if the scale is employed for evaluating conditions in those mines. I have calculated the effective temperatures represented by the various cooling powers, air speeds and dry-bulb temperatures used in computing sweat rates, and these temperatures are plotted against the corresponding stress temperatures in the right-hand half of Fig. 12. There is a close correlation between the two scales, and it is evident that over the range of conditions considered, in which the air was assumed to be humid, effective temperature is a good index of physiological stress, and much superior to wet kata cooling power.

There is one less important point to which I should refer. Mr. Kitto cites experiments from which conclusions are drawn relating to the effects of radiant heat. The description of the apparatus used and of the conditions of the experiments are so vague that no conclusions can be drawn by the reader. The duct was of square section and for a short distance it was heated to 125°—130° F. We do not know what solid angle the heated surface subtended at the thermometer bulb, nor the temperature of the unheated surface, nor are we given either the mean radiation flux or the mean radiant temperature. Nevertheless, one may conclude that the mean radiant temperature was substantially increased when the duct surface was heated. It is likely that the effect of that increased radiation on a man would be greater than its influence on the rate of cooling of a kata thermometer, for, owing to the difference in size, the ratio of radiation loss to total heat loss is greater with a man than with a kata thermometer.

MR. C. I. ROBINSON said that he was moved by what had been said to offer something of his own experience in trying to use the wet kata. The work was done over 20 years ago in a Malayan mine, where it was recognized that the conditions in the bottom of the mine were very bad and it was his task to try to get things remedied. The temperatures were well over 90° F. and the relative humidity of the air was little short of 100 per cent. His experience with the kata was that it took a very long time to get a reading, so bad were the conditions. He was also using the whirling psychrometer, and the wet- and dry-bulb thermometers. He found that they enabled him to get readings and compare conditions in various parts of the mine in a way he could not do with the kata. He was not saying anything about the accuracy of the readings or the deductions therefrom but when one was dealing with the limiting conditions of human work underground the difficulty he found himself up against was the time taken in getting a reading.

The PRESIDENT proposed a very hearty vote of thanks to the author for his paper and to Mr. Beadle for introducing it.

Investigations on the Production of Electrolytic Cobalt from a Copper-Cobalt Flotation Concentrate

H. L. TALBOT, MEMBER, and H. N. HEPKER

Report of discussion at October, 1949, *General Meeting* (*Chairman: Mr. W. A. C. Newman, President*). *Paper published in* Bulletin 514, *September,* 1949.

THE PRESIDENT said that the paper could be considered to be supplementary to that on cobalt refining given by Mr. P. S. Bryant at the Symposium on the Refining of Non-Ferrous Metals earlier in the year. He would ask Dr. Smith to be good enough to introduce it and initiate the discussion.

DR. S. W. SMITH then introduced the paper on behalf of the authors, outlining the work described.

MR. D. G. ARMSTRONG said that he would like to make a few comments on the figures given for the furnace tests for the matte leach process. On p. 7, in No. 11 campaign, there was apparently a misprint in the figures for the furnace products : the weight of the matte was given as 29,949 lb., but he thought it should be 21,949 lb. or something similar, because it did not work out correctly otherwise.

He found that under the heading of 'Percentage Recoveries' the recovery of cobalt in the matte was given as a percentage of the cobalt in the matte-plus-slag and the 'unaccounted for' figure as a percentage of the cobalt in the furnace charge, yet the two together were given under the same heading of 'percentage recoveries'. He thought that was definitely wrong, because although the two figures were given under the same heading they were calculated from different sources. It seemed to be a method of avoiding the unsatisfactory recovery of more than 100 per cent. He agreed that it was very difficult to make those figures balance accurately, but it seemed to be the principle which was wrong in calculating the percentage recovery of cobalt from the products instead of from the charge.

In three of the examples given there was a gain of cobalt during the test, which might be due to errors in weighing, assaying, or sampling, or to a pick-up of cobalt from the previous test. If there were a loss of cobalt during each test, as in the last one, it might be assumed that that was due to dusting or volatilization or something similar and the 'percentage recovery' would have to take such losses into account.

