No. 476. JANUARY, 1946.

Bulletin
of
The Institution
of
Mining & Metallurgy

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Published at two-monthly intervals by
THE INSTITUTION OF MINING AND METALLURGY,
SALISBURY HOUSE, FINSBURY CIRCUS, LONDON, E.C.2.
TELEGRAMS: MINANMET, PHONE, LONDON. TELEPHONE No.: CLERKENWELL 4884

[redacted]

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The Institution of Mining and Metallurgy

(*Founded* 1892—*Incorporated by Royal Charter* 1915.)

Bulletin No. 476.

JANUARY 10TH, 1946.

LIST OF CONTENTS.

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LIST OF THE OFFICERS OF
The Institution of Mining and Metallurgy,
1945-1946.

Acting President
G. F. Laycock.

Vice-Presidents:

H. R. Holmes.
E. G. Lawford.
G. F. Laycock.
E. A. Loring.
W. A. C. Newman.
Andrew Pearson.

Members of Council:

J. C. Allan.
G. Keith Allen.
L. H. Bartlett (*India*).
H. H. W. Boyes (*West Africa*).
A. L. Butler.
G. A. Davenport (*Rhodesia*).
Tom Eastwood.
Sir Lewis L. Fermor.
W. H. C. Geikie.
Donald Gill.
Vernon Harbord.
Arthur Hibbert.
C. E. Jobling.
W. R. Jones.
Julius Kruttschnitt (*Australia*).
J. C. Nicholls (*Canada*).
R. J. Parker (*U.S.A.*).
C. E. Parsons.
J. H. Rich (*Malaya*).
J. A. S. Ritson.
R. H. Skelton.
A. J. G. Smout.
R. S. G. Stokes.
S. E. Taylor.
S. J. Truscott.
G. W. M. Eaton Turner.
W. G. Wagner.
A. J. Walton (*South Africa*).

Ex-officio Members of Council: (Past-Presidents)

Sir Thomas Holland.
William Cullen.
S. W. Smith.
Carl Davis.
Robert Annan.
E. D. McDermott.
Thomas Pryor.
J. Allen Howe.

Hon. Treasurer:
James G. Lawn.

Hon. Technical Editors:
E. G. Lawford (*Mining*). S. W. Smith (*Metallurgical*).
W. R. Jones (*Geological*).

Solicitor:
R. T. Outen.
(Messrs. Ashurst, Morris, Crisp & Co.)

Bankers:
National Provincial Bank, Limited, 50, Cornhill, E.C. 3.

Auditors:
Messrs. Woodthorpe, Bevan & Co., C.A., Leadenhall Buildings, E.C. 3.

Secretary and Editor:
W. J. Felton, Salisbury House, Finsbury Circus, London, E.C. 2.

LIST OF STANDING COMMITTEES.

General Purposes Committee.
Publications and Library Committee.
Finance Committee.
Applications Committee.
Awards Committee.
Appointments (Information) Committee.
Committee on Education.
I.M.M. and I.M.E. Joint Advisory Committee.
Committee on Mining in Great Britain.

The Council deeply regret to announce the death of the President, Lt.-Col. EDGAR PAM, O.B.E., A.R.S.M., on 20th December, 1945, after a long illness.

A Memorial Service will be held at St. Mark's Church, North Audley Street, London, W. 1, on Friday, 11th January, 1946, at 11.30 a.m.

NOTICE OF GENERAL MEETING.

The FOURTH ORDINARY GENERAL MEETING of the Fifty-fifth 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, JANUARY 17th, 1946, at **5.30 o'clock p.m.**

The following Papers will be submitted for discussion :

Notes on the Development of the Blyvooruitzicht Gold Mining Co., Ltd., South Africa.
By A. SAVILE DAVIS, *Member.*
(*Published with November*, 1945, Bulletin.)

The San Telmo Ore-body, Spain.
By J. C. ALLAN, *Member.*
(*Copy attached hereto.*)

A Projected Central Mill for the Durham Fluorspar Industry.
By ANDREW PEARSON, *Member.*
(*Copy attached hereto.*)

Light refreshments will be provided at 5 p.m. for members and friends 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 Council desire to remind members that in addition to the more comprehensive types of papers for discussion at General Meetings, they welcome for publication in the *Bulletin* short notes recording data or describing technical experience, which may be of general interest and value. Such notes are governed by the same rules in regard to acceptance as ordinary papers, but would be open for discussion by correspondence.

FIFTY-FIFTH SESSION : 1945–1946.
DATES OF SUBSEQUENT MEETINGS.

The following dates have been provisionally fixed for the General Meetings of the Institution during the Session 1945–46 :

Thursday, March 21st, 1946. Thursday, May 16th, 1946.

ELECTION OF MEMBER OF COUNCIL.

At a Meeting of the Council held on November 8th, 1945, Mr. Arthur John Griffiths Smout, *J.P.*, *Member*, was elected a Member of Council for the remainder of the Session 1945–46 under the provisions of Section IV, Clause 7, of the By-Laws, to fill the vacancy caused by the resignation of Mr. A. Broughton Edge.

ELECTION OF PRESIDENT, HON. TREASURER, AND VICE-PRESIDENTS FOR SESSION 1946-47.

The Council of the Institution have pleasure in announcing the election of the following Officers for the Session 1946–47:

President: Mr. G. F. Laycock, *M.C.*

Hon. Treasurer: Mr. Robert Annan.

Vice-Presidents: Messrs. G. Keith Allen, H. R. Holmes, E. G. Lawford, E. A. Loring, Andrew Pearson, and S. E. Taylor, *D.S.C.*

Mr. Laycock, the President-Elect, was educated at St. John's School, Leatherhead, and received his technical training at mines in Cornwall and at the Redruth School of Mines. From 1907 until 1910 he was assistant to the late Col. R. C. Feilding, during which period he visited Newfoundland, Siberia and Europe. From 1910 until 1915 he was in charge of mining operations in Turkey and Siberia, for Messrs. Hooper, Speak & Feilding, the Cape Copper Co., Ltd., and the Atbasar Copper Co., Ltd., and in the following three years saw active service with the Royal Engineers (Tunnelling Companies), being awarded the Military Cross. On demobilisation, Mr. Laycock spent a further year in Siberia, Korea and Japan, but in 1920 joined the Anglo-Newfoundland Development Co., Ltd., and since then has been closely associated with mining and engineering work in Newfoundland. Mr. Laycock is Vice-President of Terra Nova Properties, Ltd., a subsidiary of Anglo-Newfoundland Development Co., Ltd., which owns all the mining interests of the latter company in Newfoundland (Buchans mine, etc.) and other parts of the world (Australia, Canada, Arabia, Nicaragua, etc.). He is also a Director of the Mining Trust, Ltd.

RETURN OF PROFESSIONAL WORKERS AND STAFFS TO FAR EASTERN TERRITORIES.

The Colonial Office has had under consideration the arrangements to be made for the return to Malaya, Hong Kong, North Borneo and Sarawak of professional men and women and the staffs of firms (other than ex-enemy) previously established in those countries. It is obviously desirable that these men and women should return as soon as practicable to assist in the full restoration of the economic life of the territories.

Owing to shortage of transport, and as the admission of civilians into those territories is at present subject to the consent of the military authorities, it is necessary to devise a scheme for ensuring that the entry of these staffs and members of the various professions is regulated according to the interests of the territory as a whole in the first place, and, secondly, so as to provide, as far as possible, equality of opportunity.

The Colonial Office has, therefore, decided to register and classify persons who have reason to proceed to any of the above territories from the United

RETURN OF PROFESSIONAL WORKERS, ETC.—*continued.*

Kingdom, including women and men other than those referred to in paragraph 1. Such persons are invited to apply *in writing* to the Colonial Office (Passages Department), Palace Chambers, Bridge Street, S.W. 1, for the necessary forms of application; no assurance can, however, be given at this stage as to when passages will be available.

This invitation does not apply to the staff of Rubber Companies who are members of the Malayan Rubber Estate Owners' Company and the Borneo Rubber Estate Owners' Company, or to the staff of Tin Companies who are represented on the Rehabilitation Committee of the Malayan Chamber of Mines. In these cases the Colonial Office will continue to communicate with the bodies mentioned in regard to passages for the staff of the Companies which they represent.

Similar arrangements are being made in South Africa, Australia, New Zealand, India and Ceylon for the registration of persons temporarily resident in those territories who wish to proceed to the territories referred to in paragraph 1.

The wives and families of members of His Majesty's Forces who are serving in Malaya or any of the other territories mentioned above should not apply to the Colonial Office. Information regarding the rejoining of their husbands overseas will be issued by the Service departments concerned at the appropriate time.

LIBRARY SERVICE.

The Library has now been brought back to London, and applications for books should be addressed to the Librarian, I.M.M. and I.M.E. Joint Library, 424, Salisbury House, London, E.C.2. Books at present on loan should of course be returned to this address. Members who are unable to visit the Library and borrow books in person may still borrow them by post. It is regretted that periodicals cannot be lent.

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 November 8th, 1945:—

To Membership—

Farrington, John Leonard (*Royal Engineers*).
Park, James Williamson (*Tarkwa, Gold Coast*).
Tasker, Richard Beaumont (*Heidelberg, Transvaal*).

To Associateship—

Cowlin, William Ronald (*Royal Engineers*).
Job, Arthur Leslie (*Camborne, Cornwall*).
Pratt, Norman (*Thames, New Zealand*).
Tregay, Brian Antony (*Royal Navy*).
Wright, John Richard (*South African Engineer Corps*).

CANDIDATES FOR ADMISSION—*continued.*

The following have applied for admission into the Institution since November 8th, 1945 :—

To Membership—

Naylor, Theodore Rufus (*Royal Air Force*).
Pokorny, Ernest Adalbert (*London*).
Talbot, Harold Leroy (*Kitwe, Northern Rhodesia*).

To Associateship—

Allen, Lawrence Wilcock (*Chingola, Northern Rhodesia*).
Bolton, Cedric Michael Grey (*Maidstone, Kent*).
Christie, John (*Kitwe, Northern Rhodesia*).
Foster, Douglas Frank (*Doncaster, Yorkshire*).
Keough, Martin (*Que Que, Southern Rhodesia*).
Lamba, Bhag Singh (*Khewra, India*).

To Studentship—

Bridger, Denis (*Camborne, Cornwall*).
Dogan, Mustafa Zelsi (*London*).
Dyson, Peter Gerald (*Peterborough, Northamptonshire*).
Hallé, Charles Edwin Henton (*Manchester, Lancashire*).
Hammett, Peter Henry John (*Falmouth, Cornwall*).
Moon, William Raymond Collins (*Truro, Cornwall*).
Royle, Paul Gordon (*London*).
Wilson, Walter Joakim (*Camborne, Cornwall*).

TRANSFERS AND ELECTIONS.

The following have been transferred (subject to confirmation in accordance with the conditions of the By-Laws) since November 8th, 1945 :—

To Membership—

Davidson, William Sinclair (*Accra, Gold Coast*).
Falcon, Michael (*Johannesburg, Transvaal*).
Norman, Clive Maxwell (*Thames Ditton, Surrey*).
Pryor, Edmund James (*Wembley, Middlesex*).
Rogers, Douglas John (*London*).
Walker, Edgar Aruba (*Tirodi, India*).
West, Jesse (*Barakin Ladi, Northern Nigeria*).
Williams, Arthur Howell (*Arequipa, Peru*).

To Associateship—

Kirby, Noel Spencer (*Mt. Morgan, Queensland, Australia*).
Paterson, Oliver Douglass (*London*).
Ramaswamy, Coonigal Ranganna (*Marikuppam, South India*).
Thomson-Jacob, John (*Oorgaum, South India*).

The following have been elected (subject to confirmation in accordance with the conditions of the By-Laws) since November 8th, 1945 :—

To Membership—

Phaup, Albert Edward (*Wigan, Lancashire*).

TRANSFERS AND ELECTIONS—*continued.*

To Associateship—

Blair, Thomas Hamilton (*Glasgow, Lanarkshire*).
Goudie, Laurence (*Kisumu, Kenya*).
Lambrechts, Jacobus de Villiers (*Johannesburg, Transvaal*).
Michell, James Richard (*Okehampton, Devonshire*).
Mitchell, Elvert Richard (*Estantia, Transvaal*) (reinstatement).
Nicolle, Owen Arthur Mellish (*Gwanda, Southern Rhodesia*).
Nichols, John Gibson (*Derby*).
Ransby, Peter Foreman (*Royal Engineers*).
Roberts, Henry Angwin (*Coromandel, South India*).
Schindler, Norman Rudolf (*Springs, Transvaal*).
Somerset, Francis John Burchall (*Royal Engineers*).
Thorburn, John Wood (*Mt. Isa, Queensland, Australia*).
Wood, William Olive (*Jos, Northern Nigeria*).

To Studentship—

Alexander, Hamish Johnston (*London*).
Bellman, George William (*Chester, Cheshire*).
Clive, Philip Alan (*York*).
Cocking, Joseph Albert (*Plymouth, Devonshire*).
Dyer, John (*London*).
Gawthrop, Michael (*Ulverston, Lancashire*).
Hogg, David Jenner (*Bulawayo, Southern Rhodesia*).
Lee, Edward Michael Paterson (*Cheam, Surrey*).
Morris, Derrick (*London*).
Phillips, Daniel Hague (*Luanshya, Northern Rhodesia*).
Phillips, Frederic Roger (*Slough, Buckinghamshire*).
Tinker, William Haydn (*London*).
Wright, Kenneth Paul (*Marikuppam, South India*).

MEMBERS ON SERVICE WITH H.M. FORCES.

A full list was published in Bulletin *No.* 456, *September,* 1942. *The following additions or changes are supplementary to those already published.*

Members.

Captain E. E. G. Boyd, *Perak Local Defence Corps.*
Lieutenant-Colonel J. Weekley, *Perak Local Defence Corps.*

Associates.

Captain A. W. Burne, *Royal Engineers* (Promoted).
Lieutenant A. G. Palmer (*U.K. Forces,* not *A.I.F.* as published in November, 1944, *Bulletin*).
Major P. F. Ransby, *Royal Engineers.*

Students.

Major W. R. Cowlin, *Bengal Sappers and Miners* (Promoted).
Captain M. de F. Haddon, *Royal Marines* (Promoted).
Flight-Lieutenant J. A. Konschel, *Royal Air Force* (Promoted).
Squadron Leader R. C. Pargeter, *Royal Air Force Volunteer Reserve* (Awarded Distinguished Flying Cross, April, 1944).
Lieutenant W. Symes, *Royal Engineers* (Promoted).

NEWS OF MEMBERS.

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

Mr. D. G. ARMSTRONG, *Associate*, has left England to return to Ashanti Goldfields Corporation, Ltd., Obuasi, Gold Coast.

Mr. W. S. A. BAKER, *Student*, has arrived in England after being released from a Japanese internment camp.

Mr. N. G. BALL, *Associate*, has resigned his post with Non-Ferrous Minerals Development, Ltd., and has joined the staff of Procea Products, Ltd.

Mr. H. O. BERRYMAN, *Associate*, has been released from the South African Military Forces and is on leave in England before resuming his work as inspector of mines, Tanganyika Territory.

Mr. O. J. BLAU, *Student*, who was a prisoner of war in Japanese hands, has rejoined the staff of the Zinc Corporation, Ltd., Broken Hill, N.S.W.

Mr. J. C. BOLSOVER, *Associate*, is now in England after three years in the Tavoy civilian internment camp.

Mr. E. J. BROOKER, *Associate*, has left the U.S.A. for Honduras, Central America.

Mr. W. BROWN, *Associate*, has left England on his return to the Gold Coast.

Mr. C. M. CARROLL, *Member*, has left England on his return to Peru.

Mr. N. D. CHOPRA, *Associate*, having concluded his service in New York under the British Foreign Office Volunteer Scheme, has restarted his practice in London as consulting metallurgical and mining engineer.

Cpl. T. F. V. COOPER, R.E., *Student*, expects to be in England about April.

Mr. C. W. N. CRAIG, *Associate*, has been released from the Royal Air Force.

Mr. C. C. CULLEN, *Associate*, has been demobilized from the South African Air Force and has taken up a post with East Geduld Mines, Ltd.

Dr. WILLIAM CULLEN, *Member*, is leaving shortly for South Africa and expects to be out of England for at least three months.

Mr. E. R. DEMPSTER, *Associate*, has returned to India after leave in the United Kingdom.

Mr. A. F. DICK-CLELAND, *Member*, has vacated his post as an assistant inspecting officer of armament, Ministry of Supply.

Mr. E. F. ELKAN, *Associate*, is now in England after release from a prisoner-of-war camp in Singapore.

Mr. G. H. FAIRMAID, *Member*, is now in New Zealand after his release from internment in Malaya.

Mr. J. H. FRENCH, *Associate*, has left the Transvaal to take up an appointment on the staff of the Guinea Fowl Miners Training School, Southern Rhodesia.

Mr. W. H. GOLDSWORTHY, *Student*, has left England to take up an appointment as Inspector of Mines, Nigeria.

Mr. C. D. HALLAM, *Associate*, has left the Transvaal and is now in South West Africa.

Mr. P. T. HALSEY, *Member*, has relinquished his post as managing engineer with the Royal Ordnance Factories.

Mr. E. P. HARGRAVES, *Member*, is now in England on a visit from Australia.

Mr. J. I. HARRIS, *Associate*, has left the Ministry of Supply to take up a post with Sturtevant Engineering Co., Ltd.

NEWS OF MEMBERS—*continued.*

Mr. H. L. H. HARRISON, *Member*, has left England for Malaya to resume his post with Anglo-Oriental (Malaya), Ltd.

Mr. H. L. HOLLOWAY, *M.M.*, *M.C.*, *Associate*, has left England for the Gold Coast.

Mr. D. HOSKING, *Student*, has left England for the Gold Coast to join the staff of Taquah and Abosso Mines, Ltd.

Mr. J. HUNTER, *Associate*, has left England on his return to Nigeria.

Mr. W. HUTCHIN, *Associate*, has left England for the Gold Coast on taking up duties again with Bibiani (1927), Ltd.

Mr. R. F. JARVIS, *Student*, has left England for India to join the staff of Mysore Gold Mining Co., Ltd.

Mr. H. R. KERR, *M.C.*, *Member*, has accepted an appointment with the Control Commission for Germany.

Mr. R. KUTTNER, *Associate*, has been released from the South African Air Force.

Mr. A. LEAVER, *Associate*, has left England for Curacao.

Mr. W. D. LESLIE, *Student*, has left Northern Rhodesia to take up an appointment with the London and Rhodesian Mining and Land Co., Ltd., at Eiffel Flats.

Mr. H. R. MACKILLIGIN, *Member*, is returning to England from India.

Mr. R. K. MCLEOD, *Associate*, has arrived in England on leave from Northern Rhodesia.

Mr. GILBERT MCPHERSON, *M.C.*, *Member*, has been appointed consulting engineer to Selection Trust, Ltd.

Mr. D. M. MORGAN, *Student*, has left England to join the staff of Champion Reef Gold Mines of India, Ltd.

Mr. W. L. G. MUIR, *Associate*, has joined the staff of the coal-mining section of the Control Commission for Germany.

Mr. W. A. C NEWMAN, *Member*, has been appointed Chief Assayer, Royal Mint.

Mr. D. S. NICHOLSON, *Associate*, has been released from the New Zealand Forces.

Mr. C. MAXWELL NORMAN, *Member*, has left England for the United States of America.

Mr. W. J. S. OATES, *O.B.E.*, *M.C.*, *Member*, having been released from Military Service, is now in Kenya.

Mr. J. E. OGILVIE, *Associate*, has arrived in Scotland after release from internment at Singapore.

Mr. T. V. O'HARE, *Associate*, is now in England after three-and-a-half years' imprisonment in Japanese camps.

Mr. R. S. OPIE, *Associate*, has terminated his appointment with the Ministry of Supply in the department of the Chief Inspector of Armaments.

Mr. G. C. PENGILLY, *Student*, has taken up a post with Konongo Gold Mines, Ltd., Gold Coast.

Mr. P. G. PETROPOULOS, *Student*, has left England for Cyprus.

Mr. J. J. H. PORTER, *Associate*, has joined the staff of Messrs. Fisons, Ltd., and has left England on a visit to the United States on behalf of the company.

Mr. R. A. PURVIS, *Member*, has been appointed general manager of Nanwa Gold Mines, Ltd., and expects to leave England shortly.

OBITUARY—*continued.*

Lieutenant Peter Donald Hopkins, Royal Engineers, died on July 16th, 1945, at the age of 30. He entered the Royal School of Mines in October, 1933, and graduated with the A.R.S.M. and the degree B.Sc. (Eng.) of the University of London. He joined the staff of Ex-Lands Nigeria, Ltd., in October, 1938, but left in 1940 to enlist in the Royal Engineers.

He was elected a Student of the Institution in 1937.

Reginald Frank Krall died on October 13th, 1945, at the age of 72. From 1890 to 1893 he obtained his technical training in civil and mechanical engineering at the Central Technical College, South Kensington, and was awarded the A.C.G.I. Diploma and a Works Premium Scholarship of £50. He entered the works of Messrs. W. H. Allen Sons & Co., Ltd., of Bedford, where he remained for about fifteen months acquiring a practical knowledge of engineering, and for the next three years was engaged as a designer in the drawing office of Messrs. Stothert & Pitt, of Bath. In 1897 he was appointed chief engineer of the Cyanide Plant Supply Co., Ltd., London, and his work covered the designing and supervision of plant for cyanide treatment and also for crushing and concentrating. He was also commercial manager of the company, and his connection with them lasted until 1909, when he set up in business as consulting engineer and contractor in metallurgical plants and general engineering. Mr. Krall gave up the business to enlist in the Army in 1914, and served for over four years in France, Egypt and Salonika, and attained the rank of Staff Sergeant in the R.A.M.C. (T.). He held the position in 1919 of Director of Auctions (Machinery) at the Ministry of Munitions, and on demobilization took up the appointment of chief designer, buyer and business representative for Minerals Separation, Ltd. In 1923 he joined the staff of Woodall-Buckham Vertical Retort and Oven Construction (1920), Ltd., as a mechanical engineer, and until his death held a senior position in the design department of that company, having particular relation to the design of coke ovens and by-product recovery plant. He spent some of this time abroad, especially in the U.S.A., studying American design and practice.

He was elected to Studentship of the Institution in 1899 and to Membership in 1906. He was an Associate Member of the Institution of Civil Engineers and a former Vice-Chairman of the Junior Institution of Engineers.

Flight-Lieutenant John O'Malley Lyons, Royal Australian Air Force, died on October 10th, 1945, at the age of 35. He was born in Australia and graduated in 1930 with the degree of B.M.E. of Melbourne University. He joined the staff of Mount Isa Mines, Ltd., in May, 1931, holding the positions consecutively of sub-foreman at the smelter, shift metallurgist at the concentrator, shift foreman, and mill metallurgist. He left in January, 1934, to work for three months as assistant mine manager at Commonwealth Gold Development, Ltd., and then spent another three months gaining mining experience with A.I. Consolidated Gold, N.L. He joined Talbot Alluvials, Ltd., Victoria, in September, 1934, and until the beginning of 1937 was engaged in various mining engineering duties connected with the deep-lead mines, then becoming mine manager and, four months later, general manager and attorney. He left on the closing

OBITUARY—*continued.*

down of the properties in 1941, and joined the R.A.A.F., where he rose to the rank of flight lieutenant.

He was elected a Member of the Institution in 1944.

Wolfram Hermann Albert Penseler died in Sine Road prisoner-of-war camp, Singapore, on November 2nd, 1944, at the age of 42. He was born in New Zealand, and in 1918 studied at the Thames School of Mines, entering the Waihi School of Mines in 1919. From 1920 to 1924 he was a student at the Otago School of Mines, Dunedin, during which time he also undertook practical metal and coalmining work. In 1922 he was awarded the Ulrich Memorial Prize and Medal for mineralogy and petrology, and in 1924 was granted the degrees of B.E. (Mining) and B.Sc. (New Zealand) and the Diploma of Associateship in Mining of the Otago School of Mines. He subsequently gained the degree of B.E. (Metallurgy) and the A.O.S.M. (Metallurgy). During 1925 he carried out briquetting research for the New Zealand Government and also continued his geological studies at Victoria University College, doing research work on the microstructure of New Zealand coal, for which he was awarded the degree of M.Sc. in 1926. For his subsequent researches in the micropetrological examination of the structure of New Zealand coals, he had conferred on him the degree of Doctor of Science in geology. During the 1926 Session of the Otago School of Mines he acted as professor of metallurgy, and for two years was director of Huntly School of Mines.

Dr. Penseler then held various positions in New Zealand, first in charge of prospecting operations for coal and gold at Charleston for I.C.I., Ltd., then at Golden Point mine. He was battery superintendent at Monowai G.C.L. Mines, Ltd., Thames, for five months, and then became mining engineer to the Unemployment Board, New Zealand, later joining Investigations, Ltd., a prospecting and development company, for ten months. From 1934–35 he was prospecting and reporting on alluvial gold areas for Austral Malay Tin, Ltd., at Cromwell, New Zealand, and in January, 1936, went to Malaya for the same company as manager of Puchong Tin Dredging, Ltd. In 1937 he was working with Asam Kumbang Tin Dredging, Ltd., at Taiping, Perak, and early in 1939 joined Raub Australian Gold Mining Co., Ltd., where he became general manager. Dr. Penseler was captured by the Japanese with others of the company and was interned in Changi and Sine Road camps. News of his captivity was received in 1944, and it has since been learned that he had been subjected to solitary confinement for seven months prior to his death.

Dr. Penseler was elected a Student of the Institution in 1925 and was transferred to Associateship in 1927.

Captain Ronald John Coulson Telford, Australian Electrical and Mechanical Engineers, died on August 27th, 1945, after an illness while on active service at Balik Papan, Borneo, at the age of 30. Born in Australia, he studied at the University of Western Australia from 1933 to 1936 and from June to August, 1939, graduating in March, 1940, with the degree of B.Eng., with honours in engineering geology. From March, 1937, until his return to the University in 1939, he worked as a geologist

OBITUARY—*continued.*

on the Aerial Geological and Geophysical Survey of Northern Australia, and for eight months from September, 1939, was employed by Mount Lyell Mining and Railway Co., Ltd., on an investigation of properties at Queenstown, Tasmania. In May, 1940, he was appointed draughtsman and assistant surveyor in the engineering department of Big Bell Mines, Ltd., at Big Bell, Western Australia, and a month later became mine surveyor.

He joined the Australian Imperial Force in February, 1942, attaining the rank of captain in the Australian Armoured Corps, and in 1944 was transferred to the A.E.M.E. in charge of the workshop attached to an engineer company. He was elected to Studentship of the Institution in 1939 and transferred to Associateship in 1941.

Private Brian Frank Tyson died of malaria at Bisroot on the Palong River, Northern Johore, on January 26th, 1943, at the age of 38. A New Zealander, he was a student at the Otago School of Mines from 1924 to 1928, graduating with the degrees of B.Sc. (Geol.) and B.Eng. (Min.), and thereupon took up a position as drill superintendent with the Siamese Tin Syndicate on an alluvial deposit in Otago. In July, 1929, he was appointed assistant mining engineer, Pahang Consolidated Co., Ltd., at Sungei Lembing, Pahang, F.M.S., and seven years later was promoted mining engineer, becoming underground manager in 1939. His whereabouts were unknown after the Japanese invasion of Malaya until, in 1945, Mr. Tyson was reported to have been mentioned in a letter from a member of the Institution who had been with the same company and was in Malai prisoner-of-war camp. The report of his death was received from the New Zealand Missing and Prisoners of War Agency.