He wondered if there was any significance in the fact that as the percentage of converter slag in the charge was gradually increased in the four tests mentioned the 'unaccounted for' figure gradually decreased. The 'unaccounted for' figure in the first one was +8·8, and in the next it was +1·6, in the third it was down to +0·4 and in the fourth it was on the negative side, — 8·9. He wondered if the same thing happened in the other campaigns which were not mentioned.

MR. J. JACOBI said that he had read the paper with great interest and if any of his comments appeared to be critical the criticism was on detail rather than on the work underlying the paper, which had been very painstaking.

Apparently 31·7 per cent of the total cobalt in the mill feed could be recovered as a low-grade concentrate and by the present method only 14 per cent of the cobalt in the mill feed was found in the cobalt alloy, giving a loss somewhere in the present smelting operation of 56 per cent. Presumably that cobalt was found in the slag from the final electric furnace operation, or largely found there, and it would have been interesting to learn whether it had been considered that some of that slag could be used in making up the charge for the reducing matte leach. Almost an equal amount of cobalt was contained in the copper concentrate as was in the cobalt concentrate although, of course, in a much lower concentration. Most of the cobalt in the copper concentrate presumably found its way into the converter slag, and some of that converter slag was used in the special matte smelting method discussed on pp. 6 and 7. Quite an appreciable amount of cobalt in those mattes was derived from the converter slag and it was not easy to see why the authors were so anxious to establish the minimum required of the converter slag. It might have been equally to the point to determine the maximum possible addition of that slag.

The authors explained that the iron/cobalt ratio should not be increased because of subsequent separation difficulties, but in the three campaigns quoted, 10, 11 and 12, there was hardly any significant change in the iron/cobalt ratio. That ratio varied from 6·9 to 1 to 7·6 to 1, which was not a significant change. Oddly enough the chemical composition of the mattes in those three campaigns was identical yet they gave quite different leaching results. For the explanation of that one would presumably have to look at the microscopic evidence, but from the analyses quoted it did not become evident why those three mattes should behave so differently.

It might have saved some misunderstanding if, on the detailed material balance sheet, the authors had stated which of the materials had been weighed in and in which cases the weight was calculated. That was explained, rather vaguely, on the top of p. 8. The speaker was not quite sure what the authors meant but he understood them to mean that the furnace charge had been weighed in, and the weight of the slag had been computed from the silica assay, and from that weight and the copper assay of slag and matte, the matte weight had been computed. That was quite reasonable because, presumably, the matte weight and slag weight were difficult to establish directly even on such relatively small campaigns. By adding to the figures in the paper the note 'actual weight' or 'calculated weight' perhaps some confusion would have been avoided.

On the question of acid-leaching of those mattes, the authors found that wet ground matte, probably due to oxidation, did not leach so well and when finally the reaction started it was extremely violent. They put that down to the sulphide nucleus being reached by the acid. A simple *perusal* of the thermo-chemical figures would show that iron sulphide leaching gave rise only to a very small temperature increase, something of the order of 7° C. based on the acid concentration used, but the leaching of metallic iron or cobalt gave rise to much more violent reaction, something of the order of 32° C. increase, and, setting that against a starting temperature of 80° C. and a delayed but sudden action, that violent and explosive condition was easily explained.

The thermo-chemical data in themselves did not give any details as to the rate of reaction, but he had often noticed that with sulphuric acid leaching of oxides the reaction might not proceed at all at a certain temperature and yet at a temperature 5° higher it suddenly became violent. At 80° C. one was near boiling point and the heat of reaction had to be taken into consideration.