Mr. Tyson was elected to Associateship of the Institution in 1939.

The Council regret to report the death of **Vincent Brice Carew Baker,** *Member*, in April, 1944; **John Taylor Marriner,** *Member*; **William Robert Wilson Ronald Scott,** *Associate*, on November 13th, 1945; and **Frederick Percy Tremble,** *Associate*. Obituary notices will be published later.

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

ATOMIC ENERGY. A General Account of the Development of Methods of Using Atomic Energy for Military Purposes under the Auspices of the United States Government, 1940-1945. By H. D. Smyth. 144 pp. (Published in U.S.A. by the Government Printing Office.) London: H.M.S.O., 1945. 2s. 6d.

A TECHNIQUE FOR MAKING TORSION BALANCE SURVEYS OF INUNDATED AREAS. By K. F. Hasselmann. 31 pp. and maps. Bulletin, University of Missouri School of Mines and Metallurgy, Technical Series, Vol. 16, No. 2, June, 1945. Rolla, Mo.: Published by Missouri School of Mines and Missouri State Mining Experiment Station, 1945.

BRITISH STANDARDS INSTITUTION: SERVICES SCHEDULE OF NON-FERROUS METALS AND ALLOYS. STA 7: Group 1, Copper and its alloys, September, 1945 (revision of section published December, 1942). 19 pp. 2s. STA 7: Group 6, Aluminium and its alloys. 13 pp. BS 1259, 1945: Intrinsically safe electrical apparatus and circuits. 9 pp. 2s. BS 1225, 1945: Recommended methods for polarographic and spectrographic analysis of high purity zinc and zinc alloys for die casting. 36 pp. 2s. London: B.S.I., 1945.

CANADA: GEOLOGICAL SURVEY PAPERS AND MAP: Memoir 239—Mesozoic stratigraphy of the Eastern Plains, Manitoba and Saskatchewan, by R. T. D. Wickenden. 25 cents. List of published maps (1917–1945 inclusive), compiled by P. J. Moran. Paper 44–17 (2nd edition)—Revision of the lower Cretaceous of the Western Interior of Canada (report and 12 fossil plates), by F. H. McLearn. Paper 45–19—Fall Creek map-area, Alberta (report and map), by J. F. Henderson. Paper 45–20—Greenwood-Phoenix area, British Columbia (report and map), by D. A. McNaughton. Map 828A—Windsor-Sarinia, Essex, Kent and Lambton Counties, Ontario. Ottawa: Department of Mines and Resources—Mines and Geology Branch, 1945.

CANADIAN INSTITUTE OF MINING AND METALLURGY AND MINING SOCIETY OF NOVA SCOTIA: *Transactions*, Vol. 47, 1944. 531 pp. Montreal: Published by the Institute, 1945.

CHAMBER OF MINES OF RHODESIA (Incorporated) SIXTH ANNUAL REPORT FOR THE YEAR 1944. 50 pp. Bulawayo: Chamber of Mines of Rhodesia, 1945.

DURHAM COALFIELD: Regional Survey Report (Northern 'B' Region) to Minister of Fuel and Power. 48 pp. with plan. London: H.M.S.O., 1945. 1s.

EGYPT—MINISTRY OF FINANCE: SURVEY DEPARTMENT: A preliminary report on the geology of the Eastern District of Egypt, between latitude 22° N. and 25° N., by W. F. Hume. 72 pp. (Survey Dept. Paper No. 1, 1907. 150 milliemes.) Explanatory notes to accompany the geological map of Egypt, with tables showing distribution of geological formations and economic products, by W. F. Hume. 49 pp. (1912. P.T. 10.) The geography and geology of south-eastern Egypt,

LIST OF ADDITIONS TO THE JOINT LIBRARY—*continued.*

by John Ball. 394 pp. (1912. P.T. 40.) Topography and geology of the phosphate district of Safaga (Eastern Desert of Egypt), by John Ball. 19 pp. (Survey Dept. Paper No. 29, 1913. P.T. 10.) A brief note on the phosphate deposits of Egypt, by John Ball. 6 pp. (Survey Dept. Paper No. 30, 1913. P.T. 5.) Report on the oilfields region of Egypt, by W. F. Hume, with a geological map of the region from surveys by John Ball. 103 pp. (1916. P.T. 25.) Photo-reproductive methods and processes used by the Survey of Egypt, by G. Douglas. 25 pp. (Survey of Egypt Paper No. 36, 1920. P.T. 5.) DEPARTMENT OF MINES AND QUARRIES: Rules and regulations as to mining. 94 pp. (1921. P.T. 6.) PETROLEUM RESEARCH: Bulletin No. 1—Preliminary geological report on Abu Dúrba (Western Sinai), by W. F. Hume, T. G. Madgwick, F. W. Moon, and H. Sadek. 20 pp. (1921. P.T. 10.) Bulletin No. 6—Preliminary geological report on the Abu Shaar El Qibli (Black Hill) District, by T. G. Madgwick, F. W. Moon, and H. Sadek. (1920. P.T. 10.) Cairo: Government Press.

EXPLORATION GEOPHYSICS. By J. J. Jakosky. 786 pp., illus. Los Angeles: Times-Mirror Press, 1940. £2 16s.

KENYA: GEOLOGICAL SURVEY OF EASTERN MARAMMA AND ADJOINING AREAS, WESTERN KAKAMEGA, NORTHERN KAVIRONDO (with map). By William Pulfrey. 38 pp. Mining and Geological Department Report No. 8. Nairobi: Government Printer, 1945. Sh. 2.

LA INTERPRETACION GEOLOGICA DE LAS MEDICIONES GEOFISICAS: APLICADAS A LA PROSPECCION—TOMO III. By Jose G. Sineriz. 573 pp. Madrid: Instituto Geologico y Minero de España, 1944.

MANGANESE AND IRON DEPOSITS OF MORRO DO URUCUM, MATO GROSSO, BRAZIL. By J. Van N. Dorr. 47 pp., with appendixes and illustrations. U.S. Department of the Interior Geological Survey Bulletin 946–A. Washington, D.C.: U.S. Government Printing Office, 1945. 75 cents.

MANVERS MAIN COLLIERY, SOUTH YORKSHIRE: Report on the causes of, and circumstances attending, the explosion which occurred on the 4th March, 1945, at the Manvers Main Colliery, Wath-on-Dearne, South Yorkshire. By J. R. Felton. Report to Minister of Fuel and Power. 28 pp. Cmd. 6688. London: H.M.S.O., 1945. 1s. 3d.

MINISTRY OF FUEL AND POWER STATISTICAL DIGEST FROM 1938. 69 pp. Cmd. 6538. London: H.M.S.O., 1944 (reprinted 1945). 1s. 6d.

NEW DEAL FOR COAL. By Harold Wilson. 264 pp. London: Contact Publications, Ltd., Cole & Co. (Westminster), Ltd., 1945. 8s.

NIGERIA, GEOLOGICAL SURVEY DEPARTMENT: ANNUAL REPORT, 1944. 16 pp. Lagos: Government Printer, 1945.. 1s.

NORTH EAST COAST INSTITUTION OF ENGINEERS AND SHIPBUILDERS, *Transactions*, Vol. 61, 1944–45. 356 pp. plus 250 pp. discussion, obituary and index. Newcastle-upon-Tyne: Published by the Institution, 1945.

LIST OF ADDITIONS TO THE JOINT LIBRARY—*continued.*

Ore Dressing Investigations : (1) Joint investigations of the Council for Scientific and Industrial Research and the South Australian School of Mines and Industries. Reports from the Bonython Laboratory, New Series Nos. 160, 162, 163. (2) Joint investigations of the C.S.I.R. and the Kalgoorlie School of Mines. Reports from the Kalgoorlie Metallurgical Laboratory, Nos. 259, 265–6, 268–9, 271–81, Index of Reports issued to March, 1944. (3) Joint investigations of the C.S.I.R. and the University of Melbourne. Reports from the University Metallurgical Laboratory, Nos. 273–4, 276–83, 285–6.

Portugal : Comunicaçoes dos Serviços Geológicos de Portugal. Tômo XXIV and XXV. 156 pp. and 251 pp. Lisbon : Direcção Geral de Minas e Serviços Geólogicos, 1943 and 1944.

Powder Blast Mining in Conjunction with Long-Hole Blasting. By Howard M. Fowler. 12 pp. Bulletin, University of Missouri School of Mines and Metallurgy, Technical Series Vol. 16, No. 3, September, 1945. Rolla, Mo. : Published by Missouri School of Mines and Missouri State Mining Experiment Station, 1945.

Proposed Scheme for the Recovery of Natural Gas from Coal Seams. By Gilbert McPherson and F. B. McPherson. 32 pp., with tables and plan. (In typescript.) (*Presented to the Institution of Mining Engineers by Professor S. J. Truscott.*)

Report of the Coal Commission for the year ended 31st March, 1945. 8 pp. London : H.M.S.O., 1945. 2d.

Safety in Mines Research Board : Twenty-third Annual Report, 1944. 30 pp. London : H.M.S.O., 1945. 1s.

Scottish Sources of Alkali Feldspar. By T. Robertson. 35 pp. Wartime Pamphlet No. 44 of D.S.I.R. Geological Survey of Great Britain. London : Geological Survey and Museum, 1945. 1s. 9d.

Spain : Instituto Geologico y Minero de Espana : *Notas y Comunicaciones,* numero 13, 1944. 322 pp. Data for the study of geological map sheets, 1 : 50,000—Gijon (14) and Oviedo (29). 111 pp. Geological Maps—explanations of sheets nos. 245 (Sadaba), 172 (Allo), 390 (Cervera), and 629 (Toledo). Madrid : Published by the Institute, 1941–4.

State Taxation of Metallic Deposits. By Warren A. Roberts. Harvard Economic Studies, Vol. 77. 400 pp. Cambridge, Mass. : Harvard University Press, 1944. $4.50 (£1 7s.).

Stratigraphy and Oil-Producing Zones of the Pre - San Andres Formations of Southeastern New Mexico—a preliminary report. By Robert E. King. 31 pp. and 3 plates. New Mexico School of Mines Bulletin No. 23. Socorro, N.M. : State Bureau of Mines and Mineral Resources, 1945. 50 cents.

LIST OF ADDITIONS TO THE JOINT LIBRARY—*continued.*

THE COAL MINES (CARDOX AND HYDROX) ORDER, 1945, dated November 13, 1945, made by the Minister of Fuel and Power under section 61 of the Coal Mines Act, 1911. S. R. & O., 1945, No. 1468. 1 p. London: H.M.S.O., 1945. 1d.

THE MINERAL RESOURCES OF THE LOTHIANS. By A. G. MacGregor. Wartime Pamphlet No. 45 of D.S.I.R., Geological Survey of Great Britain. 43 pp. London: Geological Survey and Museum, 1945. 2s.

THE MONMOUTHSHIRE AND SOUTH WALES COAL OWNERS' ASSOCIATION: REPORTS OF THE GENERAL RESEARCH COMMITTEE: 1—The Introduction of sodium vapour lamps underground in British collieries; April, 1941. 2—Ventilation surveying, with special reference to the inclined gauge and tube method of pressure surveying; October, 1941. 3—Atmospheric conditions in mines—a study of the physical condition of mine air; August, 1942. 4—A new and rapid method for the determination of carbon dioxide in mine dusts; September, 1942. REPORTS OF THE COAL DUST RESEARCH COMMITTEE: 1—The design and testing of a dust collector for a conveyor loading point in an intake roadway. 2—Sprays. 3—The suppression of dust at the main gate transfer point of a conveyor district. 4—The removal of dust clouds from main gate loading stations by utilization of waste workings; January, 1942. 5—Dust reduction and suppression on machine cut faces—interim report; September, 1942. 6—A new type of filter for dust work; October, 1942. 7—Experience gained in sampling; December, 1942. 8—Limitations to the use of the konimeter, by C. S. Chubb; February, 1943. 9—Pneumonokoniosis of coal workers—problems of the assessment of the dust hazard; July, 1943. 10—Dust reduction and suppression on machine cut faces—further report; September, 1943. 11—Pneumonokoniosis of coal workers—a method for measuring the health hazard from air-borne dust—interim report; November, 1943. 12—The use of water sprays for dust suppression at the coal face; March, 1944. 13—Pneumonokoniosis of coal workers—the evaluation of the health hazard of air-borne dust—further report; July, 1944. 14—The use of wetting agents for the suppression of air-borne dusts—interim report; December, 1944. 15—A survey of airborne dust conditions in the South Wales coalfield; April, 1945. 16—Pneumonokoniosis of coal workers—the evaluation of the health hazard of air-borne dust with special reference to particle size analysis—further report; July, 1945. *Presented to the Institution of Mining Engineers by the Director of Research, The Monmouthshire and South Wales Coal Owners' Association.*

TUNGSTEN DEPOSITS IN BEAVER COUNTY, UTAH. By S. W. Hobbs. Pp. 81–111, with separate illustrations. U.S. Department of the Interior Geological Survey Bulletin 945—D. Washington, D.C.: U.S. Government Printing Office, 1945. 65 cents.

INDEX OF RECENT ARTICLES.

NOTE.—All Articles indexed are contained in the Library of the Institution.

ANALYSIS AND CHEMISTRY.

ANALYSIS — COLORIMETRIC—MOLYBDENUM.—Colorimetric determination of molybdenum in iron and steel. Mitchell Kapron and Paul L. Hehman.—*Industr. Engng. Chem.*, Analyt. Ed., Easton, Pa., Vol. 17, September, 1945, pp. 573-6. 50 cents.

ANALYSIS — SILICA DETERMINATION. —Determination of small amounts of silica. M. F. Adams.—*Industr. Engng. Chem.*, Analyt. Ed., Easton, Pa., Vol. 17, September, 1945, pp. 542-3. 50 cents.

ANALYSIS — SILICATES.—Determination of sodium and potassium in silicates—an improved method. George G. Marvin and Lawrence B. Woolaver.—*Industr. Engng. Chem.*, Analyt. Ed., Easton, Pa., Vol. 17, September, 1945, pp. 555-6. 50 cents.

SPECTROGRAPHIC ANALYSIS.—Arc and spark characteristics for quantitative spectrographic analysis and methods used for their control. J. Convey.—*Bgham. Metall. Soc. J.*, Birmingham, Vol. 25, No. 3, September, 1945, pp. 168-89. 2s. 6d.

SPECTROGRAPHIC ANALYSIS.—Further applications of the spectrograph to the quantitative analysis of ferrous and non-ferrous methods. L. G. Young.—*Bgham. Metall. Soc. J.*, Birmingham, Vol. 25, No. 3, September, 1945, pp. 190-200. 2s. 6d.

COAL.

ANALYSIS.—Methods for the determination of carbon and hydrogen in coal and their practical utility: a note on the Fenton method for the determination of carbon and hydrogen. John H. Jones.—Inst. Fuel *War Time Bull.*, October, 1945, p. 37.

CHINA.—Coal in North China.—*Colliery Engng.*, Lond., Vol. 171, December, 1945, pp. 288-9. 1s.

DUST.—Dust in conveyor districts—VI: reasons why loader gate should be made an intake rather than a return airway in order to minimize the dust problems. VII: How the more common methods of loading affect the raising of dust at loading points. W. A. Terrell.—*Colliery Engng.*, Vol. 22, 1945; November, pp. 250-252; December, pp. 277-9. 1s. each.

DUST SUPPRESSION.—Dust in return roadways: a review of causes and available means for its suppression. (*To be continued.*) W. A. Terrell. (Paper read before National Association of Colliery Managers).—*Iron Coal Tr. Rev.*, Lond., Vol. 151, December 7, 1945, pp. 885-9. 9d.

DUST SUPPRESSION.—Dust suppression by wet rotary drilling. S. Flax.—*Colliery Guard.*, Lond., Vol. 171, November 16, 1945, pp. 599-601. 10d.

ECONOMICS—CANADA.—The coal industry of Canada: brief submitted by the Canadian Institute of Mining and Metallurgy to the Royal Commission on the Coal Industry of Canada.—*Canad. Min. Metall. Bull.*, Montreal, No. 402, October, 1945, pp. 517-22. $1.00.

COAL—*continued.*

FUEL—CHARCOAL.—Charcoal briquettes as a coal substitute. H. Greene and T. N. Jewitt. (Cold wet charcoal-pitch mixture, with gum arabic as primary binder.)—*J. Soc. Chem. Ind.*, Lond., Vol. 64, October, 1945, pp. 265-71. 3s. 6d.

GEOLOGY—FAULTS.—The dislocation of coal seams. A. Nelson.—*Colliery Guard.*, Lond., Vol. 171, 1945; November 9, pp. 569-72; November 16, pp. 602-5. 10d. each.

MINE LIGHTING.—Lighting and the Reid Report. C. S. Chubb.—*Colliery Engng.*, Lond., Vol. 22, November, 1945, pp. 253-4 and 258. 1s.

MINING METHODS — MECHANICAL. — Double cutting, and a new application of a short chain conveyor: two interesting schemes of machine mining. C. G. Widdas. (Paper read before National Association of Colliery Managers.)—*Iron Coal Tr. Rev.*, Vol. 151, November 9, 1945, pp. 711-5.

MINING METHODS — MECHANICAL. — Mechanization in coal mines. W. L. Wearly and Henry Thies.—*Trans.* Canad. Inst. Min. Metall., Vol. 48, 1945, pp. 638-44. (*Canad. Inst. Min. Metall. Bull.*, Montreal, No. 402, October, 1945. $1.00.)

MINING METHODS — MECHANICAL—LOADING.—Machine loading taking top and coal raises entry output. J. H. Edwards.—*Coal Age*, N.Y., Vol. 50, October, 1945, pp. 88-92. 35 cents.

MINING METHODS—OPENCAST.—Stripping improvements mark Rimersburg expansion program. (Installation of large dragline at Fox mine of the Rimersburg Coal Co., Rimersburg, Pa.).—*Coal Age*, N.Y., Vol. 50, October, 1945, pp. 93-6. 35 cents.

MINING METHODS—U.S.A.—Some notes on American coal-mining practice, based on experience gained during a recent visit to the principal coalfields. H. S. Haslam. (Paper read before Yorks. Branch of National Association of Colliery Managers.) —*Iron Coal Tr. Rev.*, Lond., Vol. 151, 1945; November 23, pp. 797-801; November 30, pp. 843-7. 9d. each.

PREPARATION FOR MARKET — BRIQUETTING.—Developments in making coal briquettes. C. Campbell.—*Colliery Guard.*, Lond., Vol. 171, October 26, 1945, pp. 510-12. 10d.

PREPARATION FOR MARKET — BRIQUETTING.—Sodium silicate in briquette-making: French wartime substitute for pitch.—*Chem. Tr. J.*, Lond., Vol. 117, November 30, 1945, pp. 609-10. 8d.

PREPARATION FOR MARKET — PLANT MAINTENANCE.—Getting results in maintaining coal-preparation plants. Fred W. Richart.—*Coal Age*, N.Y., Vol. 50, October, 1945, pp. 105-10. 35 cents.

RESEARCH — CANADA.—Applied research in the bituminous coal industry. Henry F. Hebley.—*Trans.* Canad. Inst. Min. Metall., Vol. 48, 1945, pp. 539-66. (*Canad. Min. Metall. Bull.*, Montreal, No. 401, September, 1945. $1.00.)

SPITZBERGEN.—The coal mining situation in Spitzbergen.—*Min. J.*, Lond., Vol. 225, November 10, 1945, pp. 756-7. 8d.

INDEX OF RECENT ARTICLES—*continued.*

COPPER.

GENERAL.—Copper: extracts from articles appearing in various current periodicals. —*Developments and Trends in American Industries*, S-7306, compiled by the U.S. Office of War Information.

GEOLOGY — CHILE.—Geologia de los yacimientos de cobre de Chile.—*Boletin Minero*, Sociedad Nacional de Mineria, Santiago de Chile, Vol. 57, May, 1945, pp. 284-91.

GEOLOGY — ONTARIO — KAMISKOTIA LAKE AREA.—Structure of mineralised zone, Kam Kotia mine. Stewart A. Ferguson.—*Trans.* Canad. Inst. Min. Metall., Vol. 48, 1945, pp. 610-14. (*Canad. Min. Metall. Bull.*, Montreal, No. 401, September, 1945. $1.00.)

METALLURGY.—A study of the possibility of precipitation of antimony as oxychloride in copper refining electrolyte. Yu-lin Yao.—*Trans.* Electrochem. Soc., N.Y., Vol. 86, 1944, pp. 365-9.

METALLURGY.—Functions of chloride in copper refining electrolyte. Yu-lin Yao.—*Trans.* Electrochem. Soc., N.Y., Vol. 86, 1944, pp. 371-80.

METALLURGY — WESTERN AUSTRALIA.—Revival of copper smelting in Western Australia: description of experimental work to develop small blast furnace to smelt rich gold-copper ores.—*Chem. Engng. Min. Rev.*, Melb., Vol. 37, September, 1945, pp. 377-81. 1s.

PERU — CERRO DE PASCO.—Special number of *Mining and Metallurgy*, N.Y., Vol. 26, November, 1945. The origin and development of the Cerro de Pasco Copper Corporation (Oroya, Peru). D. H. McLaughlin, pp. 509-11: Geology of the mining region of Central Peru. D. H. McLaughlin and John H. Moses, pp. 512-19: Mining methods at the Cerro de Pasco properties. V. L. McCutchan, pp. 520-33: Ore concentration. T. R. Wright, pp. 534-9: Further notes on milling practice and flowsheet details. D. S. Sanders, pp. 540-44: Ore reduction—copper and lead smelting and lead refining. W. W. Fowler, J. W. Hanley, and I. L. Barker, pp. 545-58: Metallurgical research, including indium recovery. T. R. Wright, pp. 559-60: New bismuth alloys developed to find market for the metal. Walter C. Smith, pp. 561-2. The power system—three hydro-electric and one steam-electric plants. R. McDonald and B. C. Maine, pp. 563-6. 50 cents.

ECONOMICS.

BRAZIL.—Factors affecting investment in South American mining: Brazil. Newton B. Knox.—*Min. Metall.*, N.Y., Vol. 26, October, 1945, pp. 479-80. 50 cents.

EDUCATION—AUSTRALIA.—Mining and metallurgy as a career: I.—Explanation of type of work, list of mines and works, salaries and duties, training. II.—Requisite educational background, details of training available at universities and technical colleges. (Information prepared by the Australasian Inst. Min. Metall.)—*Chem. Engng. Min. Rev.*, Melb., Vol. 37, 1945; August, pp. 351-7; September, pp. 385-89. 1s. each.

EDUCATION—SOUTH AMERICA.—Some practical aspects of mineral industries education in the Latin Americas. Edward Steidle.—*Min. Metall.*, N.Y., Vol. 26, October, 1945, pp. 475-8. 50 cents.

ECONOMICS—*continued.*

MANAGEMENT — COLLIERIES. — Administration of collieries. (*To be continued.*) F. Colin Swallow.—*Colliery Engng.*, Lond., Vol. 22, November, 1945, pp. 255-8. 1s.

ORE RESERVES.—The concept of ore reserves: many factors enter into a proper definition of the term. S. G. Lasky.—*Min. Metall.*, N.Y., Vol. 26, October, 1945, pp. 471-4. 50 cents.

STATE IN RELATION TO MINING—SOUTH AFRICA.—The State and the gold mines: a review of a partnership. H. S. H. Donald. (Address delivered to S. African Institute of Electrical Engineers.)—*Min. and Industr. Mag. of S. Afr.*, Johannesburg, Vol. 33, September, 1945, pp. 549-63. 1s.

GEOLOGY AND ORE DEPOSITS.

ABERDEENSHIRE.—Structural features of the grey granites of Aberdeenshire. James Cameron.—*Geol. Mag.*, Hertford, Vol. 82, No. 5, September-October, 1945, pp. 189-204. 5s.

ARGENTINA.—Estudio geologico de la Sierra de la Ramada (Burruyacu, Tucuman). Abel Peirano.—*Industria Minera*, Buenos Aires, Vol. 5, No. 50, September, 1945, pp. 17-27. $1.50.

CORNWALL.—Notes on the geology of Carnmenellis (Cornwall). N. J. G. Pounds. —*Proc. Geol. Ass.*, Lond., Vol. 56, Part 3, October, 1945, pp. 143-6. 5s.

ORE DEPOSITS—OUTCROPS.—Weathered outcrops: an introduction to the study of weathered and leached outcrops. Frank Ebbutt.—*Canad. Min. J.*, Gardenvale, P.Q., Vol. 66, October, 1945, pp. 681-9. 25 cents.

PORTUGAL.—Contribuicao para o estudo dos granitos portugueses. J. M. Cotelo Neiva.—*Comunicacoes* dos Servicos Geologicos de Portugal, Lisboa, Tomo 24, 1943, pp. 29-42.

PORTUGAL.—Evolucao de falha do Ponsul. Orlando Ribeiro.—*Comunicacoes* dos Servicos Geologicos de Portugal, Lisboa, Tomo 24, 1943, pp. 109-23.

PORTUGAL.—Une reconnaissance géologique dans la région de Rio Maior: l'âge des lignites du gisement d'Espadanal. G. Zbyszewski.—*Comunicacoes* dos Servicos Geologicos de Portugal, Lisboa, Tomo 24, 1943, pp. 3-22.

SPAIN.—Datos para el conocimiento estratigrafico y tectonico del Pirineo Navarro. J. M. Rios, A. Almela and J. Garrid.—*Notas y Comunicaciones* del Instituto Geologico y Minero de Espana, Madrid, No. 13, 1944, pp. 141-64.

SPAIN.—Estudio geologico de la zona de Villanueva del Fresno (Provincia de Badajoz). Jose Meseguer Pardo e Ildefonso Prieto y Carrasco.—*Notas y Comunicaciones* del Instituto Geologico y Minero de Espana, Madrid, No. 13, 1944, pp. 165-80.

SPAIN.—Las formaciones del infracretaceo de Asturias. Bermudo Melendez Melendez. —*Notas y Comunicaciones* del Instituto Geologico y Minero de Espana, Madrid, No. 13, 1944, pp. 181-216.

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METALLURGY AND CYANIDING—*continued.*

POWDER METALLURGY.—The early days of nickel-tungsten powder metallurgy. M. Pirani.—*Trans.* Electrochem. Soc., N.Y., Vol. 85, 1944, pp. 163-8.

POWDER METALLURGY COLUMBIUM. Pure columbium. Clarence W. Balke.—*Trans.* Electrochem. Soc., N.Y., Vol. 85, 1944, pp. 89-95.

POWDER METALLURGY COPPER.—Copper in powder metallurgy. J. J. Cordiano.—*Trans.* Electrochem. Soc., N.Y., Vol. 85, 1944, pp. 97-106.

POWDER METALLURGY—COPPER.—Electrodeposition of copper powder. W. H. Osborn and S. B. Tuwiner.—*Trans.* Electrochem. Soc., N.Y., Vol. 85, 1944, pp. 107-18.

POWDER METALLURGY—GENERAL.—Powder metallurgy: reprints of extracts from articles appearing in various current periodicals.—*Developments and Trends in American Industries*, S-7125, compiled by the U.S. Office of War Information.

VACUUM METALLURGY.—Vacuum metallurgy: melting and evaporating metals at reduced pressures. (Paper read before The Electrochemical Society.) W. J. Kroll.—*Metal Ind.*, Lond., Vol. 67, 1945: November 9, pp. 295-7; November 16, pp. 323-4. 6d. each.

MILLING AND CONCENTRATION.