That cast a certain amount of doubt on the figures given at the bottom of p. 11, where the temperature was kept constant. The authors did not say how it was done or whether the heat loss, either fortuitous or intentional, was equal to the varying rate of reaction. He wondered whether the authors really meant constant initial temperature or constant temperature throughout.* Similarly, he did not think it was quite fair to say on p. 11 (*a*) that the acid concentration was varied alone; actually the authors had varied two variables at the same time, the acid concentration and the stoichiometric excess. Under the last heading when they started with an acid concentration of 110 and got a leaching extraction of only 64 per cent there was hardly any free acid left.

From those results one would have thought that that was a job which called for two-stage leaching, in order to avoid the large amount of free acid left in the leaching process which had to be neutralized with ground limestone. He found that limestone and milk of lime were used liberally at every stage. That was not to be recommended because the filtration and washing difficulties would be quite enormous. Especially was that the case in the regeneration of the spent electrolyte as the authors stated that during electrolysis only 1 lb. of cobalt per 100 gal. was extracted, which meant a colossal circulating load of solution. They did not say what electrolyte was used, but presumably it was a straight sulphate solution. He did not know whether they had tried to 'buffer' it with ammonium sulphate to increase the stripping ratio, but he would have thought that if it were necessary to have such a large circulating load of electrolyte it would be better to add a pure cobalt oxide or cobalt carbonate at that stage to make up the electrolyte and kill the acid which had been generated, thus obviating the laborious filtration of 100 gal. of pure electrolyte for every pound of cobalt deposited.

Presumably the use of lime was governed by local conditions; it was not always easy in Britain to appreciate the local conditions in Northern Rhodesia but he would have thought that at the final purification stage a precipitant such as soda ash would be called for. The resulting cobalt carbonate would form an ideal neutralizing agent for that spent electrolyte. He felt that the addition of limestone at every stage would lead to

* In such leaching tests, the maximum temperature was usually more significant.

a tremendous expenditure on filtering plant if that process were carried out on a large scale.

Lastly, he referred to the principal uses of cobalt. Quite a large amount was used in the form of alloys but a great proportion was used in the form of cobalt salts, such as sulphate, naphthenate, etc. The starting material for those salts was usually cobalt sulphate and he wondered if the direct recovery of that salt had been considered.

Mr. G. L. Evans, confining his remarks to the first portion of the paper—i.e. the production of leachable mattes—said that in the section devoted to the microscopic examination of cobalt mattes the authors stated that copper sulphide occurred in the mattes in the form of CuS. Such a view was at variance with the usually accepted ideas of matte constitution, and it would be appreciated if the authors were to give information regarding the evidence which led them to such a conclusion.

When discussing matte constitution, the authors made no mention of the presence of magnetite. It was well known, and generally accepted, that magnetite occurred in appreciable quantities in many commercial mattes (*vide* Hawley, and Drummond) and there was some evidence that the magnetite content tended to be greater in the case of lower-grade mattes. In the reverberatory furnace campaigns described by the authors, comparatively large proportions of converter slag were used (in No. 12 campaign, nearly 20 per cent) and it seemed justifiable to suppose that the conditions were favourable to the production of matte carrying appreciable magnetite, in spite of the fact that reducing conditions were maintained during the smelting operations.

In describing the microstructure of cobalt mattes, the authors mentioned the presence of metallic iron in unusually large amount (Fe : FeS), as well as metallic copper. Published information regarding the liquidus surface of the ternary system copper-iron-sulphur was scanty, but it would appear from the work of Reuleaux that *either* iron-rich *or* copper-rich solid solution might crystallize as a primary constituent. It would be interesting, therefore, to have the opinions of the authors concerning the mode of formation of the metallic phases.

Professor W. R. Jones said he was interested, from the mineralogical view-point, in the authors' reference to a 'pinkish-coloured copper sulphide' which they believed was present as CuS. That was the composition of the mineral covellite, which was blue in hand specimen and in reflected light under the microscope. He would suggest to the authors that they polish the surface of the material and make use of polarized light for its examination. Covellite was perhaps the easiest of all opaque minerals to recognize under the ore microscope. It was distinctly pleochroic (pale blue to dark blue) and when the nicol prisms in the vertical illuminator were in crossed position the polarization colours were highly spectacular. In both those respects covellite was unique.