CRUSHING—CANADA.—The need for research in crushing, E. O. Lilge: Sherritt-Gordon crushing plant, Heath H. Hales: Crushing operations at Britannia, Wm. G. Hatch: Crushing operations at Pioneer, Paul Schutz: The Kelowna crushing plant, George L. Mill: Crushing at Copper Mountain and Allenby, L. H. McKay: Crushing practice at the Sullivan mine and concentrator, D. A. York: Small crushing plants, S. Gray and P. M. Elliott. (Papers presented at a symposium on crushing.)—*Trans.* Canad. Inst. Min. Metall., Vol. 48, 1945, pp. 668-714. (*Canad. Inst. Min. Metall. Bull.*, Montreal, No. 402, October, 1945. $1.00).

GENERAL.—Ore concentration, milling, dressing, flotation, secondary metal recovery: reprints of extracts from articles in various current periodicals.—*Developments and Trends in American Industries.* S-7126, compiled by the U.S. Office of War Information.

MINING (GENERAL).

ACCIDENTS—CAUSES. Mine accidents: some extraneous and environmental factors in accident causation. A. Nelson.—*Colliery Engng.*, Lond., Vol. 22, December, 1945, pp. 280-81 and 285. 1s.

ARGENTINA.—La industria minera Argentina.—*Industria Minera*, Buenos Aires, Vol. 5, No. 50, September, 1945, pp. 37-55. $1.50.

BLASTING COAL MINES.—The ignition of firedamp by explosives (the use of a clay plug at the back of the shothole as an extra precaution with sheathed explosives). H. C. Grimshaw.—*Colliery Guard.*, Lond., Vol. 171, October 19, 1945, pp. 476-7. 10d.

MINING (General) *continued.*

BRITISH COLUMBIA—HISTORY.—Vulcan in British Columbia; the life story of a mining industry. (*To be continued.*) H. G. Nichols.—*Canad. Min. J.*, Gardenvale, P.Q., Vol. 66, 1945; September, pp. 623-30; October, pp. 699-710. 25 cents each.

HAULAGE—DIESEL LOCOMOTIVES.—Diesel underground haulage. C. E. McKnight.—*Canad. Min. J.*, Gardenvale, P.Q., Vol. 66, October 1945, pp. 695-6. 25 cents.

HAULAGE—DIESEL LORRIES.—Diesel lorries underground at Mount Morgan: ore transport along 230-ft. drive, gas scrubbers fitted, adequate ventilation, conditions of work.—*Chem. Engng. Min. Rev.*, Melb., Vol. 37, September, 1945, pp. 375-6. 1s.

HAULAGE—ELECTRIC LOCOMOTIVES—N.S.W.—Electric locos at John Darling colliery (Newcastle, N.S.W.).—*Indust. Aust. Min. Stand.*, Melb., Vol. 100, August 15, 1945, p. 486. 1s.

HAULAGE—LOCOMOTIVE.—Locomotive haulage problems. R. McAdam. (Series of nomographs presented for convenient and rapid solution of problems on introduction of locomotives to underground haulage.)—*Colliery Engng.*, Lond., Vol. 22, December, 1945, pp. 273-6. 1s.

HOISTING—WIRE ROPES—LOCKED COIL.—Locked coil winding ropes. A. E. McClelland. (Tests on suspending them by means of oak clamps: long single *v.* multiple short clamps.)—*Colliery Guard.*, Lond., Vol. 171, November 23, 1945, pp. 629-31. 1s. *Iron Coal Tr. Rev.*, Lond., Vol. 151, November 30, 1945, pp. 849-51. 9d. *Colliery Engng.*, Lond., Vol. 22, December, 1945, pp. 286-7. 1s.

JAPAN.—Japan and her mining industry: severe contraction in extent.—*S. Afr. Min. Engng. J.*, Johannesburg, Vol. 56, Part II, October 6, 1945, p. 125. 6d.

MINE LIGHTING.—Lighting in hazardous situations, with particular reference to underground applications. S. W. Richards.—(From paper read before joint meeting of S. Wales Branch of National Association of Colliery Managers and two other bodies.)—*Iron Coal Tr. Rev.*, Lond., Vol. 151, November 23, 1945, pp. 809-11. 9d.

PROSPECTING—AERIAL. Aeroprospecting in the Yellowknife Area (Mackenzie). A. W. Jolliffe.—*Trans.* Canad. Inst. Min. Metall., Vol. 48, 1945, pp. 588-609. (*Canad. Min. Metall. Bull.*, Montreal, No. 401, September, 1945. $1.00.)

SILICOSIS—ALUMINIUM THERAPY.—Aluminium therapy in silicosis. (Set of nine papers on the subject, with foreword, republished from different sources.)—*Proc. Aust. Inst. Min. Metall.*, Melb., No. 138, June 30, 1945, pp. 19-148.

SURVEYING.—Spotting diamond drill holes to the nearest minute. Leon Valois and J. Richard C. Argall.—*Canad. Min. J.*, Gardenvale, P.Q., Vol. 66, October, 1945, pp. 693-4. 25 cents.

VENTILATION. Ventilation theory. (Basic principles reviewed in light of modern knowledge concerning flow of air.) A. A. Hirst.—*Colliery Engng.*, Lond., Vol. 22, November, 1945, pp. 243-6. 1s.

VENTILATION—DUST.—Stone headings: measures to combat the dust hazard and to ameliorate high temperature conditions. A. Nelson. *Mine and Quarry Engng.*, Lond., Vol. 10, November, 1945, pp. 269-72. 1s.

INDEX OF RECENT ARTICLES— *continued.*

MINING (General) —*continued.*

VENTILATION — FANS — EFFICIENCY. —The performance of fans: some considerations of design. Wm. C. Massie. (From an address to the Natal Institute of Engineers.)—*S. Afr. Min. Engng. J.*, Johannesburg, Vol. 56, Part II, October 6, 1945, pp. 131-6. 6d.

MINOR METALS.

ANTIMONY — TRANSVAAL.—S. African antimony (Murchison Range, Transvaal). —*Chem. Tr. J.*, Lond., Vol. 117, October 12, 1945, p. 402. 8d.

MANGANESE.—Effect of digesting, autoclaving and neutralizing some battery manganese dioxides. Earl Otto, George W. Vinal and Elinor H. Ostrander.—*Trans.* Electrochem. Soc., N.Y., Vol. 86, 1944, pp. 327-35.

MANGANESE — ELECTRO-METALLURGY.—The anode problem in the electrodeposition of manganese dioxide. O. W. Storey, Ernest Steinhoff and E. R. Hoff.—*Trans.* Electrochem. Soc., N.Y., Vol. 86, 1944, pp. 337-43.

MANGANESE — ELECTRO-METALLURGY.—The effect of certain variables on the electrodeposition of manganese. J. H. Jacobs, P. E. Churchward and R. G. Knickerbocker.—*Trans.* Electrochem. Soc., N.Y., Vol. 86, 1944, pp. 383-91.

MERCURY—SPAIN.—The world's oldest mercury mine: the history of the Almaden deposits (Cuidad Real, Spain).—*Chem. Tr. J.*, Lond., Vol. 117, October 5, 1945, p. 380. 8d.

PLATINUM METALS — USES.—Platinum and precious metals in the United Kingdom war-time activities.—*Canad. Min. J.*, Gardenvale, P.Q., Vol. 66, September, 1945, pp. 609-11. 25 cents.

PLATINUM METALS—USES.—Some industrial uses of the platinum metals. E. C. Rhodes.—*Bgham. Metall. Soc. J.*, Birmingham, Vol. 25, No. 3, September, 1945, pp. 132-59. 2s. 6d.

TUNGSTEN—BOLIVIA.—The Chicote tungsten deposit, Bolivia. Federico Ahlfeld.—*Econ. Geol.*, Lancaster, Pa., Vol. 40, No. 6, September-October, 1945, pp. 394-407. $1.00.

TUNGSTEN—BRITISH COLUMBIA.—Red Rose mine, Consolidated Mining and Smelting Company of Canada, Ltd. (Location, history, geology, mining, water and power, milling). H. M. Wright.—*Deco Trefoil*, Denver Equipment Co., Denver, Colorado, June, 1945, 4 pp.

TUNGSTEN — PORTUGAL. — Portuguese mineral deposits of cassiterite and wolframite. J. M. Cotelo Neiva. (In Portuguese: summary in English.)—*Comunicacoes* dos Servicos Geologicos de Portugal, Lisboa, Tomo 25, 1944, 251 pp.

URANIUM.—Uranium is widespread.—*Min. J.*, Lond., Vol. 225, November 17, 1945, pp. 777-8. 8d.

ZIRCONIUM—USES.—Mariupol zirconium mines (Ukraine). Peter Semenov.—*Min. J.*, Lond., Vol 225, November 24, 1945, p. 797. 8d.

NON-METALLIC MINERALS.

ASBESTOS — TRANSVAAL—BARBERTON. —Sunnyside Asbestos Mining Co., Ltd.: report on new claims; view of consulting engineer. Alan Stuart.—*S. Afr. Min. Engng. J.*, Johannesburg, Vol. 56, Part II, October 27, 1945, pp. 203-7. 6d.

NON-METALLIC MINERALS — *continued.*

BARITE—CONCENTRATION.—The concentration of barite-fluorite ores from the Lake Ainslie District, Nova Scotia. E. Lee Cameron.—*Trans.* Canad. Inst. Min. Metall., Vol. 48, 1945, pp. 567-87. (*Canad. Min. Metall. Bull.*, Montreal, No. 401, September, 1945. $1.00.).

FELSPAR—ECONOMICS.—Post-war feldspar outlook.—*Canad. Min. J.*, Gardenvale, P.Q., Vol. 66, October, 1945, pp. 710-11. 25 cents.

GYPSUM — ECONOMICS.—The growing field for gypsum: advances in recent years. —*S. Afr. Min. Engng. J.*, Johannesburg, Vol. 56, Part II, September 29, 1945, pp. 111-13. 6d.

INDIA — RAJPUTANA.—Mica, beryl and columbite-tantalite in Rajputana. H. Crookshank.—*Imp. Inst. Bull.*, Lond., Vol. 43, No. 3, July-September, 1945, pp. 228-35. 2s. 6d.

MICA — GEOLOGY — U.S.A.—Structural and economic characteristics of New England mica deposits. E. N. Cameron and others. —*Econ. Geol.*, Lancaster, Pa., Vol. 40, No. 6, September-October, 1945, pp. 369-93. $1.00.

SILICA SAND (FOR RAILWAY LOCOMOTIVES).—Engine sands. G. M. Hutt. —*Trans.* Canad. Inst. Min. Metall., Vol. 48, 1945, pp. 631-7. (*Canad. Inst. Min. Metall. Bull.*, Montreal, No. 402, October, 1945. $1.00.)

SULPHUR — U.S.S.R. — TURKMENIA.—Sulphur production in Central Asia: flotation upgrading at desert plant. M. Makarov. (Sulphur works in Kara Kum Desert, Turkmenia.)—*Chem. Tr. J.*, Lond., Vol. 117, November 2, 1945, p. 489. 8d.

PETROLEUM.

CHILE.—El petroleo en Chile.—*Boletin Minero.* Sociedad Nacional de Mineria, Santiago de Chile, Vol. 57, June, 1945, pp. 350-56.

FRANCE.—Le pétrole dans le Sud-Ouest de la France. M. Pierre Angot.—Mémoires de la Société des Ingénieurs Civils de France, Paris, January-February-March, 1945, pp. 81-9.

GREAT BRITAIN.—Petroleum in Great Britain. Arthur Bray.—*Mine and Quarry Engng.*, Lond., Vol. 10, December, 1945, pp. 297-300. 1s.

PLANT AND POWER.

ATOMIC ENERGY.—Nuclear energy technology: what atomic bomb production has meant.—*Chem. Tr. J.*, Lond., Vol. 117, October 26, 1945, pp. 459-62. 8d.

ELECTRIC POWER — MOTORS — CLEANING.—How to clean electric motors, windings and insulation. D. L. Gibson. *Coal Age*, N.Y., Vol. 50, October, 1945, pp. 111-14. 35 cents.

ELECTRIC POWER — PETROLEUM INDUSTRY.—Code of electrical practice for the petroleum industry. Alan D. Maclean —*J. Inst. Petrol.*, Lond., Vol. 31, No. 262, October, 1945, pp. 379-92. 7s. 6d.

POWER CONSUMPTION — COAL MINES. —The economics of power consumption at collierie. B. L. Metcalf.—*Proc. S. Wales Inst. Engrs.*, Cardiff, Vol. 61, November, 1945, pp. 103-29. 7s. 6d.

INDEX OF RECENT ARTICLES—*continued*.

PRECIOUS STONES.

DIAMONDS — CONCENTRATION.—Jigging diamonds in East Africa. H. L. Cornish.—*Deco Trefoil*, Denver Equipment Co., Denver, Colorado, June, 1945, 3 pp.

TIN.

CONCENTRATION.—The dressing of tin ores. (*To be continued.*) F. B. Michell. —*Mine and Quarry Engng.*, Lond., Vol. 10. December, 1945, pp. 289-96 and 300-301. 1s.

CONCENTRATION — AUSTRALIA. — Treatment of tin ore at Irvinebank, Queensland: criticism of Australian practice, high recovery from Queensland plant, wide range of equipment.—*Chem. Engng. Min. Rev.*, Melb., Vol. 37, August, 1945, pp. 345-9. 1s.

TIN—*continued*.

ECONOMICS.—Tin production in the Far East: problems of reconstruction and rehabilitation.—*Iron Coal Tr. Rev.*, Lond., Vol. 151, October 26, 1945, pp. 653-4. 9d.

ECONOMICS — SOUTH AFRICA.—The outlook for tin—possible effects on the Union.—*S. Afr. Min. Engng. J.*, Johannesburg, Vol. 56, Part II, October 6, 1945, pp. 121-3. 6d.

GENERAL.—The white metals—zinc, lead and tin: reprints of extracts from articles appearing in various current periodicals.—*Developments and Trends in American Industries* S-7305, compiled by the U.S. Office of War Information.

PORTUGAL.—Portuguese mineral deposits of cassiterite and wolframite. J. M. Cotelo Neiva. (In Portuguese: summary in English.)—*Comunicacoes* dos Servicos Geologicos de Portugal, Lisboa, Tomo 25, 1944, 251 pp.

Subject to revision.] [*Paper issued on January 10th, 1946, to be submitted for discussion at a Meeting of the Institution of Mining and Metallurgy, to be held at the Apartments of the Geological Society, Burlington House, Piccadilly, London, W. 1, on Thursday, January 17th,* **1946, at 5.30 o'clock p.m.**

The San Telmo Ore-Body, Spain.

By J. C. Allan, *Member.*

Introduction

The San Telmo group of pyrite masses, in the Province of Huelva, Spain, lies at the Western End of what is classified by Collins* as the 'Northern Belt' of ore-bodies. The property is connected with the Val de la Musa Station of the Zafra Huelva Railway by 11 kilometres of narrow-gauge line. The situation of this group in relation to the principal ore-bodies in the area is shown in the accompanying map (Fig. 1).

History.—Within the area of the concessions there are a number of small outcrops. These were first exploited in the early 60's of the last century, and also between 1882 and 1892, by the Bede Metal Company, by heap roasting and leaching. When the reduction in copper content rendered the working no longer profitable the concessions were surrendered to the Spanish owners after all the pronounced 'gossan' in the property had been investigated.

About 1908 an adit started by the Bede Metal Company, but abandoned owing to unfavourable indications, was reopened by the owners. It was extended some 20 metres and struck the mass now known as Santa Barbara 3 metres below its apex. At this point the mineral was high in copper, which led to the gradual development and partial mining of the mass, but operations were suspended owing to the difficulties in marketing the production in 1920.

A feature of the mass, as disclosed by this development work, was the occurrence of important tonnages of zinc-bearing complex ore, which, at that date, had no commercial value. In 1927, however, a plant was erected for the treatment of the complex ore by flotation.

*Collins, H. F. 'The Igneous Rocks of the Province of Huelva and the Genesis of the Pyritic Orebodies'. *Trans.* I.M.M., Vol. 31, 1922.

under a small outcrop to the east of the Santa Barbara surface indications. Whether this second outcrop is connected with the eastern enlargement on the 11th Level has yet to be investigated.

Below the 11th Level, however, the western end of the San Telmo mass suddenly bottomed and was connected, by means of a narrow tongue of sulphide, with another body of pyrite running approximately east and west, but 40 metres farther to the north. The eastern extension on the 11th Floor also appeared to be connected with this third body in a similar way.

Three distinct types of ore occur in the San Telmo mass—namely, a complex ore carrying zinc and barytes ; an ore consisting of cupreous pyrites, in which the copper is present as chalcopyrite ; and a ' sulphur ore ', or clean iron pyrite. The complex ore has a characteristic blocky fracture which is unmistakable and which always indicates the presence of this type of ore. The complete analyses of composites from a number of samples shown in Table I gives the characteristic distribution of the constituents in the three named types of ore.

TABLE I

	Complex Ore	*Primary Cupreous Ore*	*Iron Pyrites*
	%	%	%
Copper	1·254	2·28	0·56
Sulphur	31·530	47·650	51·30
Lead	1·870	0·150	0·04
Zinc	23·970	2·850	0·11
Cobalt	*Trace*	*Trace*	*Trace*
Arsenic	0·090	0·110	0·090
Barium Sulphate	21·680	2·320	*Nil*
Silica	0·370	1·350	0·79
Iron	18·18	41·18	45·20
Alumina	0·92	1·15	0·25
Lime	0·13	0·20	*Trace*
Magnesia	0·06	0·10	0·02
Silver*	120·00	40·00	—
Gold*	0·6	0·5	—

*g per 1,000 kg.

In some cases the contacts between the different types of ore is abrupt, changing on a clean-cut line as in the photographs (Plate I), while in others the types merge into one another, with the banded structure described by many writers on these masses. 'Horses' or remnants of wall rock are found within the boundaries of the zones in which each type is found.

The Complex Ore

The general averages of sampling by levels were as shown in Table II. The photo-micrograph reproduced in Plate II shows a

Table II

	Cu	*Zn*	*Pb*	*Ag*	*Au*	*S*	*Fe**	*Balance†*
	%	%	%	*g per ton*	*g per ton*	%	%	%
7th Level ...	1·2	20·06	2·04	155·40	1·108	32·82	20·68	23·30
6th ,, ...	0·79	20·50	3·29	58·9	0·770	33·00	19·21	23·21
4th ,, ...	0·96	16·30	2·53	90·61	1·070	33·01	24·00	23·20
3rd ,, ...	1·65	19·25	1·89	91·65	0·770	33·49	20·52	23·20
2nd ,, ...	1·73	15·92	2·08	106·78	1·110	32·89	24·18	23·20

*Calculated.
†Mainly Barium Sulphate, as shown in the complete analysis (Table I).

thin section of the complex ore taken under ordinary light and magnified 30 diameters. The black areas are pyrite, the dark grey zinc blende, and the light-coloured areas are mainly barytes, with subordinate amounts of quartz.

From the mill man's point of view the ore presented a difficult problem, the crystalline intergrowth persisting down to very small sizes. Microscope study of the flotation feed showed that, with the possible exception of the cleanest of the complex ore, both blende and galena contain inclusions of pyrite, as blebs and spots, that carry on down in size to the limits of recognition with a magnification of 150 diameters. The major part of the zinc mineral appears under the microscope as a creamy white blende. As the iron content of the ore increases, however, there is an increasing amount of iron-rich blende, probably approaching marmatite in composition. The amount was probably never sufficient to affect the assay of the zinc concentrates, but it showed characteristic fluctuations corresponding to the amount of free pyrite in the mill heads.

The mill operators classified the ore under four broad headings:

(A) *Massive Complex Ore.*—This is a fine-grained ore of a characteristic bluish colour found on all levels. Typical analyses are given in Table III.

TABLE III.

	Cu	*Zn*	*Pb*	*Ag*	*Au*	*S*
	%	%	%	*g* per ton	*g* per ton	%
7th Level	0·67	20·30	0·96	85·71	0·77	34·06
3rd „	1·76	19·32	2·10	95·50	0·77	32·82

(B) *Lead-Bearing Complex Ore.*—This type of ore carries high lead values and is more coarsely crystalline than the massive type. Its principal occurrences were towards the hanging-wall, between the 6th and 7th Levels. Typical analyses of this ore are as set out in Table IV.

TABLE IV

	Cu	*Zu*	*Pb*	*Ag*	*Au*	*S*
	%	%	%	*g* per ton	*g* per ton	%
7th Level	1·60	23·81	9·113	602·40	2·75	26·45
7th „	1·65	15·71	4·400	156·72	1·38	34·33
6th „	0·87	22·58	5·010	68·00	0·77	33·08

(C) *Compact Complex Ore with Segregation of Pyrite.*—The analyses given in Table V do not show an increase in iron content, but the higher sulphur values indicate that the pyrite content has increased at the expense of zinc and barytes.

TABLE V

	Cu	*Zn*	*Pb*	*Ag*	*Au*	*S*
	%	%	%	*g* per ton	*g* per ton	%
Raise—7th/6th Level	1·01	10·73	0·19	52·65	0·77	42·20
4th Level	0·96	16·30	2·53	90·61	1·07	33·01

(D) *Complex Ore Interbanded with Pyrite.*—This type of ore yielded assay results similar to those for type (c) (Table V).

The general relations between complex ore, cupreous ore, and iron pyrites are shown in the sections illustrated in Fig. 2. As already mentioned, contacts between one type of ore and another varied from sharp lines of demarcation, as shown in the photographs (Plate I), through contacts with gouge partings, to banded ore, where the assay, as well as visual inspection, determined which type predominated.

Assays at Complex Ore Contacts

In what follows examples are given of runs of assays taken over consecutive two-metre intervals on complex ore contacts. It is to be regretted that complete analyses of all the important constituents are not available. From a practical point of view, however, the difference between types was so marked that, at the time, analyses were only made of contents of economic importance.

(1) *Clean Contact.*—This is considered to be the case when either cupreous ore or pyrites meets complex ore and the contact is a clean smooth face, with no evidence of striation or movement. Assays are given in Table VI.

TABLE VI

Zn	*Cu*	*S*	*Pb*
%	%	%	%
4th Level: North-West Zone:			
18·70	0·77	30/33*	0·69
22·88	0·77	30/33*	3·12
	(Contact)		
0·91	0·44	47·45	Low values
1·25	1·51	45·38	,, ,,
1·47	1·10	50·90	,, ,,
4th Level: North Zone:			
1·12	2·68	45·37	,, ,,
0·84	1·34	44·10	,, ,,
	(Clean Contact)		
16·24	1·16	30·00*	0·74
7th Level: North-East Zone:			
25·04	1·82	30/33*	8·67
23·86	2·70	30/33*	10·10
21·36	1·44	30/33*	0·48
24·56	0·99	30/33*	0·02
	(Clean Contact)		
Low values	2·02	44·35	Low values
,, ,,	3·10	47·85	,, ,,

*Estimated.

(2) *Clean Contact Between Complex Ore and Cupreous Pyrites or Pyrites, with Gouge or Sterile Parting.*—The walls of the gouge-filled contact are as smooth as those usually found on an intrusive rock contact. Assays are given in Table VII. When, on contact, the

TABLE VII

Zn	*Cu*	*S*	*Pb*
%	%	%	%
4th Level :			
1·87	1·53	49·62	Low values
1·20	0·34	49·51	,, ,,
2·64	0·58	49·52	,, ,,
	(Gouge	Contact)	
11·64	1·14	41·45	,, ,,
3·62	1·10	46·08	,, ,,
7·63	0·08	40·00	1·62
6th Level : North Zone :			
	(Good grade cupreous pyrites)		
	2·67 (Zinc-bearing Pyrites)		
	1·23		
	(Band of Decomposed ' Sterile ')		
	9·63		
	11·69		
	22·24		
	18·09		
	Other values not determined.		

zinc values fade from low-grade complex ore, through zinc-bearing cupreous pyrites, to good-grade pyrites, this transitional ore is locally termed ' aureole ' ore.

Table VIII gives an example of transition from complex to zinc-bearing cupreous pyrites which is not clearly shown by assay, although the pyrites took on the blocky fracture of the complex ore for several metres. In fact, on the 2nd Level, no sharply-defined contacts were found, the whole mass being a complex ore richer or poorer in zinc, or, in other words, containing a varying amount of pyrites.

TABLE VIII

Zn	*Cu*	*S*
%	%	%
13·38	0·87	Not determined
3·18	2·87	43·5
4·63	3·25	45·2
2·45	3·01	42·3
4·50	3·44	44·7

On the 3rd Level, Main Drive, there is also a series of 50 assays in which zinc alone was determined. This zinc value varies from 3·3 per cent to over 20 per cent, but the ore does not change in appearance.

GENESIS OF THE ORE-BODY

The geology of this metalliferous field and the origin of the ore-bodies that occur have received very close study. There is an extensive bibliography, of which much is given in papers by Dr. Gordon Williams and Dr. David Williams.* In brief there appear to-day to be two broad schools of thought concerning the genesis of pyrite ore-bodies of the province of Huelva. One is that the ore-bodies are the result of hydrothermal processes and the other that they were intruded, as differentiates of some parent magma, in liquid or plastic form along pre-existing lines of weakness. By reason of the initial intrusive pressure, or pressures developed on crystallization, they have created the space necessary for their development.

Three methods have been suggested as to how this could have taken place :

(*a*) By progressive segregation and liquation from a parent rock magma resulting in the differential separation of the more fluid sulphides, which, in turn, repeat the sequence until a low enough temperature is reached to freeze the last eutectic.

(*b*) A similar condition to (*a*), but with the magma containing a limited amount of water. Here differentiation by segregation and

*WILLIAMS, G. ' The Genesis of the Perrunal—La Zarza Pyritic Orebody, Spain '. *Trans.* I.M.M., Vol. 42, 1933.

WILLIAMS, D. ' The Geology of The Rio Tinto Mines, Spain '. *Trans.* I.M.M., Vol. 43, 1934.

(2) In certain sections this produced banding of pyrite and unaltered rock. The pyrite replaced bands of rock favourable to its deposition and left unreplaced those less favourable.

(3) A later solution carrying barium, zinc, and lead found zones favourable to the deposition of the sulphate and sulphides and these zones included all the bands of unaltered rock in the zones already banded with pyrite.

Injection of fluid or plastic material seems on the other hand to account for a number of features which are difficult to explain by hydrothermal replacement. Both Gray and Rutherford* emphasize the sharp line of demarcation nearly always found on the boundaries of massive ore and even Dr. David Williams has to admit that the limits of massive sulphide are usually well defined.

In the Santa Barbara mass this clear line of demarcation, as has been shown, is also evident on certain boundaries between the complex ore and the pyrite. This would appear to rule out the possibilities of segregation of true sulphide mattes while in the molten state. The alternative put forward by Evans, Edge,† Dr. Gordon Williams, and others is to postulate a water content of such mattes of the order of 20 per cent. Assuming such a hydropyritic matte and a zinc-lead-copper differentiate as the result, it still remains difficult to reconcile the presence of barium sulphate, as such, in the final freeze out.

To obtain such regularity and homogeneity in the complex ore, a suggested alternative would be to follow Boydell‡ and postulate a condition analagous to a colloidal solution.

Newhouse, in describing the Buchans ore which also carries a high percentage of barytes, writes as follows :

> The fine grained character of the ore may well be the result of deposition by relatively concentrated solutions such as colloidal suspensions which the metallic sulphides form with the alkaline sulphides.

Collins§ in his reply to Simon writes as follows :

> He does not, however, mention that the banded structure of the Rammelsberg lode, whilst invariably parallel to the walls of the deposit, sometimes cuts across the stratification of the slates nor that the banded ore is composed chiefly of small crystalline particles of blende, galena, and chalcopyrite, scattered through a matrix of barytes, which is the principal gangue.

*See the discussion on Collins's paper, *loc. cit.*

†EDGE, A. B. 'Observations on the Pyritic Orebodies of Southern Spain and Portugal'. XIV International Geological Congress, 1926.

‡BOYDELL, H. C. 'The Rôle of Colloidal Solutions in the Formation of Mineral Deposits'. *Trans.* I.M.M., Vol. 34, 1925.

§*loc. cit.*

It will be noted that while the banding is invariably parallel to the walls of the lode, in some places the lode cuts across the stratification of the slate. Thus the banding cannot be considered as pseudomorphic after replaced slate. Simon states that the banded structure at Rammelsberg had been ascribed, after detailed investigations, to movements under great pressure.

Edge, however, pointed out that banded structure can be formed by rhythmic precipitation analagous to the phenomena known as 'Leisegang's rings' and that banded structures in these pyritic masses, parallel to the walls, may have nothing to do with foliation in the original rock alleged to have been replaced. Such banding would be more likely to have been produced under conditions analagous to a colloidal sol. Barium sulphide having formed the sol, as described by Freeman,* may have resulted in the sulphate of barium—barytes. Thus a colloidal ancestry for the complex ore might be deduced from the banded structure, which is a phenomenon known to be associated with colloids, and from the presence of barytes, which, at Buchans, is definitely associated with colloform structure.

Microscopic examination of the Santa Barbara complex ore has not, as yet, disclosed examples of the concentric structure described by Newhouse as being found in the Buchans mine and by Dr. David Williams at Rio Tinto. Newhouse describing this structure writes:

> The presence of pyrite with concentric structure suggests that part of the solutions may have been colloidal.

and Dr. David Williams:

> Although this concentric structure obviously implies a rhythmical precipitation of the sulphide, it does not necessarily signify their colloidal ancestry.

It will be seen, however, that both these authorities admit the possibility of colloidal ancestry.

The processes by which crystal structure can have been developed from colloidal material have been discussed at length by Boydell. Masses of the size of these pyritic deposits would remain for a long period under temperature conditions favourable to the growth of crystal structure. The absence of colloform structure, therefore, does not necessarily signify non-colloidal ancestry. If, however, the possibility of colloidal ancestry be admitted for the complex ore, assuming barium sulphide as the possible sol, the picture is

*FREEMAN, H. 'The Genesis of Sulphide Ores'. *Eng. Min. Jour.*, Vol. 120, No. 25, p. 973, 1925.

less clear in regard to the pyrite. On the other hand, if the banded structure of pyrite and complex ore be due to a condition analagous to a colloidal sol, then, under certain conditions, the pyrite may also have behaved as a colloid, so that the colloform structure noted by Dr. David Williams may have more significance than he has given to it.

There remains, however, the difficulty of explaining the transition from massive pyrite to disseminated ore so strongly emphasized by Collins, Dr. David Williams, and Newhouse. It is generally admitted that massive sulphides and disseminated ore are usually divided by a sharp line of demarcation. Dr. David Williams admits that the regular transition between massive ore and disseminated pyrite is the exception rather than the rule and Newhouse states that only locally is the transition from unaltered tuff to banded ore found in the Buchans mine.

Boydell has discussed at length the possibility of replacement by colloidal solutions and there is no reason to suppose that under exceptional and favourable conditions such replacement may not have taken place to a limited extent, masking the boundary, at those points, between solid sulphide and disseminated ore.

Such a point might well be where the colloidal sol was invading a section of slate, or, as at Buchans, of banded tuff, from a direction roughly parallel to the bedding or shear planes. Colloidal sols are, however, also capable of producing a banded structure owing to entirely different causes and because, under certain conditions, banded structure can be definitely traced to the slate it is not sufficient proof that all banded structure must be so caused.

It is suggested that the shape of the Santa Barbara mass and the distribution in it of the three sulphide types can but be explained by the intrusion of liquid or plastic material under pressure. The complex ore has been found cutting both cupreous and iron pyrites in a manner, to quote Collins in his reply to Broughton Edge, 'practically identical with that of an ordinary eruptive dyke'.

It would seem probable that this intrusion took place in three phases : (1) Pyrite ; (2) complex ore ; and (3) cupreous pyrite, as differentiates from some parent source and that, owing to pressure or other conditions, some re-liquification took place in sections of the mass which permitted marginal absorption between the phases, exhibiting characteristics similar to those shown by colloidal sols.

Conclusions

The structural relations of the complex ore in the Santa Barbara mass point to a mode of genesis other than hydrothermal replacement.

The indications of colloidal ancestry at San Telmo, Buchans, and Rammelsberg, where barytes is the principal gangue mineral, cannot be considered to be without significance.

All authorities agree that the source of the mineralization was ascending magmatic differentiates, but it seems possible that the differentiates ranged from a hydro-pyritic matte rich in sulphides to the more normal hydrothermal solutions. Under the wide range of pressure and temperature conditions that must have obtained in a large metallogenetic field—such as is found in the south of the Iberian Peninsula—an unvarying water-sulphide ratio throughout the whole field is unlikely.

While the hydrothermal mattes were still liquid further differentiation took place as the result of partial crystallization of their sulphide contents.

At one stage in the progressive increase of the water content of the residuum, a condition analagous to colloidal sols existed.

There is no reason to suppose that at this phase metasomatic replacement could not have operated, under local conditions that were favourable, thus masking the normal clear line of demarcation between massive pyrite and the ore of undoubted hydrothermal replacement origin.

The water-pyrite ratio of the differentiates may have varied not only, as suggested by Edge, during the crystallization of the mass, but also during the period when the solutions were emanating from the parent source. This could also occur from point to point in the metallogenetic province.

It is probable that there were no definite boundaries between hydro-pyritic mattes, colloidal sols, and hydrothermal solutions. In a metallogenetic province as large as that under consideration, the conditions were so varied as to give rise to ores deposited from intrusive hydro-pyritic mattes, from colloids, and as replacements by hydrothermal solutions, depending on the prevailing conditions —even though the parent magma throughout the field was substantially the same.

ACKNOWLEDGMENT

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*** *Extra copies of this paper may be obtained at a cost of* 1*s.* 6*d. each, at the office of the Institution, Salisbury House, Finsbury Circus, London, E.C.* 2.

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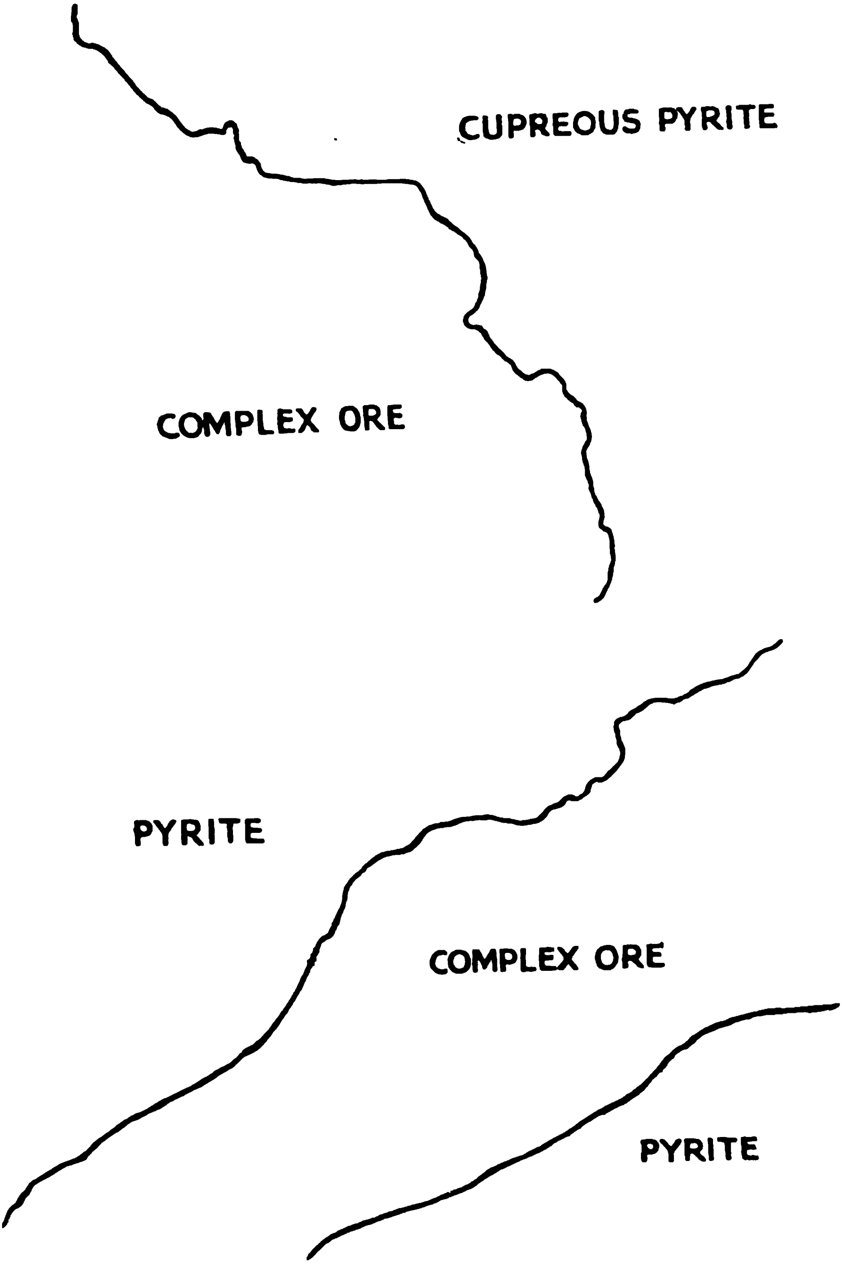

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COMPLEX ORE
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PYRITE

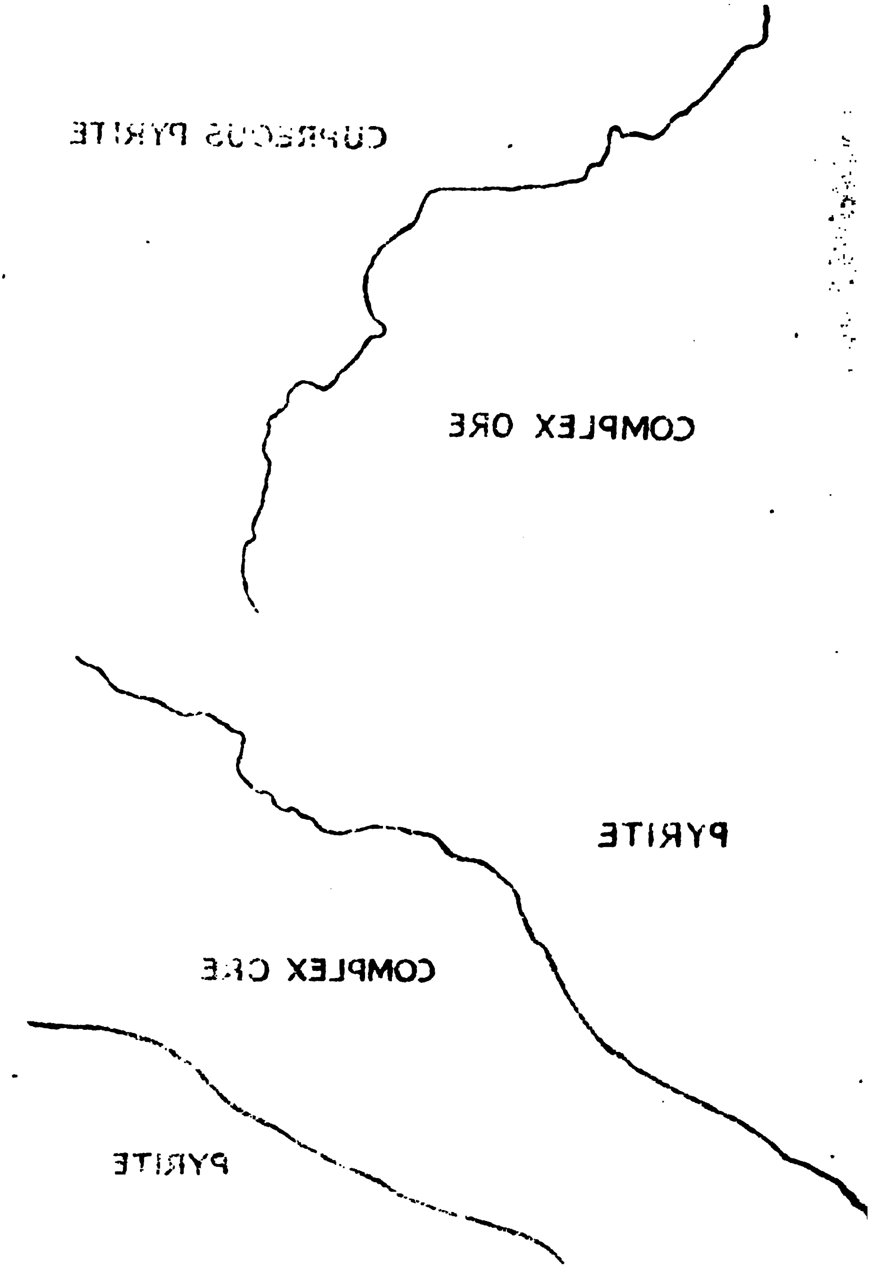

CUPREOUS PYRITE
COMPLEX ORE
PYRITE
COMPLEX ORE
PYRITE

FIG. 3.

FIG. 4.

As a first step it was decided to include fluorspar amongst those materials administered by the Chrome Ore, Magnesite, and Wolfram Control of the Ministry of Supply. That the measures subsequently taken proved to be successful from the war point of view can be seen from the fact that no consumer went in short supply and that the material now sold is of a higher grade than that with which the market was formerly forced to be satisfied.

As a sequel to these investigations and the various other steps taken by the Chrome Ore, Magnesite, and Wolfram Control, the principal producers got together and formed the British Fluorspar Producers Association, a body which can now speak for the industry as a whole, when dealing with grades, specifications, prices, quotas, etc. Table I shows the latest estimates of production and consumption of fluorspar in this country.

Nature of the Product Required

As will be seen from Table I fluorspar is sold under various grades—acid, ceramic, metallurgical grade, etc. Acid grade

TABLE I

APPROXIMATE FLUORSPAR PRODUCTION AND CONSUMPTION DURING 1944.

		Tons
Production		75,000 (all grades)
Consumption :	Steelworks	63,700
	Non-ferrous	3,600
	Chemical	3,500
	Glass and enamels	330
	Sundry users	490
		71,620

represents the purest material and should contain *plus* 98 per cent CaF_2 and under 1 per cent SiO_2, with freedom from sulphur (which causes corrosion in the acid makers' plant). For this material there is at present in this country a market of something of the order of 5,000 tons per annum and the tendency should be for this demand to increase when developments in refrigeration and insecticide products based on fluorine compounds make themselves felt.

Welding, ceramics, and light-alloy technology also call for a fairly high-grade product, around 96 per cent CaF_2 and under 2·5 per cent SiO_2, but the demand is in no way comparable with that for metallurgical-grade material, under which classification about 85 to 90 per cent of the fluorspar production is sold.

Strictly speaking metallurgical spar should contain *plus* 85 per cent CaF_2 and less than 8 per cent SiO_2, but much material of an inferior quality still finds its way to the consumer.

Table II shows the recognized grades for metallurgical spar and the present prices paid.

TABLE II

STANDARD GRADES AND PRICES OF DERBYSHIRE AND DURHAM FLUORSPAR FOR METALLURGICAL USE

DERBYSHIRE PRODUCTION :

Graded Fluorspar :	A	B	C	D	E
CaF_2 (minimum), per cent	80 to 85	75 to 80	70 to 75	65 to 70	60 to 65
SiO_2 (maximum), per cent	8	10	12½	15	15
Price per ton (F.O.R.)......	90s.	72s. 6d.	62s. 6d.	57s. 6d.	52s. 6d.
Ungraded Fluorspar :					
CaF_2 content 60 per cent and under, price F.O.R.					30s.
CaF_2 content over 60 per cent, price F.O.R.					40s.

DURHAM PRODUCTION :

Graded Fluorspar :	1	2	3
CaF_2 (minimum), per cent......	85 to 90	80 to 85	75 to 80
SiO_2 (maximum), per cent......	6 to 8	12 to 14	14 to 16
Price per ton (F.O.R.)	90s.	80s.	65s.
Ungraded Fluorspar :			
CaF_2 content below 75 per cent, price F.O.R.			50s.

(*Ex* British Fluorspar Producers Association.)

RAW MATERIALS

In 1941 some 60 per cent of the production of fluorspar in this country came from the retreatment of old base-metal mine dumps and only 40 per cent from actual fluorspar mining operations. Derbyshire and Durham are the main producing centres and Appendix I shows the scale of these operations.

PRESENT METHODS OF PRODUCTION

As already stated a very large proportion of the fluorspar produced here is obtained from the treatment of old mine dumps, jig tailings, etc., and the extent to which such material is processed before sale varies with the locality and the nature of the material composing the dumps.

Generally speaking, as little as possible in the way of treatment has been done in the past and for this two reasons were operative. Much of the fluorspar trade was in the hands of small producers, who lacked the necessary capital to indulge in what to them was the luxury of expensive washing plants when a market could be found for a low-grade material which could be loaded direct from the dumps, or at most carried the cost of a simple preliminary screening and log-washer treatment. The second reason arose from the lack of interest on the part of some of the steel makers regarding the grade of material supplied and their apparent complacency with existing conditions.

To the steel maker fluorspar has always been regarded as a very mixed blessing. The advantage of its use in increasing slag fluidity on the one hand is offset by the reduction in citrate solubility when basic slag for fertilizer purposes is one of his sources of revenue. As the total cost of fluorspar needed per ton of steel produced is somewhere in the region of 4½d.—i.e., roughly 0·1 per cent of the value of the product—the attitude of the steel maker is to a certain extent understandable, but nevertheless has not been conducive to the development of a healthy fluorspar industry. It was mainly the energetic action of the Fluorspar Control in calling for graded material which focused attention on the necessity for more efficient methods of treatment.

With one or two exceptions log-washing, screening, jigging, and tabling comprise the flow-sheet of the majority of treatment plants operating in this country to-day. Fluorspar, however, although a relatively heavy mineral compared with the accompanying gangue, of which approximately 60 per cent of the dumps are composed, is also an extremely friable mineral and in the course of handling and treatment large quantities of fines are produced. These fines carry a very large proportion of the fluorspar values and consequently flow-sheets based on gravity separation alone are incapable of achieving high recoveries.

Suggestions for treatment by sink-and-float methods of concentration or by table agglomeration have been put forward and have even been successfully employed in the United States to give good recoveries as compared with normal jigging and tabling practice, the former of which especially is more conducive to the production of fines, but only a combination of gravity and flotation, or of flotation alone, will permit of reasonable recoveries being obtained over all size ranges in which the fluorspar occurs in these dumps, which form so large a proportion of the present source of supply.*

*At this point it may be of interest to draw attention to the possibilities which lie in some of these dumps being regarded as a source of both barytes and fluorspar. This applies more particularly to certain dumps situated in the Eyam district of Derbyshire, where operations for recovery of the barytes by jigging have been conducted for some time past. Tests made on this material showed that quite a good grade of barytes could be separated from the fluorspar in the flotation circuit and as much of the barytes is of the amorphous quality it would, after bleaching if found necessary, find a ready market in the paint trade.

Realizing this, the examination of the various sources of supply then being conducted by Dr. Dunham and Mr. Jackson included a comprehensive series of tests made with the purpose of determining the general flotation characteristics of the spar from various localities. The tests were carried out by one of the Non-ferrous Mineral Development Control's metallurgists (Mr. L. A. Wood), and Appendix II shows the results he obtained and the conclusions drawn therefrom.

From these results it will be seen that Mr. Wood considered the Weardale spar to be more amenable to flotation treatment for the production of a high-grade product than that produced in Derbyshire, since the latter material contains appreciable quantities of calcite and barytes, both of which are difficult to remove to the degree necessary for the production of a really first-class acid-grade spar. As the main impurity in the Weardale material was found to be silica, which was easily depressed in the flotation circuit, better recoveries also were possible when treating the Weardale spar for the production of an acid-grade product.

Proposal for New Mill

In 1943 a deficiency of 5,000 to 6,000 tons of high-grade spar for acid makers, light alloy, and ceramic work was anticipated by the Fluorspar Control and to meet this demand it was decided to erect a new mill.

As a mill based on such a limited tonnage was considered to be an uneconomic unit it was decided to instal a plant of larger capacity with a section capable of producing the high-grade product required, whilst the main production could be marketed as a good quality metallurgical spar, for which there was also an increasing demand.*

As the suitable ore reserves of Weardale were considered by geological opinion to exceed those of Derbyshire and as the preliminary test work had indicated the advantage they possessed in ease of treatment and higher recovery it was decided to instal the new mill in the former district, where it could be fed from the ore mined in Stotfieldburn mine, Stanhopeburn mine, and Hope Level mine.

Detailed test work on ore from these three mines was therefore authorized and the results obtained from Stotfieldburn material are shown in Appendix III.

*A preliminary estimate indicated that the cost of milling should not exceed 35s. per ton of finished product, a portion of which might, however, require to be briquetted at a cost around 7s. 6d. per ton.

Flow-Sheet.—The flow-sheet shown in Appendix V (Plate 1) was only decided upon after careful study of these test results and discussion between the writer and the engineers in charge of the work. The capacity of the plant at 100 tons per day was decided upon as being in the circumstances the most economical unit both as regards capital outlay and working costs.

At this capacity it was estimated that the output would be from 30 to 35 tons of metallurgical and 25 to 30 tons of acid-grade spar and about one ton of 80 per cent lead concentrates per day. It was recognized that a portion of the acid-grade product might not necessarily conform to the rigid acid specification and would be available for ceramic or metallurgical service, where fineness is not an objection.

This flexibility in the flow-sheet was deemed necessary in order that the production ratio of the products could always be kept in line with market requirements, whilst at the same time rendering permissible a certain degree of latitude in the nature of the crude mill feed. The last point was of some importance as (although geological opinion was quite satisfied on the ore reserves) Stanhope-burn mine was not fully developed and the test work could only be done on samples which, especially in their silica content, might not be fully representative of future development.

Site.—A small area of waste ground belonging to the Ecclesiastical Commission situated about half a mile north of Stanhope, adjacent to a railway siding, and in close proximity to a good road and a power cable, was selected and surveyed by the engineers entrusted with the design of the mill.

Mill Design.—The mill was designed by the British Geco Engineering Co., Ltd., in consultation with engineers and consultants of the Control and the general layout is shown in Fig. 1 (Plate I, Appendix V). As the basis of the operation was to be of a customs nature—the various producers, however, being paid outright for the CaF_2 content of the material delivered to the mill—storage and sampling equipment was given careful consideration and the layout finally agreed upon seemed to give promise of most success.

As will be seen from the analysis in Appendix III the ore from Stotfieldburn mine was found to be the most suitable for the production of a low-silica acid-grade concentrate and for this reason it was proposed to store and treat it separately. The ore received from other sources would be weighed, sampled, and used as a composite feed for the production of a jig concentrate of metallurgical grade and a flotation concentrate of ceramic grade, part of which could, if necessary, after briquetting, be sold as metallurgical-grade spar.

When treating Stotfieldburn ore alone for the production of a high-grade acid-spar flotation concentrate the mill capacity was rated at 65 tons of throughput per day (24 hours), from which it was estimated that 40 to 45 tons of good-quality acid-grade concentrates would be produced.

The type of dryer to be used for drying these acid-grade concentrates received careful consideration and a final decision was made in favour of an indirectly-heated rotary type, although the mill designers were impressed with the possibilities and thermal efficiency of the Kestner direct-contact apparatus for drying material in a fine state of division—such as, flotation concentrates.

Although no provision is shown in the mill plan for the briquetting of the metallurgical-grade flotation concentrates, comprehensive tests were conducted which showed that no difficulty would be experienced in obtaining a good-class briquette using under 5 per cent pitch as the binder. A sample lot of briquettes was despatched to a large steel maker and tested on a furnace charge and a favourable report on their efficiency was received. The suitability of a briquette having a carbon content, due to the pitch binder, for use in electric furnaces engaged on high-class special steel production raised certain doubts in the mind of the author, but it was felt that a sintering or agglomeration method of treatment of the concentrates could be adopted if necessary.

A summary of the estimated cost of the mill, excluding the briquetting section, is shown in Appendix IV.

Although, as has been shown, a considerable amount of work had been done in preparing this scheme for a central customs mill in Durham, the plant was never erected. Owing to the efforts of the producers, particularly in Derbyshire, the threatened deficiency in supplies of both metallurgical- and acid-grade spar had been averted and priority for construction of the new mill was considered unjustified in the face of urgent demands on labour and plant for more pressing war needs. It is felt, however, that the work done and the information obtained must have a post-war value to those interested in fluorspar production, particularly to those operating in the district in question. In the hope that it may prove so lies the justification for submitting this paper.

Thanks are due to the Raw Materials Department of the Ministry of Supply, and the Controller of the Non-ferrous Mineral Development Control for *permission* to publish; also to the

managements of the mines mentioned for their valuable collaboration. and to the officers of the Geological Survey and Controls whose work has been so freely quoted.

APPENDIX I

PRODUCTION OF FLUORSPAR IN DURHAM AND DERBYSHIRE

It was estimated in 1941–42 that the producing properties in Durham and Derbyshire had proved and probable reserves equivalent to 920,000 tons of 100 per cent CaF_2.

In 1941 about 350 men were directly employed in the production of fluorspar in these two areas, and of these 135 were employed on mining, 100 on washing and milling, and the remainder on reclamation from the dumps.

The production figures, which represent spar used for metallurgical purposes only, were as under :—

Source	*Monthly Output, Tons*
DURHAM	
Mines	1,830
Dumps	50
DERBYSHIRE	
Mines	360
Dumps	3,110

Cumberland and Yorkshire produced another 275 tons per month. equally divided between dump and mining operations.

It will be seen that about 60 per cent of the production in 1941 came from dumps. Since then, however, mining activities have increased and the quantity of mined ore and dump material are approximately the same.

It has been considered that as a result of further development the probable reserves in mines in Derbyshire has been substantially increased during the past two years.

APPENDIX II

PRELIMINARY TESTS ON CONCENTRATION BY FROTH FLOTATION

In this investigation 15 samples in all from Weardale and Derbyshire were examined with the object of determining the most suitable source of supply for acid-grade spar.

Four examples of the procedure adopted in testing these samples are given, together with a summary of results obtained from all the samples tested by Mr. Wood in this preliminary investigation. Extracts from laboratory reports are quoted hereunder.

Preliminary Preparation of the Ore.—Preliminary preparation, except where expressly described, consisted in reducing the size of the material by dry crushing to pass a screen of 1/10 in. square aperture and grinding wet in the laboratory rod-mill to about

100-mesh. The screen analysis given in the report on Stotfieldburn No. 1 is approximately representative for the series except where natural accumulated slimes were treated.

EXAMPLE (1)

Stotfieldburn No. 1, 15 Fm. Level Forehead.

This sample (L.A.W.6) assayed 78·89 per cent CaF_2; 7·47 per cent SiO_2; 5·54 per cent Pb. The lead was present as galena; with it occurred a very small amount of zinc blende and some siderite. The degree of grinding adopted for testing was as shown in Table III.