Professor C. W. Dannatt said that a paper of that type on experimental work was most welcome and he felt that the Institution was indebted, not only to the authors, but also to the companies for giving permission for its publication. He trusted that further information would be made available as the work progressed.

He agreed with Dr. Smith that one should not cavil at the figures of a balance sheet for a small-scale, batch-smelting operation. Accuracy was impossible with so many unknowns, but the difficulties gave to such a problem a particular fascination. In his own experience, it was possible to arrive at a reasonable balance in the case of a copper smelt, provided due allowance was made for entrainment of matte in the slag and of slag in the matte. The silica content of the matte was not given in the paper and there had been no attempt at such a correction. There was the further complication, to which the authors made reference on p. 8, that some metal was reduced and absorbed into the hearth. With batch working one would expect that metal to seal itself so that no serious loss should occur in the later campaigns, but the presence of such metallics would make accurate sampling so difficult that the analyses would be unreliable, quite apart from errors due to entrainment. Some closely controlled, small-scale crucible fusions should give useful information that would enable suitable corrections to be incorporated in the main calculations. Such adjustments should prove more satisfactory than estimations based upon an over-simplification.

The method adopted by the authors for calculating the weight of slag made from the silica charged ought to give reasonable accuracy, but their calculation of matte weight would suffer from an accumulation of errors. The theoretical recovery of metal could be calculated from the analyses and the estimated slag weight, and there seemed little advantage in determining an 'unaccounted-for' figure from so unreliable a quantity as the estimated matte weight. There would be as much justification for obtaining that weight from one of the other constituents and so finding an 'unaccounted for' copper figure, unless it was assumed that no copper was reduced to the metallic state.

No reference was made to the presence of magnetite, either in the charge or the products. Converter slags always carried magnetite and that should react with the sulphides at the smelting temperature. Reduction of some metals, particularly cobalt, to the metallic state would be expected and the authors had confirmed that in their microscopical examination. Moreover, if magnetite acted in that way, an increase of converter slag in the charge should produce a higher proportion of reduced metal and, in accordance with the authors' statement on p. 9, improve the leaching properties of the matte. The leach-extraction figures on pp. 6 and 7 supported that assumption.

He found it difficult to accept the authors' statement that copper was present in the matte as cupric sulphide. That compound was unstable at elevated temperatures and it seemed most unlikely that it would be formed at smelting temperatures and under reducing conditions when some cobalt, iron and copper were being brought to the metallic state. He did not know what the authors meant by their statement that 'microscopic evaluation . . . indicates that copper sulphide occurs . . . as CuS', and he asked if they had any more satisfactory supporting evidence. Incidentally, if it were assumed that were true, the analyses on pp. 6 and 7 would show that the mattes contained about 20–25 per cent of uncombined metal—a surprisingly large quantity.