TABLE III

Screen Size Tyler Standard		*Distribution*	
On	*Thro'*	*Direct*	*Cumulative*
		%	%
100	—	1·7	1·7
150	100	8·9	10·6
200	150	17·3	27·9
—	200	72·1	100·0

Separation of Sulphides.—The sulphides floated readily and were removed with :—

Cresol = 0·25 lb. per ton of ore
Potassium Ethyl Xanthate = 0·10 „ „ „

No depressant was used and no conditioning period required. A total of six minutes frothing time was ample for complete extraction. The concentrates from six charges were mixed together and retreated with a trace of cresol only, yielding the separation shown in Table IV. It is considered that substantially all the sulphides will leave the primary machine in the form of a 70 per cent Pb. product or better. It may or may not be necessary to retreat this with the use of depressants to get a smeltable grade concentrate.

TABLE IV

Reference, L.A.W.	*Product*		*Assay*					*% Distribution of Lead*
	Type	*% Wt. of Crude Ore*	*Pb*	*Zn*	*Fe*	CaF_2	SiO_2	
			%	%	%	%	%	
4 + 5	Primary Concs.	10·94	38·2	—	—	43·61	—	100
4	Final Concs. ...	5·66	70·02	1·83	1·4	5·13	0·36	94·8
5	Midds.	5·28	4·61	0·78	—	80·32	—	—

Separation of Fluorite.—Some degree of fluorite concentration was obtained by using oleic acid and an aerating agent (pine oil or other non-acid frother) together with sodium silicate, but it was found that as the proportion of silicate was increased the froth became very fragile and recovery fell away. If the *p*H of the circuit was kept between 7 and 8 the fluorite floated well. More silicate could be employed and the silica more effectively depressed. The following test was chosen as indicative :—

Conditions :—

Conditioned 15 min. with :—

Sodium Silicate	2·5 lb. per ton of ore	*p*H = 7·2
Sulphuric Acid	0·8 lb. „ „ „	
Oleic Acid	0·7 lb. „ „ „	
Cresol	0·3 lb. „ „ „	

The primary concentrates were retreated in similar circuit. Results are given in Table V. More fluorite could be extracted from the residues of this test, an overall recovery of 90 per cent of the fluorite being considered quite possible. The froth was almost perfectly white when wet and separated rapidly at a rate similar to that of a normal sulphide ore.

TABLE V

Ref. No. L.A.W.	*Product*		*Assay*		*% Distribution*	
	Type	*% Wt. of Original*	CaF_2	SiO_2	CaF_2	SiO_2
			%	%		
1	Concentrate ...	72·43	99·53	0·03	84·7	5·81
2	Midds............	4·75	73·7	13·50	4·1	6·28
3	Residues.........	22·82	41·67	38·92	11·2	87·91
	Original	100·00	85·1	10·2	100·0	100·00
	Mixed 1 and 2	77·18	98·2	1·59	88·8	12·09

(Assays by Imperial Institute.)

Note.—For the sake of uniformity in comparing with other ores, yield and recovery are calculated on the sulphide-free product—i.e., the total of the products from the fluorite separation.

EXAMPLE (2)

Hope Level Washed.—This sample assayed 81·21 per cent CaF_2 ; 15·54 per cent SiO_2 ; 0·41 per cent Pb, and contained no zinc. In general characteristics it resembled Stotfieldburn Samples 1 and 2.

Separation of Sulphides.—The small quantity of lead present was first removed with cresol and potassium ethyl xanthate.

Three charges of 600 gr. each yielded 100 gr. of concentrates assaying 8·16 per cent Pb. The quantity was too small for retreatment and the question of reconcentration was left over as being of minor importance at present.

Separation of Fluorite.—By the use of neutralized sodium silicate to the extent of 3·6 lb. per ton of ore, as with Stotfieldburn No. 2, a concentrate was obtained which weighed 63·6 per cent of the original and which assayed 99·58 per cent CaF_2: 0·19 per cent SiO_2. It contained 74·6 per cent of the fluorite originally present. The primary residues carried too much fluorite, however, and therefore the following test was chosen as more indicative of possibilities. The charge of 600 gr. was in this case ground for 8 min. in place of the usual 5 min.

Primary Separation:

Conditioned 15 min. with :—

Sodium Silicate	1·7 lb. per ton of ore
Potassium Bichromate	0·37 „ „ „
Oleic Acid	0·7 „ „ „
Cresol	0·3 „ „ „

First Retreatment:

Sodium Silicate	0·9 lb. per ton of ore
Potassium Bichromate	0·2 „ „ „
Cresol	0·1 „ „ „

The concentrates appeared fairly good in the drained froth, but the total froth as separated was not good enough.

Second Retreatment:

Sodium Silicate	1·7 lb. per ton of ore
Potassium Bichromate	0·4 „ „ „
Cresol	0·1 „ „ „

The results are given in Table VI. The finished concentrate was white when wet.

TABLE VI

Ref. No. L.A.W.	*Product*		*Assay*		*% Distribution*	
	Type	*% Wt. of Original*	*CaF_2*	*SiO_2*	*CaF_2*	*SiO_2*
			%	%		
22	Concentrates ...	74·0	99·19	0·54	89·59	2·50
23	2nd Midds.......	6·9	83·64	11·62	7·08	5·03
24	1st Midds.	4·5	40·62	51·31	2·23	14·42
25	Residues.........	14·6	6·17	85·45	1·10	78·05
	Original	100·0	82·0	16·0	100·0	100·0
	22 and 23	80·9	97·86	1·49	96·67	7·53

(Assays by Michie & Davidson.)

EXAMPLE (3)

Stanhopeburn, General.—This was a good white fluorspar containing a little lead. It assayed (L.A.W.41) 84·15 per cent CaF_2; 9·9 per cent SiO_2; 0·55 per cent Pb; 0·25 per cent Zn. A trace of pyrites was also present.

Separation of Sulphides.—The sulphides were removed from one charge only. The concentrate from 600 gr. weighed 42 gr. and assayed 5·59 per cent Pb; 1·65 per cent Zn; 1·81 per cent Fe. The primary concentrates obtained in practice would be far richer, but would need differential retreatment.

Separation of Fluorspar.—This material reacted in a very similar way to Stotfieldburn No. 1. As a variant on the Stotfield conditions the following test was made conditioned 20 min. with :—

Sodium Silicate	1·7 lb.	per ton	of ore	
Potassium Bichromate	0·4	,,	,,	,,
Oleic Acid	0·7	,,	,,	,,
Cresol	0·3	,,	,,	,,

Retreated froth in similar circuit (less oleic acid); results in Table VII.

TABLE VII

Ref. No. L.A.W.	*Product*		*Assay*		*% Distribution*	
	Type	*% Wt. of Original*	*CaF_2*	*SiO_2*	*CaF_2*	*SiO_2*
			%	%		
37	Concentrates ...	73·8	98·15	0·86	80·96	6·7
38 & 39	Midds............	11·5	76·83	21·63	9·91	26·3
40	Residues.........	14·7	55·44	43·02	9·13	67·1
	Original	100·0	89·4	9·5	100·0	100·0

(Assays by Michie & Davidson.)

EXAMPLE (4)

Masson Mines—Average.—No lead was present in this sample, which assayed (L.A.W. 57) 68·0 per cent CaF_2; 12·9 per cent SiO_2; 4 per cent $BaSO_4$; 7·2 per cent Fe. It was coloured strongly brown with limonite and carried calcite and a clay-like slime. This was more troublesome than the washed material. The reagents used for the washed material did not eliminate all the limonite, but when the final froth from this treatment was again retreated with silicate and bichromate a good-grade concentrate was obtained. It was then found that silicate and bichromate could be applied for elimination of limonite by first removing colloidal slimes. This treatment still left in the product a proportion of barytes which could, however, be removed with quebracho. The following test is illustrative :—

Removal of Colloidal Slime.—Pulp diluted to 6 water : 1 ore and deflocculated with :—

Sodium Carbonate	1·5 lb. per ton of ore
Sodium Silicate	0·9 „ „ „

The pulp was allowed to settle at the rate of 1 in. in three minutes and the slimes siphoned off. It is to be inferred that these slimes consisted largely of clay, as they contained only 10·6 per cent CaF_2.

Separation of Fluorite.—In pulp of 3 water : 1 ore the charge was conditioned for 15 min. with :—

Sodium Silicate	1·7 lb. per ton of ore
Potassium Bichromate	0·4 „ „ „
Oleic Acid	0·8 „ „ „
Pine Oil	0·15 „ „ „

A further 0·6 lb. per ton of oleic was necessary during separation of the froth. Most of the limonite, calcite, and quartz was depressed, but barytes floated strongly. The froth was retreated with :—

Sodium Carbonate	3·0 lb. per ton of ore
Quebracho	0·1 „ „ „
Cresol	0·3 „ „ „

and again retreated with :—

Sodium Carbonate	3·0 lb. per ton of ore
Quebracho	0·05 „ „ „
Cresol	0·3 „ „ „

The froth was pale buff coloured when wet and off-white when dry. The results are set out in Table VIII.

TABLE VIII

Ref. No. L.A.W.	Product		Assay				% Distribution CaF_2
	Type	% Wt. of Crude	CaF_2	SiO_2	$BaSO_4$	Fe	
			%	%	%	%	
52	Slimes	5·2	10·6	—	—	—	1·3
53	Concentrates	37·1	99·2	0·5	0·05	—	54·9
54	2nd Midds	19·0	83·9	3·8	—	—	23·8
55	1st Midds	13·0	37·4	25·6	12·8	13·3	7·3
56	Residues	22·7	37·7	—	—	—	12·7
57	Crud	100·0	68·0	12·9	4·0	7·2	100·0
	53 and 54	56·1	94·1	1·62	—	—	78·7

(Assays by Dunford, Smith & Moore.)

It will be noted that by cutting out the final retreatment a very good metallurgical-grade spar is made and there is a very fair chance that this type of product could be made from one treatment. The product for this purpose would need briquetting. From Table IX it will be seen that the yield per ton treated is definitely in favour of the Weardale spars. Combining the two factors of treatability and yield it is probable that the cost of treatment per ton of finished acid-grade spar would be twice as much in Derbyshire as in Weardale.

TABLE IX

SUMMARY OF RECOVERIES AND YIELDS OBTAINED IN FINAL PRODUCTS

Sample	*Assay*		*Product*			% *Recovery of* CaF_2
				Assay		
	CaF_2	SiO_2	*Yield*	CaF_2	SiO_2	
	%	%	%	%	%	
Stotfield 1	85·1	10·2	72·4	99·53	0·03	84·7
,, 2	83·9	12·1	74·5	99·18	Nil	88·1
,, 3	76·6	18·9	73·4	97·79	0·99	92·0
Hope 1	82·0	16·0	74·0	99·19	0·54	89·6
,, 2	79·4	17·4	73·3	98·22	0·85	90·7
Stanhopeburn......	89·4	9·5	73·8	98·15	0·86	81·0
Masson Washed...	78·3	—	51·5	99·1	0·60	65·2
,, Averaged	68·0	12·9	37·1	99·2	0·50	54·9
,, Slimes ...	61·6	—	42·8	98·8	0·9	68·6
Glebe	73·5	14·9	51·3	96·9	1·76	67·6
Cupola...............	39·6	—	36·3	97·2	0·7	89·3
Tideslow............	66·8	9·4	58·1	99·2	0·1	86·2

It must be concluded, therefore, on the basis of the samples actually tested, that Weardale is the most promising district for the production of acid spar.

APPENDIX III

FLOTATION CONCENTRATION OF STOTFIELDBURN FLUORSPAR SAMPLES 1 AND 2

These tests were conducted with the purpose of developing a satisfactory flow-sheet for the production of high-grade acid spar. The procedure adopted in the tests and the conclusions reached are shown hereunder, as extracts from the laboratory reports.

Description of Ore.—The material employed in these tests—i.e., ore—consisted of a mixture of equal parts of two samples submitted by Dr. Dunham in February, 1942, and labelled respectively :—Stotfieldburn No. 1, 15 Fathom Level forehead ;

Stotfieldburn No. 2, East Stope. Preliminary tests in these samples were made as part of a review of the possibility of making acid spar from several ores and were reported on May 16th, 1942.

Both samples consisted of fluorite carrying silica in fine vein-like cracks. Galena, together with a small amount of blende and pyrites, occurred as fairly large crystals in the fluorite. Some siderite and some calcite were also present. Analysis of a representative sample made by H.M. Government Laboratories showed the presence of :—74·5 per cent CaF_2 ; 10·6 per cent SiO_2 ; 3·5 per cent CaO : 5·6 per cent Pb : 0·3 per cent Zn : 0·6 per cent Fe : traces of $BaSO_4$.

Preliminary Tests.—Preliminary tests consisted in trying the effect of employing the following circuit reagents with oleic acid for the separation of fluorite :—

(1) Sodium silicate in various strengths.

(2) Sodium silicate and potassium bichromate in various strengths.

(3) Sodium carbonate to *p*H 10 to 10·5 *plus* quebracho in various quantities.

(4) Sodium silicate neutralized to *p*H 7 to 8 with sulphuric acid.

It was found that the best yield of high-grade concentrates was made with sodium silicate neutralized with acid. The product was not quite so white as those in alkaline circuit by the other reagents, but the silica was eliminated with far less loss of fluorite.

SECTION B : TEST ON 10,000 GR. OF ORE IN DENVER NO. 7 MACHINE

Preparation.—Ore was crushed dry to pass a screen of 1/10 in. square mesh. Ten charges of 1,000 gr. of *minus* 1/10 in. material with 500 cc. of water were ground for ten minutes in the laboratory rod-mill to the degree (about 65-mesh) indicated by the screen analyses of the concentrates and residues, each made in 200 grs. after 15 min. on the Ro-Tap. (Table X.) The full charge was diluted to a total of 30,000 cc. of water in the Denver machine.

TABLE X

Mesh Tyler Standard		*Residues % Wt. of Fraction*		*Concentrates % Wt. of Fraction*	
On	*Through*	*Direct*	*Cumulative*	*Direct*	*Cumulative*
65	—	1·0	1·0	0·28	0·28
100	65	5·3	6·3	2·44	2·72
200	100	39·3	45·6	32·50	35·22
—	200	54·4	100·0	64·78	100·00

Separation of Lead.—Pulp was conditioned for 1 min. with the addition of :—

Cresylic Acid, 0·5 cc. = 0·1 lb. per ton of ore.
Potass. Ethyl Xanthate, 1·5 cc. of 10% Soln. = 0·03 per ton of ore.

Separation of lead froth was complete in 5 min.

The primary concentrates were retreated in a smaller machine with the addition of a trace of cresol. Separation was as shown in Table XI. The final concentrates represent, roughly, the grade to be expected as primary concentrates in continuous practice. They require retreatment. It will be remembered that those obtained in a series test assayed 79·19 per cent Pb. from Stotfield No. 2 and 70·02 per cent Pb. from Stotfield No. 1. As no depressants were employed in these tests the prospects of making high-grade concentrates are good.

TABLE XI

RETREATMENT OF PRIMARY LEAD CONCENTRATES

Mark, L.A.W.	Product		Assay					% Distribution	
	Type	Weight, gr.	Pb	CaF_2	SiO_2	Zn	Fe	Pb	CaF_2
			%	%	%	%	%		
170	Concentrates	824	69·9	10·8	2·8	2·9	0·6	96·0	50·0
171	Middlings......	158	15·2	56·4	4·8	0·8	2·6	4·0	50·0
170 and 171	Primary Concs.	982	58·8	17·5	3·01	2·4	1·87	100·0	100·0

Separation of Fluorite.—After separation of the lead product the pulp in the machine was conditioned with :—

Sodium Silicate, previously neutralized to pH 7–8 with Sulphuric Acid ... } 24 gr. = 4·8 lb./ton ore
Oleic Acid ... ... 5 cc. = 1·0 „ „ „

It was intended to condition for 10 min., but after 2 min. the froth gathered freely and overflowed. Aeration was therefore recommenced and froth removed with addition to circuit as make-up liquor, a solution containing neutralized sodium silicate of strength 0·8 gr. of water-glass per litre of water ; 15 min. was ample for this separation.

The residues were removed, settled, filtered, and dried. They weighed 610 gr. and assayed 11·3 per cent CaF_2; 82·0 per cent SiO_2.

The concentrates were returned for retreatment with addition of make-up solution only (0·8 gr. of neutralized sodium silicate per litre). Separation was again rapid, 10 min. being sufficient.

First middlings were then removed from the machine, settled, filtered, and dried. They weighed 590 gr. and assayed 7·8 per cent CaF_2 ; 79·4 per cent SiO_2.

The concentrates, now assaying 97·05 per cent CaF_2 and 1·72 per cent SiO_2 were again returned to the machine and retreated without addition except for make-up liquor (usual strength).

The second middlings were then removed from the machine. They weighed 762 gr. and assayed 79·6 per cent CaF_2 ; 14·35 per cent SiO_2.

The final concentrates were filtered and dried yielding 6,750 gr. of dry concentrates assaying 99·0 per cent CaF_2, 0·3 per cent SiO_2, 0·4 per cent CaO, and traces only of zinc, lead, and iron, together with a total of 6,925 gr. of water (W/S ratio = 1·02/1).

The second middlings were twice retreated in a smaller machine, using the same circuit liquor and with two drops of oleic acid (about 0·3 lb. per ton of middlings) added at the first retreatment. The second retreatment had little effect.

The total retreatment yielded 632 gr. of concentrates, assaying 93·6 per cent CaF_2 ; 4·6 per cent SiO_2 ; 1·0 per cent CaO ; traces of Pb; Fe; Zn; and 130 gr. of residues assaying 12·3 per cent CaF_2 ; 61·6 per cent SiO_2.

In the second-grade concentrates thus obtained, the silica was present largely in the form of compound grain of silica attached to fluorspar, the proportion of silica in some of these grains being quite large.

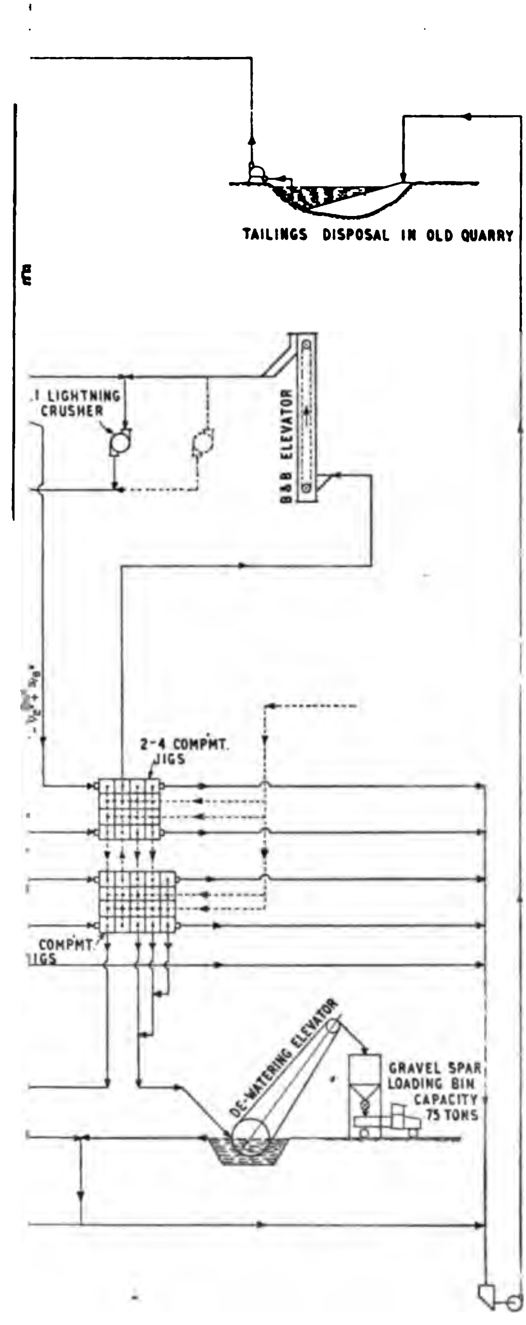
In the first-grade concentrates nearly all the silica was present in compound grains, the proportion of silica in the grains being mostly quite small.

The weight, proportion, assay and proportionate main mineral contents of each of the various products made are tabulated for easy reference.

_ *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.*

CROWTHER & GOODMAN (*Branch of the Argus Press Ltd.*), 2, Carmelite Street, London, E.C. 4.

Plate I.


TAILINGS DISPOSAL IN OLD QUARRY
.1 LIGHTNING CRUSHER
B&B ELEVATOR
2-4 COMPMT. JIGS
COMPMT. JIGS
DE-WATERING ELEVATOR
GRAVEL SPAR LOADING BIN CAPACITY 75 TONS

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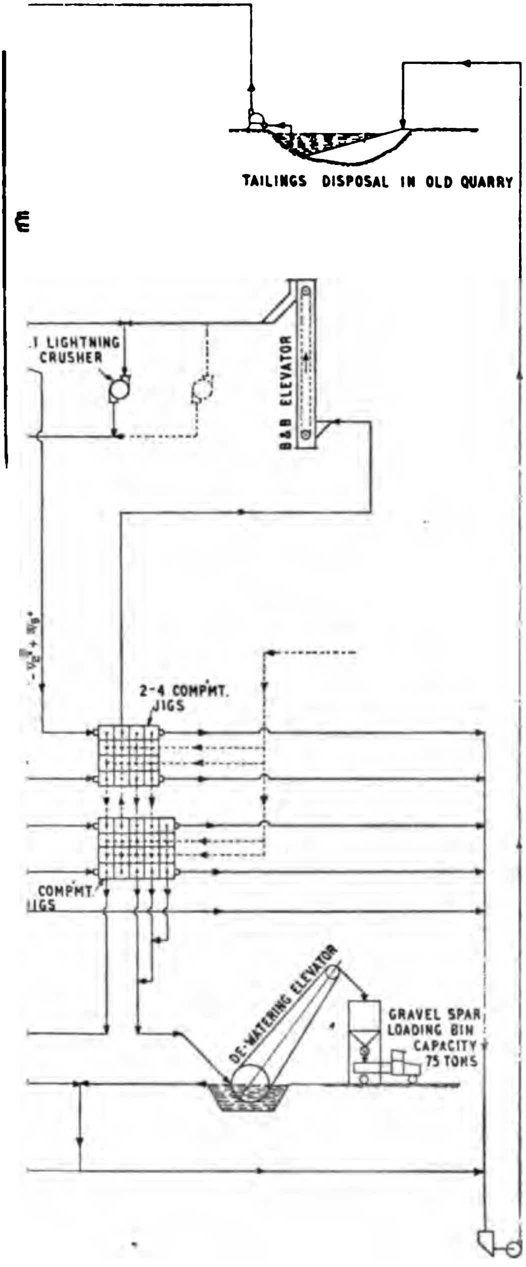
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THE INSTITUTION OF MINING AND METALLURGY

THIRD ORDINARY GENERAL MEETING

OF THE

FIFTY-FIFTH SESSION

Held in the Rooms of the Geological Society, Burlington House, Piccadilly, W. 1,

ON

Thursday, November 15th, 1945,

Mr. G. F. Laycock, *Vice-President,* in the Chair.

DISCUSSION

ON

Tunnelling in Gibraltar during the 1939–1945 War.

By W. H. Wilson, *Member.*

The Chairman said that the first paper to be submitted for discussion was one on Tunnelling in Gibraltar, by Major W. H. Wilson. It was unfortunate that the President was unable, owing to continued ill health, to preside at the meeting. as he was an old Tunneller himself and would be most disappointed not to be present. A telegram had been received from him expressing his deep regret. However, as his deputy it was a happy coincidence that he, the Chairman, also happened to be an old Tunneller and he was very pleased to see such a large audience—particularly of Tunnellers.

It would not be out of place to mention the part played by the Institution in connection with the formation of Tunnelling Companies in the early part of the war. In October, 1939, the Institution was informed by the War Office that it was the intention to form a Tunnelling Company immediately along the lines of those employed in the war of 1914–18 and asking for the names of members who had had tunnelling experience in the previous world war and were available for service. As a result of the information the Institution was able to supply certain members, almost in the matter of a few hours and rather to their surprise, found themselves in uniform again and with commissions in the Royal Engineers, or transferred to that Corps from other branches of the Services. Almost all the officers of the first Tunnelling Company which was formed were members of the Institution, and as other Companies were created the services of more members were called upon. In all about 225 members served in Engineering units of the Imperial Forces in different parts of the world. Of these 138 held Commissions in the Corps of the Royal Engineers, many of them in the Tunnelling Companies, where they served in every rank from the lowest to the highest—namely, from 2nd Lieutenant to C.R.E. of a Group. In addition some others were co-opted as members of the R.E. & S. Board, and a small sub-committee of the Council was appointed to deal with matters brought before the Institution by the Army authorities at various times.

The Institution has always taken an almost fatherly interest in everything connected with Tunnelling Companies and the Council was only too delighted to have this paper submitted for discussion. Although at times in fairly close touch with some of the activities of the Tunnelling Companies he did not realize that they were engaged upon such large-scale underground excavations as were described in the paper. It was very different work from that carried out in France and Flanders in the previous world war when most Tunnellers were engaged in sinking shafts and driving small galleries under very trying conditions, and also in dealing with delayed action mines, booby traps, etc. This was very tricky work, and a large number of men were lost in the process. If the present paper was compared with the paper by Major Standish Ball, read in 1919, the tremendous change that had taken place in the kind of work done by Tunnelling Companies would be appreciated. Modern mining methods could now be more fully employed and Tunnelling had taken on an entirely new aspect. There was a great deal of valuable information given in the present paper as to the methods used in excavating the rock and handling the broken material, and he felt sure that many members would have questions to ask and also he hoped some constructive criticisms to offer.

There were many distinguished guests present, including Lieutenant-General Sir Noel Mason MacFarlane, former Governor of Gibraltar, General Sir Guy Williams, an old friend of the Tunnelling Companies in the last war, Brigadier E. J. B. Buchanan, Director of Fortifications and Works, Colonel H. M. Fordham, former Chief Engineer of Gibraltar, and Brigadier T. W. R.

Haycraft, also a former Chief Engineer of Gibraltar. From the Government came Mr. Cook, Deputy Chief Inspector of Mines. The Institution was honoured to welcome the guests and he invited them to take part in the discussion. Major Wilson, the author of the paper, was also present and would be able to introduce his paper.

Major W. H. Wilson said that the first suggestion that he should prepare a paper on the activities of the mining profession in the Royal Engineers was made to him in 1941 by the late Mr. E. H. Clifford, then President of the Institution. The opportunity of putting this suggestion into practice came later in Gibraltar and he was glad to take it, thus to provide a record of the work of certain of the Tunnelling Companies during the war. This account was quite properly presented to the Institution, since most of the mining engineers who had participated in directing that work were members.

The Chief Engineers, Gibraltar, during the war-time years were, in succession, Brigadier H. M. Fordham, Brigadier T. W. R. Haycraft, Brigadier E. N. Clifton, Brigadier W. G. R. Nutt, and Colonel R. Colville—all regular R.E.'s. Their technical advisers in tunnelling matters, on the Royal Engineers side were, in order, Lieutenant-Colonel D. M. Thomson, Lieutenant-Colonel A. R. O. Williams, Major F. N. Spettigue, and the author, who also had command of a Company. Colonel, now Brigadier, Colin Campbell, and Lt.-Col. C. B. North and Major Tatham, commanded the Royal Canadian Engineers on The Rock. Major G. A. P. Moorhead, Major E. F. T. Maunsell and Major D. H. Williams commanded companies. The total number of officers concerned was large and he hoped he would be forgiven for not naming them all. Nor could anything but a passing mention of gratitude be given to the very large number of N.C.O.'s and men who had given years of hard work to the underground projects on The Rock.