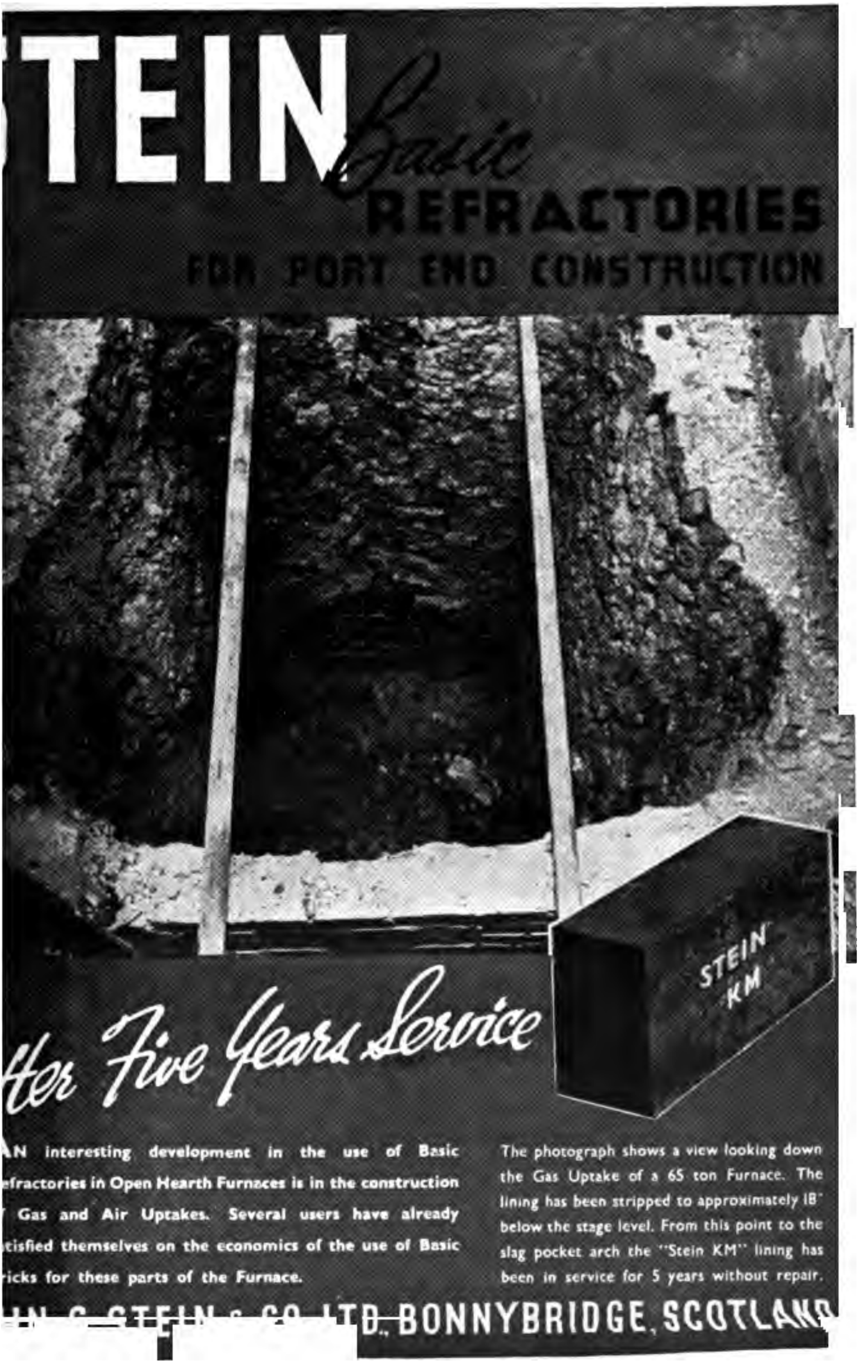
MR. G. L. EVANS, referring to p. 4 of the paper, where the authors stated: 'It was already known that the addition of cold converter slag and additional coal to the reverberatory charge had a marked effect on the leaching characteristics of the resulting mattes', asked if molten converter slag additions produced similar results, and if not, had the authors any theories to account for the difference in behaviour.

The PRESIDENT proposed a very hearty vote of thanks to the authors for their paper, to Dr. Smith for his summary in opening the discussion, and to all those who had commented on the paper.

CONTRIBUTED REMARKS

MR. J. H. CLUTTON: The treatment of cobaltiferous matte by formation of cobalt sulphate at the expense of copper sulphate and subsequent solution is similar to the Ziervogel silver process, in which a maximum silver sulphate content corresponds to a minimum of copper sulphate. Testing of such conditions used to be done by a lad, trained in the laboratory, who took a sample from the roaster, rubbed it fine, weighed a portion, washed it with hot water, added ammonia to the filtrate, adjusted the volume and compared it with standard solutions by colour. The point of maximum silver sulphate was thus quickly and easily determined and the calciner men (stimulated by a bonus) were very keen on the testing; it was interesting to see the skill with which they hit the desired point.

I think a similar system of testing could be applied for the cobalt sulphate.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