While the great bulk of the tonnage had been mined by the Royal Engineers, the Canadians had contributed 14 per cent of the war-time total. It was noteworthy, and should not be overlooked, that part of the machinery indispensable to the success of the tunnelling operations had come from North America. The attention of the Institution was now being directed to Gibraltar, but it should be observed that the activities of the Tunnelling Companies were much more widespread. Work had been done in France and a great deal in England, Malta, and elsewhere.

Doubtless members, some more so than others, would have knowledge of the great history of The Rock—the Key to the

Mediterranean. The Moors entered Spain by way of Gibraltar; it had been taken and retaken, time and again, and not the least heroic episode in its momentous story was the great siege of 1779–83, when for four long years it had withstood the strenuous combined assaults of the Spaniards and the French. It was during this siege that the old galleries were driven so that guns could be suitably sited to cover the neutral ground and to provide cover for the defenders from cannon balls and primitive bombs. In modern warfare underground cover alone provided a measure of safety, but who could say how much safety was given or how deep the workings should be?

On the technical side the author said he thought some discussion might arise regarding the hand-held drill *versus* the drifter. The hand-held drill was an extremely useful machine and was versatile, if such an expression could be applied in this connection. It could be used where it was difficult, or perhaps impossible, to use the drifter. But it was a 'brutal' machine; the wear and tear on the miners using it was considerable, the effect of the incessant vibration being particularly severe when it was used for drilling holes at shoulder height. In South Africa, where the native miners might not use the machine for a long period of years, being a migratory or transitory class of workers, the effect might not be unduly great, but it was significant that in North America the drifter seemed to hold the field. The correct choice, of course, was the drifter for 'straight tunnelling' or drifting, and the hand-held machine for stoping and similar operations where the drifter could not be used.

While dealing with drills he would like to say a word about drill steels and bits. The Rip-Bit was only one of several types of detachable bit. To sharpen it a grinding machine was used, which could be placed conveniently near the working place. Grinding had the advantage of eliminating heat treatment of the bits, but the greatest possible care had to be taken against overheating while grinding. The miners quickly learned to stop using the bits before they became unduly blunted, thus reducing the amount of grinding needed. Some other types of detachable bit were sharpened in a hot-mill and then re-tempered.

The standard Army issue of Rip-Bit had a $\frac{5}{8}$-in. stud. This was suitable for the $\frac{7}{8}$-in. steel used in hand-held machines, but for 1-in. drifter steel $\frac{3}{4}$-in. studs were needed. The diamond-drill was used underground on the Rock for drilling holes of the order of 100 ft. long for the comparatively new method of underground blast-hole mining. One project entailed no less than five miles of such drilling. The particular machine used had a compressed-air motor rather after the style of a motor-cycle engine. From a perusal of North American publications* and advertisements it was clear that much ingenuity had been expended on the numerous types of blast-hole machine now on the market, and it seemed that most of the basic principles of design had been exhausted.

As the paper mentioned, research on diamond-drills for drifting had been in progress before the war. In 1937 the author saw such an experimental machine at work at the Witwatersrand University. It was a coring drill, in contrast to the normal blast-hole drill, which used solid bits. The reason why research favoured the coring drill might well be that diamondiferous bits were used and these bits, in common with all tools containing diamonds, were extremely expensive. Possibly the coring bit was a measure of economy. The author thought that the use of tungsten carbide instead of diamonds might overcome this difficulty of excessive cost, at least for rocks of moderate hardness, and he hoped for further information from the trials being made on The Rock. With the heat engendered by atomic energy it might well be—in fact, it was reported in the Press—that the synthetic diamond was a possibility.

The Eimco-Finlay type of loader was admirable for tunnelling on the level, but a word of warning was necessary to Army users of the machine—it needed 1-in. air hose and would not function properly on the standard issue of $\frac{3}{4}$-in. hose. Inclines up to 1 in 7 were driven on The Rock with a scraper-loader, fitted with an air winch. Steeper slopes had not been tackled with mechanical loaders and information on suitable machines for such an operation would be welcomed.

Standard Ruston-Bucyrus 10-cu. ft. and 19-cu. ft. excavators were used underground for loading fairly big to really big rock, and he wondered whether the bucket design was quite correct for this type of spoil. The cross-section of the tunnels had to be sufficiently large to permit the passage of the excavator. The massive Ruston-Bucyrus 37 would have been much more suitable if the size of the entries and of the chambers had permitted its use.

The comparative figures for the two sorts of ducting—rubberized canvas and metal—were interesting. The superiority of the former was due to the small number of joints (only one per 100 ft.), the absence of air leaks, and its complete smoothness when inflated.

*Douglas, R. S., Brehaut, C. H., Taylor, W. N., and Shannon, H. A. 'The New Horadiam Method of Mining at Copper Mountain'. *Mining Technology*, September, 1945.

Metal ducting in use underground quickly became battered. The air speeds were quite considerable—over short distances the fans delivered some 2,000 cu. ft. per minute, the equivalent air speed being over 40 miles per hour, while over the longest distances of ducting used the air speed was of the order of 8 m.p.h. From the figures given in the paper it would be seen that benefit from the extra power derived by coupling a second fan in series or in parallel became material only when the fans were blowing air through long lengths of ducting—a condition of slow air speeds combined with the movement of a heavy column of air. The canvas ducting was invaluable when lateral drives were being driven in addition to a main heading, as at blasting time it was very simple to slip out a 100-ft. length of ducting over the portion where it would otherwise suffer from the effect of blast.

The rounds in the tunnels were absolutely standard. The face was marked out with red-lead paint before drilling. Hole directors were used and they were very simple and quite effective. Carefully-prepared statistics were kept and showed that the advances of the rounds were remarkably constant. Sockets left after blasting were extremely short, averaging certainly not more than 3 in. The drilling task, of the order of 60 ft., was regulated to fit in with the cycle of work. The area of tunnel face per drill was 32 sq. ft. in the 8-ft. by 8-ft. tunnel, 40 sq. ft. in the 15-ft. by 8-ft. tunnel, and 48 sq. ft. in the 12-ft. by 12-ft. tunnel. This was concordant with practice in the United States of America, as Richardson and Mayo, in their excellent modern treatise entitled 'Practical Tunnel Driving'* stated on p. 325, referring to tunnel driving with drifters, that 'In general, one drill is required for each 30 to 45 sq. ft. of face area.'

Diamond-drilling was not on a task basis. The footage drilled per machine shift was just over 30 ft. In assessing the merit of that figure it was necessary to bear in mind that only one, or at the most two, holes were drilled from each set-up, that the machine had to be meticulously aligned, and that some of the drilling sites were exceedingly awkward. In contrast to this figure, it was found at Copper Mountain (see footnote on p. 5) that some 80 ft. per shift was drilled, but this figure pertained to easy set-ups, from each of which many short holes of some 50-ft. length were drilled.

The drilling rate of the hand-held machine was 0·5 ft. per min., and of the diamond-drill 0·33 ft. per min. at some 1,500 r.p.m.

*McGraw Hill Book Co., Inc., 1941.

The corresponding figure for the Eimco loader was a wagon in approximately 90 secs. The usefulness of such figures as these depended almost entirely upon the experience of the engineer judging them, but they showed clearly enough that the cycles of work had an ample margin of safety—there was power and time in reserve.

There was no doubt that the section of the paper dealing with diamond-drilling blasting would be of particular interest to members. Before embarking on a big programme of drilling, Brigadier W. G. R. Nutt had asked for comparative statistics to be prepared—diamond-drill blasting *versus* conventional mining. The former method was well in the lead, but it had other merits that could not easily be assessed in figures—such as, the absence of overbreak, the good condition of the walls after blasting, and the unhurried cycle of work.

With regard to the ' High Chambers ' (Fig. 17 of the paper) the author said that Colonel R. Colville had kindly written him to the effect that the method of working had been modified to overcome the delay in mucking out the top half with the Ruston-Bucyrus 10, by diamond-drill blasting the back, the sides, and the lower haunches in one operation. The sequence of operations was, therefore, 1, 2, 4, 5, and in a single diamond-drill blast, 3 and 6.

One of the most difficult tasks undertaken that year was that of raising the height of a chamber 25 ft. high an additional 10 ft. This was eventually accomplished in a single blast.

With regard to the arching of underground excavations : When an excavation was made, somewhere in its back (or roof) was a line above which all the stresses were in compression, as in an arch. It was difficult to say where this line would be in any particular medium, at any given depth, but it seemed clear that the wider the excavation, the higher the line of the arch would be. He thought it was right to say that all rock below the ' equilibrium arch ' was hanging on to the arch and depended for its stability on the tensile strength of the rock. A simple analogy would be a railway arch in which the arch had been filled up with brick—who could say when the filling would fall out ? To be structurally sound, therefore, the excavation must be correctly arched, the arching being progressively steeper as widths increased. He thought that it was safe on The Rock to leave excavations flat up to some 12 ft. to 15 ft. in width, but thereafter they should be arched. It could not be too emphatically pointed out, however, that arching would not prevent the odd block of rock between slip planes from eventually becoming dislodged. He doubted if that spans of any great width would be stable under heavy shock, the condition which they were made expressly to defy.

A point of interest was that as work in these big chambers proceeded the back was inspected with the aid of a strong light, and small fragments of rock that had become detached were shot off, using a rifle and lead bullets. A ·22 gave an astonishingly heavy blow and brought down any piece that was on the verge of falling.

The author added that he had not thought it necessary to recount the various stages by which methods had been improved and the sizes of chambers increased. Evolution had taken place in this, as in all other work.

In conclusion, the author expressed thanks to his excellent surveyor-draughtsman, Corporal N. W. G. Haynes, who had assisted with the illustrations.

Lieut.-Gen. Sir Noel Mason Macfarlane, M.P., expressed his thanks for the invitation to attend the meeting to hear Major Wilson's talk on his interesting paper. This was not the first time that he had talked to a tunnelling gathering. He had spoken on several occasions to his Companies in Gibraltar. He remembered one occasion on which he spoke to about 300 Canadian Tunnellers at a time when they thought they had been let down and he had to explain that things were not as bad as they thought. This was less of an ordeal than his task in addressing so distinguished a gathering to-day. Another occasion he would never forget was the official opening of the Great North Road on its completion by 170 Company.

In The Rock at Gibraltar there were the original tunnels dating back to the great siege, and there was another phase of tunnelling at the end of last century, or the beginning of this, when a certain amount of Naval tunnelling was done—including the Naval Dockyard tunnel, which went right through The Rock. There was a certain amount when the civilian air raid shelters were built and they proved of tremendous value during the whole of the time of the war. The final stage was undertaken after Dunkirk, when, for the first time, there seemed to be a possibility, in fact a strong probability, that the Germans might descend upon the garrison through Spain. Some very anxious times were passed through during those days. One of the great difficulties was the question of just where to start and what should be done. It was a situation where a short-term policy had to be married to a long-term policy ; the short-term policy had to be very active. It was decided to make The Rock capable of standing up to a siege without relief for at least a year, which meant accommodation for many thousand people, with all the ammunition, rations, distilleries for water, etc., inside The Rock, as it was open to bombardment not only from the air, but from the ground from Spain.

Work was commenced and the initial tunnelling served both purposes; it helped the short-term and was the beginning of the long-term policy, and as time went on the tunnelling policy was developed and the whole network of tunnels as it was to-day finally emerged. When he left in January, 1944, already 1,000,000 tons of rock had been removed and probably many more had been taken out since then.

In the early days Colonel Thomson, who went out with the original R.E. Tunnellers, Brigadier Fordham, Chief Engineer, Colonel Clifton, and Colonel Pidsley, the G.I., and himself, used to lunch together on Sundays and then go round to see the week's work. It was extraordinary to note the progress and the advances made by each Tunnelling Company. The results became better and better as time went on. They did not ever take to the local form of tunnelling, where the local Spaniard or Gibraltarian lay on his back and worked the drill with his feet!

The speaker had various recollections of tunnelling in Gibraltar. He could remember the original shots being fired for the whole of the new system. Another recollection was of a tunnel which he had been wanting to make for a long time on the east side of The Rock where there was a very bad corner and nobody would ever let him build it. At last, in building the aerodrome, there was an extra large blast which, combined with digging, caused a landslide which covered the road and blocked the passage round the east side, so he was finally allowed to build the tunnel. There was only one bad accident the whole time he was there.

The real point he wished to make was the question of the Tunnelling personnel. He had 178, 179, 180, 170, and 172 Tunnelling Companies, Royal Engineers, and the Canadian tunnellers, with their special diamond-drill detachment. They were, without exception, absolutely first-class units and it did one's eyes good to see the way they carried out their work. In the days between June, 1940, and the spring of 1941, one never knew what was going to happen. They were always being told that something might happen at any moment and very few people realized what the Gibraltar garrison was doing. They worked 24 hours round the clock; they did not know what a Saturday afternoon or Sunday meant. Having done an eight-hour shift they would come off and spend two or three hours learning to man a Bren gun or a mortar or something else, because everybody on The Rock was going to fight if necessary. He could honestly say that he had never seen a finer lot of men than the Tunnelling Companies who were working all the time he was at Gibraltar. The machine was of immense interest, but it was the man behind the machine which counted in the end and he would like, once again, to say 'thank you' to the Tunnelling Companies who put up such a magnificent show in Gibraltar during the time he was there during the war.

Colonel H. M. Fordham said that General Sir Noel Mason MacFarlane had left him very little to add, but he entirely agreed with every word that he had said. The staying power of the officers and men in the Tunnelling Companies was magnificent, no Chief Engineer could have been better served.

On the technical side there was one point which might be of interest. During his time on The Rock, from September, 1940, to the beginning of 1942, the diamond drill was not used, broadly speaking, for 'slashing', but a great deal of use was made of it for prospecting and it proved extraordinarily valuable. The core would come out beautifully clean and one could see the nature of the rock through which one intended to proceed. In one place they were working where it was known that the mud seam was fairly close, holes were made with the diamond drill at both ends of 1,100 ft. of core, and it was found that the mud seam on that line was only 2 or 3 ft. wide, the rest being firm limestone. In making the very big chambers in the place known as 'Gorts', where, again, a mud seam was known to be near, the diamond drills were used in five directions and showed where it was and how far it was safe to go. Once the mud was reached reinforced concrete linings had to be used. That was the other aspect of diamond drilling and it was of the greatest value in planning where new works should be sited.

Professor J. A. S. Ritson said that Major Wilson was an old colleague of his and the paper was the kind he would expect from him. It was extremely well written, concise, amply illustrated, and particularly applicable in its description of precise methods to the phase of mining in which Major Wilson made his pre-war reputation—surveying. He had already been awarded both the Student's Prize and the Gold Fields Premium, so that one was entitled to expect something very good from him.

Before discussing the paper he should explain that he was not a Tunneller during the 1914–1918 war, but he took a Company of Durham miners to France and later the survivors from the first gas attack were transferred to form one of the earliest Tunnelling

Companies and he was left behind as an Infantryman. The Infantry lost some magnificent men, but the Tunnelling Company was the gainer. Although he remained an Infantryman all the rest of the war he frequently visited his friends in the Tunnellers and knew something of the work they did and it was because of that experience he crossed swords with Major Wilson when he said that tunnelling in this war was a much more technical business than tunnelling in the last war. He could conceive very little more difficult work technically than driving a small gallery, silently, in the presence of enemy listening posts, through the running sands of the Flanders Plain. The support of such galleries, the removal of the spoil and the packing of hundreds of tons of explosive into a small space were achievements of no small technical order. The successful operations of Messines Ridge and the mining and counter-mining in the Ypres Salient were examples which illustrated his point.

The mechanical assistance which the Tunnellers had was crude, but it was made to function. What mattered was the men! To the earlier generation the conditions under which Major Wilson was and the other Companies were working in Gibraltar would be heavenly ; firm ground, no noxious gases, little enemy interference, easy access and ample space to work in.

The first impression of the paper, apart from this particular point, was that it was written by a military and not a civilian engineer. The Corps of Royal Engineers were justly proud of their work and carried it out with extraordinary precision. This precision did not apply to the mining engineer, who had to break ground at minimum cost. It was very difficult for him to understand why it should be necessary for these tunnels and large excavations to be so beautifully made. He knew of cavities in limestone which had remained open for many years, some of them hundreds of years, quite safely, but they were not so meticulously made nor were they so correct in every dimension. After being excavated the sides were dressed down with a bar and all loose and hanging stones removed. They were periodically examined and had stood the test of time. Was there anything in the Gibraltar limestone which rendered it different from the Carboniferous and other limestones of this country ? He suggested that natural and wholly admirable pride of workmanship was carried too far in this case and the author's adherence to his Corps traditions masked his economic mining training. It might well be, however, that for security reasons many interesting points which would explain this extreme accuracy had had to be omitted. Major Wilson would probably put him right on this point.

The story of the diamond drill-holes was most interesting. As the author suggested, this method might have an increasing use in mining, but here again not quite as it was used in Gibraltar. In commercial mining there was no necessity for such accuracy as would necessitate the making of exploratory slots every 100 ft. or 150 ft., and, if the hole was 9 in. out, having to re-drill it. In any case, if it was necessary to be so accurate, he was surprised that Major Wilson was not informed that methods were developed early in the war by the R.E. Signal and Research Board, of which certain members of the Institution were members, whereby holes of much greater length than the ones at Gibraltar could be accurately surveyed. This was known to quite a number of senior Royal Engineer officers.

There was no adverse comment to be made on the blasting technique. He assumed that the insertion of every cartridge into the hole was accurately measured, so that each butted up against the next, otherwise there would be partial burning, possibly incomplete combustion, and partial misfire, which would account for the excessive fumes and cause a very serious delay. He also assumed that the electrical circuits for blasting were tested for continuity by the use of some instrument and that excessive resistance, if it occurred, would be detected. He would, however, like to know why detonating fuse was not used for this 100 ft. of explosive. It was commonly used in this country for large blasts in limestone which brought down 250,000 to 500,000 tons at a time. The remarks in the paper as to the position of the detonator were too vague. Experimental work in this country showed that there were only two positions for the detonator—preferably in the uppermost cartridge pointing axially down the centre of the charge, or in the innermost cartridge pointing up the charge, again, axially. From the diagram which he, the speaker, showed it was obvious that the proper place to put the detonator cartridge was in the end if full advantage was to be gained from its initiating shock. The point he was trying to make was that there was only one place for the detonator—at the end of the charge pointing along the charge.

On p. 9 there were some tables showing fan tests. He had analysed these tables, particularly Nos. II, III, and IV, and the results were very inconsistent. He could only assume that the quantity readings were incorrect. They did show, however, the extreme roughness of metal ducts as compared with canvas ducts. If water-gauge readings had been taken the results would have been more valuable. When two identical fans were run in series and the water gauge was not increased the quantity produced by the two fans was $\sqrt{2}$ times the quantity given by one. When the combined power was 2 P, the quantity would be $\sqrt[3]{2}\,Q$. When the two identical fans were run in parallel, the water gauge did not add as in series, and advantage was only gained when the resistance to the flow of air through the fans themselves was considerable. Auxiliary fans on a mine were fans in series, but in this case only a portion of the main air current passed through the auxiliary fans, which was not the same problem as the case where the two fans were in close proximity.

Another point which was not quite clear was how the 15-ft. by 30-ft. tunnels were driven. A 15-ft. by 8-ft. tunnel was drilled first and the back was drilled by the man standing on the rubbish. Was the primary tunnel kept only a few feet in advance of the main tunnel? If not, how was sufficient rubbish removed to give the men standing room when drilling the 'back'?

Why was the drilling done dry? He disliked dust in all circumstances, even if it was limestone dust, and he wondered why dry drilling was used and why salt water could not have been used for laying it. There was probably a good reason, but he did not know it.

Brigadier R. S. G. Stokes said he would like to support the congratulations accorded to the author upon his lucid description of a special and important task, efficiently executed. Of chief interest, clearly, were his observations upon diamond-drill blasting practice. Experience of its merits, and also the limitations, had been rapidly accumulating in recent years from Canada, Southern Africa, and Australia. The new data from Gibraltar would be widely appreciated.

The increasing scope for the adoption of rotary drills for blastholes in mining was recognized everywhere, but hopefully, as he saw the picture, only where their use could be integrated with daily routine practice, and not for such intermittent services as in the partial cutting of underground engine and pump stations sumps—the commonest rock chambers in mining.

For obvious reasons, comparisons of cost between the methods described and common practice could not be presented, but the paper would have been much enriched and conclusions clarified by an analysis of time, labour, and materials employed upon a typical chamber, from start to finish, and a corresponding estimate of requirements and results with older methods, using long drifter holes. Without this indication of over-all economic benefits, he felt that practical conclusions could be drawn only with great reserve. They should know what was the saving, particularly in time, when due allowance was made for the delays in rigging up, the facing of the rock for collaring, rejection of holes for faulty alignment, excavation of slots, and, after the blast, for breaking up of big slabs and boulders for easy handling.

Any interruption of the routine cycle of daily operations that was commonly so important a factor in the establishment of economy underground demanded that the evidence presented in favour of dual-method practice should be particularly strong. But the author boldly claimed that a combination of diamond drill and jackhammer or drifter work would apply to the "excavation of all very large chambers in the future." This was surely much too sweeping. It would often be found that these heavy blasts were too severe upon the roof or walls. The remarkable strength of the steeply-dipping limestone of Gibraltar, carrying little weight, was a favourable characteristic not found in the average mine.

Again, they must consider the time saved by the easy handling and disposal of the spoil in big slabs, as shown in the photo (Fig. 23), without all the popping or spalling that would be necessary if the spoil had to be passed through a shaft system to the surface. Such big-scale operation with R.B.10 and 19 excavators, feeding lorries direct, would rarely be capable of repetition, except for similar accommodation tasks with adit facilities.

Under ventilation, he found it difficult to accept as characteristic the author's comparisons between the efficiencies disclosed with special canvas ducting and metal piping, even though the lengths were 100 ft. and 10 ft. respectively. The contrast bewteen 2,600 c.f.p.m. and 630 c.f.p.m., at 300 ft., called for some explanation. He could not believe that the pipe jointing received the meticulous care which admittedly had to be devoted to the preservation of the canvas. Galvanized iron piping was so popular in mines on account of its comparative durability, strength, and cheapness that he suggested the author could not leave Table II as it stood without elaboration in his reply to discussion.

Major Wilson gave him his conception of the part to be played by the Tunnellers in future warfare. He feared it would be necessary to have a vast army of Tunnellers at work for a good many years before the next 'D' Day, or Doom Day, to provide cover of any critical significance.

Although Major Wilson paid fine tribute to the activities of the Tunnellers in the first great war, the speaker regretted that, in a context of particular concern to the Institution, he had made comparisons between the rôles played in the two wars in somewhat controversial terms. Major Wilson said that the main task of the old Tunnellers called for much endurance and courage, ' but not for a very high degree of technical skill. The tunnelling organization, however, was large.' The word ' however ' was truly devastating. The speaker could not accept the implications. ' Technical skill ' postulated the application of expert knowledge and not necessarily mechanical refinements, which were sometimes poor economics. And surely technical skill of a very high order was displayed in the silent excavation of squeezing clay and hard chalk. in sinking through wet sands, in mine-listening systems, in Proto rescue work, in the estimation of rupturing effects of charges in variable ground, and in the reopening of galleries after enemy blasts.

The success of the first war's most important tunnelling operations, culminating in the capture of Messines, was chiefly due to the high technical skill of the Royal and Dominion Engineers engaged. Success demanded the element of surprise, and surprise was attained, simply and solely because the technical advisers of the German Staff had declared deep mining, in that area, for technical reasons, would not be possible.

The speaker referred to this feature of Major Wilson's admirable paper, with some reluctance, in the interests of historical accuracy. Quite unconsciously, he was sure, Major Wilson had presented an unbalanced picture of the work done by a branch of the Royal Engineers, in the creation of which—under the leadership of the late General Harvey, of General Williams and General Hyland—the Institution had played no unworthy part.

Lt.-Col. A. R. O. Williams, after congratulating the author, said that, having had some connection with the tunnelling work done during the recent war at Gibraltar, it seemed clear to him that the mining practice described was that prevailing during the past two years, when the only Tunnelling Company then at work had had ample mining equipment and, apparently, never suffered from a shortage of essential stores. The great bulk of the underground work carried out in The Rock during the war, however, was done between the late summer of 1940 and the autumn of 1943. Throughout these three years a force of some three to four Tunnelling Companies R.E.—for about two years one of these was a Canadian Company—and a Tunnelling Workshop Section, all grouped under Tunnelling H.Q., R.E., was continuously at work.

The paper was of particular interest to those who had been engaged on this earlier work, for the methods described by Major

Wilson were the evolutionary product of those introduced or tried out experimentally during those first three years. Throughout the major part of that period, but particularly during 1940 and 1941, the choice of tunnelling methods was influenced, in fact dominated, by shortages of equipment and stores. When work started the only drills available were 25-lb. jackhammers, while a quite inadequate number of aged 100-cu. ft. portable compressors was the sole source of air. The dropping of a bomb in September, 1940, on part of the R.E. store building was singularly unfortunate.

A point mentioned by the author was the virtual absence of any hand-mucking. For about the first 12 months all broken rock was removed from the working faces by hand shovel. Later, Eimco loaders and Holman scraper-loaders arrived on The Rock, but not until the tunnelling force was reduced to one Company were there sufficient machines to allow of the elimination of hand-mucking. Infantry and the R.A. did most of this hand-mucking, under the direction of the R.E. Tunnellers. The scraper-loaders were not very successful, due mainly to the difficulty of obtaining suitable steel wire rope, but the Eimcos were of tremendous help.

Diamond drilling occupied a large part of the paper. Some drills were brought to The Rock by the Canadians and, initially, were used for making bomb- and shell-proof ways for signal and power cables, essential pipe-lines, etc., and, to some extent, for prospecting. Some experimental blast-hole work was done in 1942 in connection with the excavation of large chambers, but it was really the inability to repair broken-down drifters, due to the non-arrival of spare parts, that led to the adoption of the diamond drill as a mining or excavational tool.

All the diamond drills and the operators were supplied by the Royal Canadian Engineers. To permit of diamond drilling being continued after the withdrawal of the Canadians it was decided to train Royal Engineers to do the work. A small detachment (approximately 20 men and one officer) were carefully selected from the R.E. Tunnellers on The Rock, none of whom, except for some of the officers, had had any previous experience of diamond drilling. They spent a month with the Canadians before the latter left and then carried on on their own with great enthusiasm and keenness. The marked success that attended the work of this small unit was a credit to the Tunnellers of the Royal Engineers.

He would like to mention that the Tunnellers received every help and encouragement from each of the several Chief Engineers that in succession directed R.E. work at Gibraltar. He would also like to mention in particular General Mason Macfarlane—' Mason

Mac '—whose unfailing interest and enthusiasm had played no small part in such success as the Tunnellers achieved on The Rock.

Lastly, he wished to say a word about the fellow who formed the backbone of the Tunnelling force, the 'other rank', or, as a previous speaker had aptly termed him, the man behind the machine. He came from the coalfields of England, of South Wales, and of Scotland. Seldom, if at all, had he had any experience of hard rock mining, but by dint of solid hard work, carried out with cheerful enthusiasm, he had rapidly mastered the new technique and soon became highly efficient. He knew that every mining engineer who had had the privilege of serving in the Tunnellers with these men, these British coalminers, looked back with gratitude, and pride, upon that association.

Major G. A. P. Moorhead wished to give some facts in defence of the hand-drill as used for drifting in the limestone in Gibraltar. Major Wilson was of the opinion that if in the early years of the war it had been possible to forecast a long programme of tunnelling, the hand-held machine should have been given up for the drifter. Major Wilson also said that in an 8-ft. by 8-ft. heading working with two machines an advance of 72 ft. was made in a week of 18 shifts, which accorded with his own experience on The Rock, except that in his Tunnelling Company one driller and one driller's mate were employed per machine and not two drillers. The task was the same—namely, 46 ft. per machine, or a 4-ft. advance per shift, but this task was usually completed after four hours' drilling. On occasion, when the urgency of the job warranted, six-hour shifts were worked, which gave 16-ft. advance per day, and in this manner 178 Company R.E. did as much as 110 ft. in a week, which, for a short time, was the footage record on The Rock. It was improved on by the 170 Company, which did 180 ft. in a week. On both occasions the machines were hand-held and he did not think that drifters would have done better footages. One always found ventilation and spoil removal the bottle necks.

Tests were run with the S.L.9's mounted on cradles and used as water leyners, but the advantages gained did not compensate for the time lost in erecting the columns and laying on the water; the more pleasant working conditions were not appreciated by the men, whose one aim was to complete their task as quickly as possible.

Although heavier water leyners were used by the Canadian Tunnellers, their overall time on development was no better than the British.

Professor W. R. Jones said that it was very fortunate that a member of the Institution was present when these very important operations were being conducted and particularly one who had the gift of describing them in clear detail. Major Wilson stated that during these operations there were severe concentrations of dust, but that no driller suffered any apparent harm from inhaling the limestone dust. The speaker believed there was a reason for this. The dust of limestone was not so harmful as the dust of silica and silicate minerals. When one examined sections of lungs one found that the dust of silica and silicate minerals was in discrete particles, whereas that of limestone had a tendency to aggregate, forming little clots of dust which had not the tendency to penetrate into the alveoli of the lung as did the individual particles of the former minerals. There was no recorded case of pneumoconiosis contracted by the inhalation of pure or almost pure limestone dust.

The cave known as Wilson's Cave had yielded a wealth of information from mammoth bones and implements and Neolithic ware found there. Some bronze rings were also found, which the British Museum had recently dated as 500 B.C. The various articles were being examined and it was believed—he stated this on the evidence of a letter he had recently received from Captain C. B. Alexander—that a great amount of historical information would be obtained from them. Major Wilson sent him a specimen of thel ead minerals found in the cave and it seemed possible that metallic lead, taken into the opening, had altered to pyromorphite and lead oxides as the result of solutions acting on organic deposits—such as, the guano and bones so plentiful in the cave.

Major Wilson was to be congratulated on his lucid presentation of a very valuable and most imteresting paper.

Mr. W. W. Varvill said that Major Wilson's paper was most timely and interesting and gave a wealth of detail which would be very useful to anyone who had the same kind of work to do and, as he stated, they did not know to what depths the new atomic age would drive them to seek shelter.

Speaking as a Sapper of the 1914–18 War, he must, however take exception to the author's remarks that their work then did not 'call for a very high degree of technical skill.' There was a tendency for the new generation to belittle the efforts of their predecessors, but he could assure the author that those who had to mine with inadequate tools in the running sands and squeezing clay of the Lys Valley, or amongst the coastal sand dunes, would have considered working the limestone of Gibraltar to be a picnic.

The author stated that most of the drilling by compressed air-driven machines was done dry. Was this owing to the military necessity—that is to say, a lack of wet machines—or was there an objection to using salt water for the purpose? The author stated that salt water was used for the diamond drills, so there must have been water laid on to the faces.

The speaker's experience had been that it was inefficient to compel men to work where they inhaled any kind of dust—injurious or otherwise—and there must have been a strong sense of discipline amongst the miners for them to put up with it. He was sure that the civilian miners would have objected. Did they wear respirators? With regard to the observation that X-ray photographs of the men's lungs did not reveal any development of lung trouble, the speaker would like to hear if the period of their employment was considered to be long enough for such symptoms to develop.

He saw that the Hunslet-Hudson locos. were fitted with exhaust conditioners. Was this precaution also applied to the diesel shovels, the bulldozers, and the lorries? If not, was there any trouble from the fumes from those engines?

It was noted that the fans which were used to blow the dust from the working faces had to be cleaned every 48 hours; it was to be hoped that there were similar arrangements for clearing the men's throats when they were off duty!

Would the author enlarge upon his statement that electric blasting could not be allowed in close proximity to the radio transmitting station? Was it through fear of the blasting affecting the radio, or fear of 'statics' causing premature firing of the shots?

It would be very interesting if the author could tell them why it was desirable to ensure such a high degree of perfection in adhering to the designed shape and size of the chambers, and the smoothness of their sides and roofs. The speaker had had no experience of the Gibraltar limestone, but had seen many excavated stopes and natural caverns in the British limestones whose roofs and sides were of the most irregular shapes. These, after a little trimming down, would stand open indefinitely. Most of them knew of unlined railway tunnels in limestone in this country. Possibly the limestone of Gibraltar was not so strong? The Gibraltar shelters were for cover against air raids and to the layman it was not easy to understand why there was need for such smooth sides, beyond the normal desire of the Royal Engineers to do a job tidily.

Was the author satisfied that the long diamond-drilling method was justified for use in what presumably was not a very hard rock, compared with what was generally dealt with by metal miners? Apart from tidiness, would it not have been quicker and cheaper to have drilled out the whole of the chambers with hand-held drills, using the system shown in Fig. 15?

The author stated that the diamond-drill holes, which were upwards of 100 ft. long, had to be re-drilled if they were deflected more than 9 in. from their mark, whilst the average speed of drilling was only 30 ft. per shift. The speaker stood open to correction, but it seemed to him that much of the work of raising and cutting the 'slots,' the extra surveying needed to align the diamond drills, the slow rate of drilling, the holes lost by being out of line, the need for secondary blasting of large boulders, and the time and care required for charging these long holes, could all have been done more quickly and more cheaply, or avoided altogether, by using the older methods.

Very heavy simultaneous blasts were required (upwards of a ton of explosive at a time) by the diamond-drilling method. This showed that the rock was capable of standing a good deal of shaking without leaving dangerous roofs and sides to the chambers.

His remarks are not intended as criticism, but simply put to seek information. The man who lived on the job usually had a very good reason for his methods, and he thought the paper would be made even more interesting and valuable if they had this additional information.

The diamond drilling of long blast-holes was becoming established practice for mining some large ore-bodies where it could be used without fear of breaking country rock and causing dilution, but it would not be economic practice if there was much risk of having to re-drill holes that had run slightly out of alignment. It would be interesting to hear what proportion of the holes went astray.

The speaker had had no experience of use of the diamond drill for blast holes, but had read that the main advantage of the method lay in the protection it gave to the men from falling roof, owing to their not having to remove their equipment to fresh set-ups at frequent intervals; otherwise the method was more costly than ordinary rock drilling. This advantage of being safer could not apply at Gibraltar. In an account of its use at the East Malartic mine it was claimed that a footage of 35·4 ft. per machine shift was drilled with the diamond drill. This was rather more than the speed recorded at Gibraltar. Compared with the speed of drilling by ordinary methods the comparison was much less favourable to the diamond drill.

In West African gold mines, where labour was far from being efficient, it was common practice to set up and drill a round comprising about 80 ft. of holes in a single 8-hour shift. This was why it appeared to the speaker that there must have been some good reason, which was not given in the paper, for the decision to use the diamond drill.

Brigadier E. J. B. Buchanan said that he would not raise any technical aspect because his experience of tunnelling was confined to a few years before the war when, using bow and arrow methods, they tried to get the ammunition at Malta under cover. It was with great interest that he listened to the Chairman's opening remarks when he referred to the tunnelling activities of the Royal Engineers in the late war because he could well remember how the members of the Institution came from all corners of the earth to do the work. He remembered, too, that he was very glad that he did not have to go underground and take part in the terrifying activities necessary in the Flanders terrain, and therefore could well appreciate the comments of some of the speakers apropos of the references made to their activities at that time. He could equally well remember the joy when, at the beginning of this war, many of the members came back as old friends to rejoin the Royal Engineers. Because operational tunnelling was not required they lent a hand at whatever job there was, from building aerodromes to laying and lifting mines. There was no activity in which they could not take the lead. The Tunnelling Companies had provided a chapter of Royal Engineer history of which the Army and the Institution had every reason to be proud.

The Chairman said that owing to the lateness of the hour any further contributions would have to be submitted in writing. He asked the author whether he would like to reply to any points at the meeting or would reply in writing.

Major Wilson said that there were one or two points he would like to make. One was with regard to the clearing of dust from the miners' throats—they did it in the time-honoured way, with beer. He regretted the remark regarding the operations of the last war at the beginning of the paper, but it served a good purpose in bringing forth an avalanche of criticism, thereby showing how excellent were the Tunnellers of 1914–1918. He had anticipated that something of the sort would happen.

With regard to the drift of the diamond-drill holes; not many were discarded as care was taken that they were oriented correctly. There had to be a limit to the amount of discrepancy permitted—a hole could not be allowed to be, say, 4 ft. out, or even 2 ft. out. He had said 9 in. in the paper and that was achieved. If the roof was arched, with the holes 8 ft. apart, and if one was 2 ft. out on one side, and another was 2 ft. out on the other side, the result would be odd, to say the least. Various erections had to be made in the excavations and if the chamber was small then they would not have gone in.

The cutting of the slots was quite an economic operation. The rate of diamond drilling was slow, but these holes took an immensely greater burden than ordinary drill-holes, and as far as efficiency was concerned the method proved to be well ahead of other methods. It was important to remember that it was really *civil engineering* mining, rather than *mining* mining. It was easier to be accurate than inaccurate. The tunnels were used as roadways, and buildings were put inside the large excavations. When he said the diamond-drilling method should be used wherever big excavations were made, he meant such excavations as the Army or Navy might have in mind. On the question of the drifter *versus* the hand-held machine, the drifter took a bigger bite, say an 8-ft. round, while with the hand-machine a 5-ft. round was obtained. In big tunnels drifter practice was the best.

The Chairman proposed a very hearty vote of thanks to the author for his interesting paper, which was accorded by applause. The Chairman added that he was sorry that the second paper for discussion on the development of the Blyvooruitzicht Gold Mining Co. by Mr. Savile Davis had not been reached; it would be postponed until the next meeting.

CONTRIBUTED REMARKS

Lt.-Col. D. M. Thomson: All Tunnellers will welcome Major Wilson's paper. On behalf of the men, many of whom will not know of Major Wilson's time given to putting something of their work on record, I can add their thanks to my own. Eight Tunnellers awaiting demobilization in a holding unit out here all expressed their interest and pleasure in the paper about 'The Rock'. These men, typical of Tunnellers, show genuine regret that the Tunnellers' partnership is finished, when they recall the true comradeship and strenuous work done—not for a pay cheque or dividend, but for their ' Company '.

The author is too modest in dealing only with the machine, and not the Sapper, of 1939–45. Perhaps, like Kipling, he thinks ' that is another story '. Myself, I feel the *Transactions* are always enriched with an account of the human element, put on record and not left to be taken for granted. I hope one of the younger men will put on record somewhere the story of the Miner at War, 1939–45. It is known that three Tunnellers jumped into Arnhem and remained there. Young Captain Bennett, a great Tunneller, ' went West ' with his assault tank and crew in the last days before victory in North Germany.

The miner of this war lived and worked in the tradition of the earlier model Sapper and Miner. The machine age found the man unaltered. Earlier than India, on Gibraltar itself during the great siege, he not only left tunnels, but founded families with Cornish names, who still bore into the rock for the town council or water works committee, and keep adding to the Admiralty labyrinth in the dockyards. Main Street, Gibraltar; Strada Reale, Malta; Chausée Bon-Enfant, Doullens, places I've never seen or heard of in Burma; and now the Strasser of the Ruhr—all have known the British miner, and, cautious, even afraid of him at first, have loved him in the end.

Others will have commented on Major Wilson's remark that the work of the 1914–18 Tunnellers did not call for ' a very high degree of technical skill '. I still take my hat off to the timbering, calculations, and accurate survey work of the Tunnellers in question. Their memorial used to stand in the Hall at the Institution's office. The memorial to the 1939–45 Tunnellers will be a bronze plaque on a gallery wall in Gibraltar.

Mr. A. F. Skerl: The pureness and homogeneity of the hard limestone appear to have been of prime importance in the achievement of the good results so well described and illustrated in the paper. It would be interesting to learn whether the Rip-Bits were usable down to almost $\frac{7}{8}$ in. diameter—i.e., to the size of the Rip-Rods. That the author found that drillers prefer to use as small a bit as possible confirms Mr. Irving's statement in a recent paper.* Can the author give any actual reasons for the drillers' preference.

*IRVING, C. J. ' Some Aspects of Rock Drilling Practice (The Witwatersrand Goldfield) '. *Bull.* Instn. M.M., No. 474, September, 1945.

The use of solid crowns for diamond drilling seems, at first consideration at least, to be wasteful of power and diamond. The time gained in not having to pull cores is problematical, as core drilling may be reasonably expected to be faster. The writer suggests that the experiments mentioned by the author might usefully include trials using annulus crowns containing not diamond, but tungsten carbide. That the holes bored with solid crowns deflected downwards is a point of interest, since in core drilling the usual tendency is for the deflection to be upwards.

The supersession of the pneumatic percussion drill by an electrically-driven automatically-fed rotary drill seems certain for many conditions of drilling. Mr. S. O. Hatton in the discussion* on Mr. Irving's paper mentions several good reasons why this is likely to come about—viz., greater power efficiency, less dust hazard, and greater mechanical efficiency, with moving parts operating under conditions of much less strain. Light, easily-erected supports would suffice and a single driller would be in charge of a battery of such rotary drills, each operating automatically except for setting-up, changing crowns and insertion of additional lengths of rods when necessary. This latter would be perhaps once for each hole in face drilling. A hydraulic feed of about 18 in. per run would seem feasible, when the whole drill would be drawn back, the rod unclamped from the driving mechanism, passed forward to the bottom of the hole, reclamped, and then drilling would be continued.

The 'underground quarrying' using diamond-drill blast holes described in this paper demonstrates the advantage of using diamond-drill holes to enable blasting to be performed in the position of most free face.

Mr. A. Alexander: A taste of good fare whets the appetite for more. What follows has the constructive object of educing more detail.

(1) On p. 2, 'Geology' is dismissed with two short paragraphs on the plea of complexity. I suggest that no problem of 'breaking ground' can be dissociated from that branch of geological study known as 'Tectonics.' It would therefore be helpful if the author would supplement some of his excellent drawings and diagrams with a description and/or illustrations to indicate the general disposition of bedding planes, cleats, faults, etc. Every practical miner determines the minimum number of holes in a drilling round and generally saves on explosive consumption by taking

*Bull. No. 475, November, 1945.

such tectonics into account. Futhermore, we should be able to understand better what problems arose in respect of ' Deflection Control ' for the rotary drilling if such knowledge was available. Here again is scope for some standard yardstick, as Hildick-Smith* has expressed it, ' in a form capable of world-wide interpretation.'

(2) Consequent upon the blasting of diamond-drill holes, the author draws attention to the excellent feature of ' smooth and safe condition . . .' Will he confirm the impression that: (a) dressing-down after a blast was relatively small, and that having been accomplished, the chambers would be considered *safe* without further periodical attention? (b) The dislodgment of small lumps by rifle fire was mentioned by Major Wilson in his closing remarks; has he anything further to report for this novel method in such details as spot-lighting for the marksman and whether ricochets were a danger? In view of McPherson's hints† as to the use of explosive impact without the mass and not by impact within the mass, what hope is there of developing this method as a new technique for dressing down inaccessible ceilings? Many of us know how a machine-gun or Piat can penetrate masonry if you can afford to use the ammunition; it would be interesting to know the reactions of H.M. Deputy Chief Inspector of Mines to this germ of an idea for metalliferous mining?

(3) Will Major Wilson make it clear why delayed-action detonators were not used in the smaller section headings? He has made it clear that instantaneous blasting is an essential feature for the diamond-drilled holes.

(4) There are two small points in the paper which call for an indication of the underground temperature and humidity: (a) It is presumed that there are not temperatures high enough to create a creep or instability of the tar macadam? (b) My experience goes to show that the durability of the canvas ducts is related to underground humidity and temperature. There will be agreement that they are more satisfactory than sheet-iron tubes, but here would appear to be a field for research for a better medium than either iron or canvas; the practical development of pliable plastics may afford the answer.

(5) The author mentions exhaust conditioners for his diesel locomotives. Were similar filters fitted to his R.B. diggers and bulldozers?

(6) Major Wilson is a specialist in mine surveying; will he tell us how deep it was necessary to set his floor stations and the means

*Op. cit. supra, p. 7.
†Op. cit. supra, p. 11.

taken to allow such stations (presumably flush with floor level) to be *readily* found in a floor area relatively vast ?

(7) In his opening remarks concerning the reconditioning of detachable bits, the author mentioned the need for re-tempering. One of the valuable attributes of detachable bits as received from the makers is the uniformity of hardness; in order to regain such uniformity, did Major Wilson have any special equipment for tempering control ?

(8) I believe it was mentioned that a fleet totalling 18 Eimco loaders were available at Gibraltar; the total number of R.B. diggers, bull-dozers, and scrape-loaders was not revealed. Will Major Wilson give an outline of the organization for servicing these machines with special reference to: (a) Whether each loader was withdrawn for overhaul after loading a specified tonnage ? (b) Whether the maintenance workshops were underground and how equipped and lit ?

In conclusion I would join the chorus of sincere thanks to Major Wilson for a paper of special value to mining engineers confronted with the ever-present problem of breaking enough tonnage to keep loading plant efficiently employed. As an old Tunneller my thanks are due for this spotlight on the fact that our military miners in the past six years have upheld the reputation established by their predecessors in 1914–18. The military mining engineer has one outstanding advantage over his civilian equivalent in Great Britain; his personnel are thoroughly reliable because they are disciplined.

Mr. S. H. De La Mare: It is difficult to criticize the author's statement that the efficiency of canvas ducting is greater than metal ducting without the definite statement that the metal ducting was also of 11-in. diameter.

In the 1914–18 war in Flanders, 3-in. piping, whether sheet metal or rubber-lined, or whether serviced by the largest blacksmith bellows, a Holman air pump or a Bosch fan, all hand operated, would not deflect a candle flame at 500 ft., whereas by using 150 cm. (say 6-in.) captured Bosch sheet metal piping a sufficient supply of air was obtained ot 1,600 ft., using identical means of propulsion.

The National Fire Service found it necessary when filling static water tanks from distant ponds to space their rotary pumps not more than 1,000 ft. apart. Admitting that loss of pressure by air due to pipe friction may not be precisely the same as with water (the N.F.S. allow for a drop of 2·5 lb. per square inch per 100 ft. run in 2½-in. rubber-lined hose), it is well to remember that to reduce the output of an electrically-driven rotary water pump it is quite common to choke the delivery side, but not the suction, and that there is practically a corresponding fall in the current consumed. It would therefore seem that to use two fans in parallel would be definitely wrong if the duct was a long one and that little improvement could be expected with the fans in series if they were kept close together.

In the case of water, and probably also of air, the pressure or valve against which the pump is working restricts the amount of new load it can pick up and the load already on the vanes follows round with the vanes without requiring further energy to provide velocity for new load.

It is fair to suggest that taking only the surveying in the 1914–18 War, which had no static points grouted into solid rock either on the surface or underground but always mud and blood, sweat and water, and curses, as well as continuous enemy action, called for a high degree of technical skill.

Brigadier W. G. R. Nutt :* I would like to pay a tribute to the Tunnellers of Gibraltar. With the exception of the Canadian Company, the rank and file were almost all coal miners. Their dogged persistence and their devotion to their ideals and tradition were typical of all that is best in the British miner. Most of the members of the Institution will know what these ideals are, but to me, as a newcomer, they were a continual source of admiration.

A particular characteristic of the officers was that they were never satisfied that a thing could not be done better or more safely. There was a continual restless quest for improvement, which resulted in work of a high quality with a very low accident rate. I am proud to have been in command of such officers and men.

*Chief Engineer, South Burma District, formerly Chief Engineer Gibraltar.

FURTHER CONTRIBUTED REMARKS

ON

Some Aspects of Rock-Drilling Practice (The Witwatersrand Goldfield).*

By C. J. Irving, *Member.*

Mr. J. B. Richardson: Several contributors to the discussion, both in South Africa and this country, on Mr. Irving's interesting and painstaking paper pointed out the desirability of a plastic explosive that could be easily squeezed into the drill-hole to avoid air spaces between the explosive and the hole. Mr. McDouall, for instance, pointed out that the gelignite cartridge in common use has a stout wrapper that is difficult to rupture. The present type of waterproofed wrapper is, of course, designed to keep the explosive mixture in good condition under humid conditions, as well as to contain it, owing to the hygroscopic nature of the major constituent, ammonium nitrate. As Mr. Alec Jones reminds us, a type of wrapper has been in use for some time in the United States that has lines of indentations, similar to those in some food and sweetmeat wrappers, which wrappers are also for the purpose of keeping the contents in good condition under humid conditions.

Wrappers on these lines, containing a plastic explosive, could be easily developed, but it is doubtful if a highly plastic explosive could be produced at the same price as the ammo-gelignites in normal use and the price of explosives is an important factor in the cost of hard-rock mining.

An alternative suggestion is a granular explosive with good flow properties, tightly compressed into a wrapper with part-perforations in a spiral form, the lines of such part-perforations being an inch or less apart. As explosive cartridges are not usually exposed to extreme rough handling and certainly not to a strong compressive blow until placed in the drill-hole such a wrapper would withstand normal underground treatment and yet be easily ruptured on being struck a normal blow with the tamping rod so that the free-flowing powder would completely fill the cavity.

If these artificial lines of weakness in a spiral form proved to be expensive to make, parallel lines at an inch or less apart would probably serve equally well.

**Bull.* 474, September, 1945.

It is doubtful if double wrappers as suggested by Mr. McDouall would solve the problem in practice, because the responsibility for removing the outer wrapper would be with the miner charging the hole and lack of time or other circumstances might frequently prevent the removal and a check by officials would be difficult. In any case simplification and not the introduction of extra sub-operations should be the aim in underground practice, as the author infers throughout his paper.

Apart from the fact that there is room for doubt that jack-hammers, with or without detachable bits on the 'jumper', will ever produce other than a slightly conical form in a 42-in. hole in hard rock, the author's own suggestion of explosive cartridges of several diameters, even though well distinguished by colour of wrapper or some other means, is a complication to be avoided. It would lead to further problems in organizing the collection, return and redistribution of surplus explosive.

It was interesting to read the contribution by Mr. Theo Meyer, who indirectly referred to the material that he found most suitable as stemming for sealing and containing the explosive charge. It is good to know that he arrives at the same general conclusion as Mr. Willson and I did before the war,* which conclusion was subsequently confirmed a year or two ago in the U.S.A.† by the experiments carried out at the Mount Weather tunnel under the direction of Mr. Daniel Harrington: namely, that dry granular material is the best form of stemming.

A small point on the standardization of mining terms is that it is a pity that all English-speaking mining engineers cannot agree either to 'stem the tamping' or 'tamp the stemming'.

Mr. A. Haworth: The use of the modern jackhammer on mines is very widespread so that Mr. Irving's paper will have an equally wide sphere of interest. The subject is dealt with in a clear, painstaking manner and many of us now have a far clearer insight into the many things which practical experience had taught us to accept. The fundamental principles are laid before us and in many cases the accepted practice is found to be far from perfect. For example, it appears to be fundamentally wrong to be drilling a hole with a 1¼-in. bit and using a 1-in. diameter explosive to blast the hole.

*RICHARDSON, J. B., and WILLSON, J. D. 'Stemming in Metal Mines'. *Trans.* Inst. Min. & Met., vol. 48, 1938–9, pp. 349–73.

†JOHNSON, J. A., AGNEW, W. C., and MOSIER, McH. U.S. Bureau of Mines Reports of Investigations 3509, 3528, 3612. 'Stemming in Metal Mines' (Progress Reports 1, 2 and 3, 1940 and 1942).

With the principles of drilling so clearly defined we can look ahead and find interest in speculating on future developments.

First let us consider the 1-in. explosive in the hole drilled with the 1¼-in. bit. The speed of drilling which may be practical and the disruptive force required to break the rock depend very much indeed on the hardness and toughness of the rock to be drilled and broken. Rocks vary very considerably in hardness and toughness, but the quartzite of the Central Rand can be accepted as a good standard rock. The best diameter of the hole required to break the rock (Central Rand quartzite) will, no doubt, be a subject of considerable argument, especially if the economics of drilling are to be taken into account. The average hole drilled on the Rand begins at about 1¼ in. and finished at a little over 1 in. It is a taper hole averaging 1⅛ in. in diameter and it would be reasonable to accept 1⅛ in. as being the diameter of the hole required for stoping purposes.

The taper hole does not impress one as being very efficient—the shape itself would call for very careful tamping of the explosive and to be compelled to use a small diameter of explosive because the deepest portion of the hole is too small for any larger size rather suggests that something ought to be done whereby the part of the hole which is to contain the explosive is drilled specifically for the size of the explosive.

The first half of the hole could be drilled according to the rules which promote the fastest drilling speed. The first half of the hole is only the means whereby we get to the deeper half of the hole which will contain the explosive. The deeper half deserves some consideration. This portion needs to be as nearly as possible the same gauge from start to finish. With detachable bits some special shape (even the cruciform bit with ⅛-in. cutting pieces added at the outer edge of the bit set between the wings) could be devised to guarantee that the loss of gauge would not exceed $\frac{1}{16}$ in. In other words a bit could be used which would guarantee a full 1⅛ in. at the bottom of the hole and have not more than one-sixteenth taper in the portion where the explosive charge was placed.

There would be no real difficulty in getting that guarantee. It would cost a little to get it relative to the cost of a taper hole, but the advantage to be obtained (quite probably 20 per cent in disruptive force) would be worth the extra cost. The bit which could produce the small-taper hole would be too valuable to use on the first half of the hole and this would involve two-stage drilling.

It is easy to visualize two machines—one using a starter stem and going over the stope face, drilling all the holes to a depth of say 20 in., then another machine taking over, to drill small-taper deeper halves with far more careful attention to any signs of loss of gauge on the bits used. The explosive required would be 1⅛ in., which the makers could easily reintroduce. The proposed two-stage drilling is not new to the mines. It is noted that some mines—e.g., the Consolidated Main Reef—already adopt two-stage drilling.

The use of two machines may not, at first, be favoured, but a possible advantage would follow. There would be no necessity to change the length of the jumper, the length of stem could remain constant, and the shank and the stem could be regarded as standard parts of the jackhammer.

A considerable amount of trouble arises at the present time due to the water tube of the machine, but if the shank and the stem need not be changed during the shift the water tube and the shank need not be separate units, the water tube could rotate with the shank (could, in fact, be threaded into the shank), and the position be taken care of by a water seal in the back end of the machine. No air would find its way into the water, supplied at the bit, the machine could be thoroughly well lubricated, and there would be no danger of a water cushion between the piston and the shank. Once the shank and stem of the jumper can be looked upon as a part of the machine design it would be within the capacity of the makers to provide far better machines with dust, water and oil well under control.

In conclusion, it is difficult to avoid the comment that so valuable a paper makes no reference to costs. It is common practice to use the standard of cost per foot drilled or cost per 1,000 ft. drilled and there is a tendency to avoid the more important figure of cost per ton broken. The main object of rock-drilling is to break rock and attention to the more detailed refinements of rock-drilling must maintain close contact with the economics of breaking rock.

FURTHER CONTRIBUTED REMARKS

ON

The Nationalization of Mineral Rights in Great Britain. —Part II.*

By W. R. Jones, *Member.*

Mr. W. W. Varvill: When reading this important paper and the subsequent discussion one is struck by the extraordinary complexity of the subject, owing to the wide variety of interests, minerals, and systems of tenure involved in any scheme of nationalization. It appears to me that any scheme, to be successful, must be simple, elastic, and to some extent drastic. It must not enter too much into details and definitions, which only serve to provide new and rich pastures in which the legal profession can exercise their extraordinary talents for substituting the letter of the law for its spirit.

Owing to long absence abroad I am not conversant with the procedure followed when the Coal Royalties were nationalized, and the following is submitted as a simple way of carrying out the task:

(1) Each district, comprising one or more counties, should have a District Minerals Commission, the duty of which it would be to hear claims for compensation. The Commission would be presided over by a retired County Court judge and would consist of a lawyer. an estate agent, and a mining engineer.

(2) A National Appeal Tribunal would also be set up, whose duty it would be to lay down in broad outline the general lines upon which compensation should be awarded, for the guidance of the District Commissions.

(3) The National Appeal Tribunal would hear and give rulings on appeals from the District Commissioners.

(4) The decisions of the Appeal Tribunal on all such appeals would be final.

(5) The costs of all unsuccessful appeals both to the District Commissioners and to the Appeal Tribunal should be borne by the appellants. This would provide some safeguard against claims being entered on an inadequate basis.

* *Bull.* No. 474, September, 1945.

(6) A simple Act of Parliament would enable the State to take over all Mineral Rights, subject to compensation being paid in genuine cases of loss of tangible assets.

If some such procedure is not adopted there will arise such a deluge of claims from all and sundry that the only beneficiaries from nationalization will be the lawyers. The case will be far more complex than it was in the case of coal, owing to the very large number of minerals of economic value. The suggestions made here may appear to be drastic and to ride roughshod over the rights of the individual, but this is now being done every day in many directions.

The country is now faced with a national economic crisis far worse than that which was faced after the last Great War. Then, in the interests of national economy, disabled war victims suffered many injustices in the curtailing of disability pensions and gratuities and had very limited rights of appeal, and that was under a Conservative Government. The great majority accepted their troubles, doubtless with much grumbling, but knowing in their hearts that it was as inevitable as being called up for military service.

The owners of Mineral Rights are in the same predicament and will no doubt grumble, but accept the change patriotically. After all, should a man's body be considered to be of less importance than his property ? It should be possible to give generous compensation for Mineral Rights which are still productive of revenue, whilst ruling out vague claims based on past records or prospects which can only be realized under a system of national ownership.

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NOTICE OF GENERAL MEETINGS.

The FIFTH ORDINARY GENERAL MEETING of the Fifty-fifth 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, MARCH 21st, 1946, at 5 o'clock p.m.

The following Papers, copies of which are attached hereto, will be submitted for discussion:

A Study of Sizing Analysis by Sieving; and
A Comparison of Methods of Measuring Microscopical Particles.
By Harold Heywood, Ph.D., M.Sc.

Application of Sizing Analysis to Mill Practice.
By Harold Heywood and E. J. Pryor, *Member.*

Crushing and Grinding Efficiencies.
By T. K. Prentice, *Member.*

The SIXTH ORDINARY GENERAL MEETING of the Session will be held, also in the Apartments of the Geological Society, on THURSDAY, APRIL 11th, 1946, at 5 o'clock p.m., when the following Paper, of which a copy is attached, will be submitted for discussion:

A Survey of the Deeper Tin Zones in a Part of the Carn Brea Area, Cornwall.
By Brian Llewellyn, *Member.*

Light refreshments will be provided at 4.15 p.m. for members and friends attending the Meetings.

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 Council desire to remind members that in addition to the more comprehensive types of papers for discussion at General Meetings, they welcome for publication in the *Bulletin* short notes recording data or describing technical experience, which may be of general interest and value. Such notes are governed by the same rules in regard to acceptance as ordinary papers, but would be open for discussion by correspondence.

FIFTY-FIFTH SESSION: 1945–1946.
DATE OF ANNUAL GENERAL MEETING.

The Annual General Meeting of the Institution will be held at Burlington House on Thursday, May 16th, 1946.

ELECTION OF NEW PRESIDENT.

At a Meeting of the Council held on January 10th, 1946, Mr. C. F. Laycock, *M.C.*, was elected President of the Institution for the remainder of the Session 1945–46, in succession to the late Col. Edgar Pam. Mr. Laycock had been Acting President during Col. Pam's long illness, and was President-Elect for the Session 1946–47.

'NOTES ON THE MINERAL RESOURCES OF THE MIDDLE EAST.'

This publication was issued in March, 1945, by the Middle East Supply Centre, Industrial Production Section, and was compiled by Major J. D. Boyd, R.E., and Mr. R. Duncan, *Associates.* It provides a summary of information on mineral resources in the Middle East collected from various sources, including inspection reports by officers of the Centre, publications by geologists of local governments and correspondence with local government departments and private firms. The territories concerned, which are grouped in alphabetical order, cover the area from Tripolitania on the west to Saudi Arabia and Iraq on the east, and there is an alphabetical index of minerals and the territories in which deposits occur. A copy of this 78-page cyclostyled foolscap publication is contained in the Institution's Library, and is available for loan to members.

LIBRARY SERVICE.

The Library has now been brought back to London, and applications for books should be addressed to the Librarian, I.M.M. and I.M.E. Joint Library, 424, Salisbury House, London, E.C.2. Books at present on loan should of course be returned to this address. Members who are unable to visit the Library and borrow books in person may still borrow them by post. It is regretted that periodicals cannot be lent.

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 January 10th, 1946 :—

To MEMBERSHIP—

Anderson, Robert Beattie (*Gatooma, Southern Rhodesia*).
Ingham, Frank Tinley (*Exmouth, Devonshire*).
Legge, Alan John Walker (*Entebbe, Uganda*).
MacKay, Donald Asire (*Penang, Straits Settlements*).
Shaw, Sydney Herbert (*Jerusalem, Palestine*).

To ASSOCIATESHIP—

Andrew, Robert Bruce (*Dunedin, New Zealand*).
Blau, Osmar Julius (*Sydney, N.S.W., Australia*).
Blight, Charles Garfield (*Rooiberg, Transvaal*).
Cameron, James (*London*).
Cave, Ronald Arthur (*Burnley, Lancashire*).
King, Austin Gerald (*Nkana, Northern Rhodesia*).
Oxley-Oxland, Richard (*Johannesburg, Transvaal*).
Spencer, Thomas Frederick Bridgewater (*Pilgrim's Rest, Transvaal*).
Wallace, Reginald Cathorwood (*Dunnottar, Transvaal*).

CANDIDATES FOR ADMISSION—*continued.*

The following have applied for admission into the Institution since January 10th, 1946 :—

To Membership—

Norris, Gerald Chad (*Tarkwa, Gold Coast*).

To Associateship—

Batni, Ranganatha Rao S. R. (*Robertsonpet, South India*).
Lloyd, Arthur L. (*Mount Morgan, Queensland, Australia*).
Mackenzie, William (*Silos de Calañas, Spain*).
Polglase, John Henry (*Camborne, Cornwall*).

To Studentship—

Pegg, Charles William (*London*).
Rao, C. E. Narayana (*Bangalore, South India*).
Stephens, John Nolan (*Mosgiel, New Zealand*).
Taute, Andries Hendrik (*Erfpacht, Cape Province*).

TRANSFERS AND ELECTIONS.

The following have been transferred (subject to confirmation in accordance with the conditions of the By-Laws) since January 10th, 1946 :—

To Membership—

Bott, Charles Arden (*Salisbury, Southern Rhodesia*).
Dempster, Eric Richard (*Mosaboni, India*).
Goode, Kenneth Burden (*Wembley, Middlesex*).
White, George Benn (*Oorgaum, South India*).

To Associateship—

Morgan, George Clifford (*Royal Engineers*).
Standerline, Geoffrey Victor (*Egremont, Cumberland*).
Winsor, Thomas Henry (*Que Que, Southern Rhodesia*).

The following have been elected (subject to confirmation in accordance with the conditions of the By-Laws) since January 10th, 1946 :—

To Membership—

Richardson, Clement Mason (*London*).

To Associateship—

Lambert, Hugh Henry John (*Royal Engineers*).
Oxford, Desmond de Villiers (*Luanshya, Northern Rhodesia*).
Trounson, John Hubert (*Redruth, Cornwall*).
Ward, Jack (*Penhalonga, Southern Rhodesia*).

To Studentship—

Bridger, Denis (*Camborne, Cornwall*).
Dyson, Peter Gerald (*Peterborough, Northampton*).
Hammett, Peter Henry John (*Falmouth, Cornwall*).
Moon, William Raymond Collins (*Truro, Cornwall*).
Rees, John David (*Holywell, Flintshire*).
Royle, Paul Gordon (*London*).
Wilson, Alan James (*Oorgaum, South India*).
Wilson, Walter Joakim (*Camborne, Cornwall*).

MEMBERS ON SERVICE WITH H.M. FORCES.

A full list was published in BULLETIN *No.* 456, *September,* 1942. *The following additions or changes are supplementary to those already published.*

ASSOCIATE.

Major H. Dudley, *Royal Engineers* (Promoted).

STUDENTS.

Captain G. E. A. Banfield, *Royal Engineers* (Promoted).
Captain P. C. M. Bathurst, *Royal Engineers* (Promoted).
Captain D. H. Brook, *Q.V.O. Madras Sappers and Miners* (Promoted).
Captain A. Peck, *Royal Engineers* (Promoted).
Major R. R. B. Pitt-Chambers, *Royal Bombay Sappers and Miners* (Promoted).
Captain P. H. Rumsey, *Royal Engineers* (Promoted).

ROLL OF HONOUR.

Acting Wing Commander R. A. Atkinson, D.S.O., D.F.C. and Bar (*Associate*), *Royal Air Force.* (Previously reported missing from air operations, December, 1944; now presumed to have lost his life.)

Major C. R. Pullinger, M.C. (*Student*), *The Green Howards.* (Died of wounds, November, 1943.)

Captain H. S. Duncan (*Associate*), *F.M.S.V.F.* (Reported to have died of dysentry and injuries in June, 1943, at Kanyu, while a prisoner of war in Japanese hands.)

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,* having terminated his appointment with the Commonwealth Department of Munitions, has left Melbourne for Western Australia. He hopes to visit England this year.

Mr. L. D. AIREY, *Student,* has left Nigeria and joined the staff of Mufulira Copper Mines, Ltd., Northern Rhodesia.

Mr. N. M. AIREY, *Associate,* has been demobilized and has joined the staff of Rand Leases (V) Gold Mining Co., Ltd., Transvaal, as mine overseer.

Mr. H. DOUGLAS ALLEN, *Member,* has been appointed manager of the Overseas Engineering Company in West Africa.

Mr. R. E. ARCH, *Student,* has joined the staff of Ingersoll (Engineering) Co., Ltd., in London.

Mr. F. M. BALL, *Associate,* has relinquished his appointment with the Ministry of Aircraft Production.

Mr. G. P. BENNETT, *Associate,* has been released from the Forces and has returned to Randfontein Estates gold mine.

Mr. C. W. F. BOND, *Associate,* is in London on leave from the Gold Coast.

Mr. E. E. G. BOYD, *Member,* has resigned his position as managing director of Austral Malay Tin, Ltd., and is at present in Australia.

Mr. W. T. MEREDITH BROWNE, *Associate,* has left England for Geita, Tanganyika.

Mr. M. W. BRYCE, *Associate,* has left Scotland to join the Geological Survey, Northern Nigeria.

NEWS OF MEMBERS—*continued.*

Mr. F. C. CALVERT, *Associate*, has returned to Malaya from New Zealand.

Mr. W. N. CARNAGHAN, *Associate*, has left England to return to Sierra Leone.

Mr. F. B. CHAMPNESS, *Associate*, has arrived in England from Australia.

Mr. J. T. CHAPPEL, *Associate*, is leaving Australia on a short visit to England, and hopes to arrive in April.

Mr. W. H. COLLINS, *Associate*, is returning to England on furlough from Nigeria.

Mr. D. H. CRAMPTON, *Associate*, has been released from the Royal Navy and has resumed his position as manager of the Turkois mine, Southern Rhodesia.

Dr. D. A. BRYN DAVIES, *Associate*, will be at Kumasi, Gold Coast, until July, 1946.

Mr. E. B. DAVIES, *Associate*, will leave England shortly on his return to Malaya to resume duties with the London Tin Corporation, Ltd.

Mr. T. DOWELL, *Associate*, is returning to England from Nigeria.

Mr. I. S. DUDMAN, *Student*, has been released from the Forces and has returned to New Modderfontein Gold Mining Co., Ltd., Transvaal.

Mr. R. DUNCAN, *Associate*, has been discharged from the Army and has accepted an appointment as assistant mine manager to the Egyptian Phosphate Co.

Mr. T. EDWARDS, *Student*, has been released from the Forces and has returned to Crown Mines, Ltd., Transvaal.

Mr. G. H. FAIRMAID, *Member*, now on leave in New Zealand, will return to the Pahang Consolidated Co., Ltd., later in the year.

Mr. C. H. FELDTMANN, *Member*, has left England on his return to Tanganyika.

Mr. K. A. FERN, *Associate*, who has been released from the Army, has joined the research staff of Associated Portland Cement Manufacturers, Ltd.

Mr. F. GILBERT, *Associate*, has returned to England after his release from internment in Malaya.

Mr. DONALD GILL, *M.C.*, *Member*, has resumed his practice in London after having served with the British Economic Mission in Algiers, Ministry of Supply.

Mr. S. D. GORDON, *Associate*, has returned to England after his release from the Royal Indian Naval Reserve.

Capt. C. E. GREGORY, *Member*, has returned to Australia for demobilization.

Mr. M. GREGORY, *Member*, is returning to England from Nigeria.

Mr. E. L. HALLÉ, *Associate*, has left England for Bolivia.

Mr. H. C. HANNAY, *M.C.*, *Member*, who was previously interned at Singapore, is now working for the British Military Administration as Custodian of Property, Ipoh, F.M.S.

Mr. E. P. HARGRAVES, *Member*, has left England for Malaya.

Mr. G. M. P. HORNIDGE, *Member*, has left England for the Federated Malay States.

Dr. W. R. INGALLS, *Member*, has been elected an Honorary Member of the Mining and Metallurgical Society of America, of which he is a past-president.

NEWS OF MEMBERS—*continued.*

Dr. F. T. INGHAM, *Associate*, is in England after his release from internment in Singapore.

Mr. E. H. JAQUES, *Associate*, has left England on his return to Nigeria.

Mr. A. R. JONES, *Associate*, has joined the staff of the Cementation Co., Ltd., on leaving the Directorate of Opencast Coal Production, Ministry of Supply.

Mr. A. JOSE, *Student*, has left England to join the staff of Gold Coast Main Reef, Ltd.

Mr. F. H. LATHBURY, *M.C.*, *Member*, has returned to Nairobi from England.

Mr. T. LEARMONT, *Student*, has been released from military service and has returned to New Zealand.

Mr. O. MCCULLOCH, *Associate*, at present in England after release from the Royal Air Force, has accepted the position of chief surveyor, Frontino Gold Mines, Ltd.

Mr. R. A. MACKAY, *Associate*, has left the Nigerian Mines Department to take up the appointment of mining geologist to the Nigerian Geological Survey.

Mr. R. K. MCLEOD, *Associate*, has arrived in England on leave from Northern Rhodesia.

Mr. E. P. MEATON, *Student*, has left South Africa for England.

Mr. C. F. MEES, *Associate*, has left England for Geita, Tanganyika.

Mr. H. W. MILLETT, *Associate*, has left England to take up an appointment with Ariston Gold Mines (1929), Ltd., Gold Coast.

Mr. J. P. MILLS, *Associate*, has been released from the Royal Air Force and has returned to Cornwall.

Mr. H. L. MITCHELL, *Student*, has left England to join the staff of the Consolidated African Selection Trust, Ltd., Gold Coast.

Mr. P. G. F. MONEY, *Associate*, has been released from the Royal Canadian Air Force and has returned to Kirkland Lake, Ontario.

Mr. GEORGE NEW, *Associate*, is expected to leave England soon on his return to Taquah and Abosso Mines, Ltd., Gold Coast.

Mr. A. P. NEWALL, *Associate*, has left England on a year's visit to Australia.

Mr. G. C. NORMAN, *Associate*, has returned to the Gold Coast from England.

Mr. J. E. OGILVIE, *Associate*, has left England on his return to Malaya.

Mr. R. C. PARGETER, *Student*, has been released from the Royal Air Force and has left England to join the staff of the Geological Survey, Uganda.

Mr. A. L. PARMA, *Associate*, has returned to England from the Gold Coast.

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

Mr. T. I. PINER, *Student*, has joined the staff of the Sierra Leone Chrome Mines, Ltd.

Dr. WILLIAM PULFREY, *Associate*, has been awarded one moiety of the Lyell Fund by the Geological Society of London.

Mr. J. D. REES, *Student*, expects to be released from military service shortly, when he will resume his former post with Halkyn District United Mines, Ltd.

Mr. D. F. REEVES, *Student*, has arrived in England on leave from Nigeria.

NEWS OF MEMBERS—*continued.*

Mr. A. M. Robinson, *Member*, has been appointed Mining Consultant to the Government of Tanganyika Territory on the termination of his employment as Assistant Director, Opencast Coal Production, Ministry of Supply.

Mr. A. S. Robinson, *Associate*, has been demobilized and has returned to his duties with Consolidated African Selection Trust, Ltd.

Mr. W. E. Sinclair, *Associate*, has left Kimberley on his appointment as manager of the Cape Asbestos Co.

Mr. A. F. Smith, *Associate*, has left Australia and returned to Malaya.

Mr. F. N. Spettigue, *Member*, has been released from the Forces, and has left England to rejoin the staff of Rio Tinto Co., Ltd., Spain, as chief mining engineer.

Mr. J. E. Struthers, *Associate*, has left Rezende Mines, Ltd., to join the staff of the Government Metallurgical Department, Gwelo, Southern Rhodesia.

Mr. C. B. Taylor, *Member*, has returned to the Kolar Gold Field from England.

Mr. E. H. Taylor, *Associate*, is now serving with the Malayan Administration.

Mr. S. H. J. Taylor, *Associate*, having been released by the South African Naval Forces, is now employed at Springs, Transvaal.

Mr. J. B. Toms, *Associate*, has left England to join the staff of the Cyprus Mines Corporation.

Mr. J. W. C. Treeby, *Associate*, having been released from internment in Singapore, has resumed his consulting practice at Ipoh before proceeding on home leave this year.

Mr. W. R. Trethewey, *Associate*, is returning to England from Brazil.

Mr. J. E. G. Williams, *Associate*, has left England for Nigeria.

Mr. C. C. Walker, *Associate*, has been demobilized and has returned to the staff of Sierra Leone Selection Trust, Ltd.

Mr. C. W. Walker, *Associate*, has arrived in England on leave from the Gold Coast.

Mr. W. D. Wallace, *Student*, has returned to Southern Rhodesia on his release from the Forces.

Lt.-Col. J. Weekley, *Member*, Commanding Perak Local Defence Force, was awarded the O.B.E. in the New Year Honours for services rendered during his internment.

Mr. F. T. M. White, *Member*, has been transferred from the Colonial Mines Service, Fiji, to the Malayan Administration.

Mr. A. Hedley Williams, *Associate*, has been released from his duties as Regional Controller, Ministry of Supply.

Mr. Arnold S. Williams, *Member*, has left England on a visit to Nigeria.

Mr. F. H. Williams, *Member*, has taken an advisory post with the Ministry of Supply in connection with the rehabilitation of the mines of Malaya and Siam.

Mr. W. Broadhead Williams, *Associate*, has left England for Geita, Tanganyika.

REVIEW.

Chronic Pulmonary Disease in South Wales Coalminers: Medical Research Council Special Report Series No. 250. III.—**Experimental Studies:** A. Report by the Committee on Industrial Pulmonary Disease. B. The mineral content of the lungs of workers from the South Wales coalfield. By E. J. KING and G. NAGELSCHMIDT. C. The estimation of coal and of aluminium in dried lung. By E. J. KING and MARGARET GILCHRIST. D. Tissue reactions produced experimentally by selected dusts from South Wales coalmines. By T. H. BELT and E. J. KING. E. The solubility of dusts from South Wales coalmines. By E. J. KING. Paper boards, 94 pp., illustrated. London: H.M. Stationery Office, 1945. Price 5s.

This report is the third published by the Medical Research Council in their Special Report series and the Committee on Industrial Pulmonary Disease are to be congratulated on the excellence of the three reports. The first report (No. 243. 1942) dealt with the clinical and pathological aspects of the disease. The second (No. 244. 1943) dealt with the physical and chemical nature of the dust to which the worker is exposed and to the chemical and petrological characteristics of the coal seams.

King and Nagelschmidt have reported on the chemical analysis of the lungs of 54 workers from the South Wales coalfield. Their analysis dealt with silica, alumina and coal after isolation of the residues and determined the quartz, mica and kaolin by X-ray diffraction pattern analyses. They were able as a result of their investigations to show a correspondence between employment, *e.g.*, collier, hard heading worker, and the mineral content of the lungs. In the case of the anthracite collier the analysis of the lung dusts corresponded almost identically with the dust at the coal face. The interesting point of their observations, and one which might be anticipated, is that the concentrations of quartz and mica ran more or less parallel with the degree of fibrosis present in the lungs. No relationship between fibrosis and the concentration of coal and kaolin was found. Their final conclusion was that quartz was the major etiological agent in production of pulmonary fibrosis.

In section C, King and Gilchrist discuss the methods for the estimation of the coal and alumina in lung tissue residues. They investigated experimentally several methods (Badham and Taylor, Hydrogen Peroxide Digestion, Multiple Soda Treatments) and finally adopted a method involving two soda digestions. They found the accuracy of this method to be generally within $\pm$ 5 per cent.

The aluminium content of the ashed lung was determined by the precipitations of the metal as aluminium hydroxyquinoline. This method is described in detail.

D.—Belt and King investigated in much detail the tissue reactions produced experimentally by selected dusts from South Wales coalmines. In the original report of this series (No. 243. 1942) it was shown that the pneumokoniosis of coal miners is produced by a mixture of dusts. The authors of this section planned their investigation with the object of assessing the relative harmfulness of these dusts in so far as they could be separated and tested as components of the whole mixture. The method of introducing the dust into the lung was the well-known one of Kettle

REVIEW—*continued.*

and Hilton. The authors summarized their investigations and classified the dusts used into pure coal, dusts of mixed composition, stone dusts, isolated fractions, artificial mixtures of pure quartz and various mine dusts, and mineral matter recovered from human lungs. As already mentioned the essential finding is that silica appears to be the true fibrosis-producing substance, but the authors stress that tissue reaction to silica is not a simple matter but a complex biological phenomenon. This section is excellently illustrated by a number of photomicrographs.

E.—The solubility of dusts from South Wales coalmines. In view of recent work in Canada and America on the prevention of silicosis by the inhalation of finely divided aluminium and alumina, this section written by Earl J. King is of great importance. King discusses the various factors influencing the solubility of silica and the literature of silicic acid. He shows that the South Wales dusts show a low solubility, a solubility which is depressed by the admixture of shale and other dusts. This depression in solubility is apparently due to the release of aluminium and the formation of a protective coating. King discusses the effect of various coal dusts on the solubility of silica and shows that there is considerable difference between the effect of anthracite, steam and bituminous coal.

The whole report is excellently written and produced and the Committee on Industrial Pulmonary Disease are to be congratulated on a fine piece of work.

A. J. AMOR.

OBITUARY.

Acting Wing Commander Richard Ashley Atkinson, D.S.O., D.F.C., Royal Air Force, is presumed to have lost his life on December 13th, 1944, having been reported missing from operations on that date. He was 31 years old and the son of Mr. John Atkinson, *Member*, of New South Wales. Born in Australia, he began his mining career with Katu Tin Dredging Co., Ltd., in West Siam, where he was engaged mainly in surveying from January to June, 1932. He then came to London and studied at the City and Guilds Engineering College for three years, graduating in 1935 with the Associateship of the College and the degree of B.Sc. in Electrical Engineering. He then entered the Royal School of Mines and on obtaining the A.R.S.M. in Mining in 1937, returned to Australia to join the staff of Wiluna Gold Mines, Ltd. From February, 1938, he was employed in contract mining and, later, in the surveying office at North Broken Hill, Ltd., N.S.W., but left in May, 1939, on his appointment as shift boss to Renong Consolidated Tin Dredging Co., Ltd., Thailand.

Having been a Flying Officer in the R.A.F. reserve since 1933, Mr. Atkinson was called up immediately on the outbreak of war and attached to the Royal Australian Air Force, serving as Squadron Leader in the Pacific area of operations until his posting to Coastal Command of Great Britain in 1944. For his work in the Pacific he was awarded the D.S.O. and D.F.C. with bar. He was in command of a Mosquito squadron attached to Coastal Command when his aircraft failed to return to base from an anti-shipping operation off the coast of Norway.

OBITUARY—*continued.*

Wing Commander Atkinson was elected to Studentship of the Institution in 1936 and was transferred to Associateship in 1944. He was also an Associate of the Australasian Institute of Mining and Metallurgy.

Vincent Brice Carew Baker died in the jungle in Malaya on April 29th, 1944, at the age of 56. He received his professional training at the Camborne School of Mines from 1906 to 1909, obtaining a first-class Diploma of the School. He was awarded a first-class certificate, Cutler's Company's Prize and the Institute's Silver Medal in the City and Guilds of London Institute's examination in 'Occurrence, Raising and Dressing of Ores', and held an I.M.M. Post-Graduate scholarship at the Great Boulder Proprietary mine, Kalgoorlie, W.A., from 1909 to 1911. For seven months subsequently he was employed in prospecting in Western Australia for the same company, but in February, 1912, obtained an appointment as surveyor to Pahang Consolidated Co., Ltd., in the Federated Malay States. He became underground manager in November, 1913, of three of the company's mines, becoming ten years later acting general superintendent and, in 1929, general manager on their lode tin mines in Malaya. When the Japanese overran the country in December, 1941, he managed to escape into the jungle with his sister, and they eventually joined with the Chinese guerillas. They both fell ill with malaria, however, and with the approach of the Japanese had to be left behind as the Chinese withdrew. They were not captured, but Mr. Baker died a few weeks later and his sister rejoined the guerillas and fought with them until the Japanese surrender.

Mr. Baker was elected a Student of the Institution in 1909, and was transferred to Associateship in 1915 and to Membership in 1924.

William John Barnett died on November 8th, 1945, at the age of 77. He was a student at the Camborne School of Mines for four years, where he won the Medal of the Mining Association of Cornwall and Devon, and then worked for four years in Cornish mines. He took up the appointment of assistant general manager to La Joya copper mines, Huelva, Spain, in 1892, and a year later joined the Spes Bona Gold Mining Co. in the position of assistant manager and assayer, leaving after eight months to become head assayer to the Paarl Central Gold Mining and Exploration Co., Ltd., Johannesburg, where he also did private reporting. From 1896 to 1900 he was employed as mining engineer to Consolidated Mines Selection Co., Ltd., and then spent some months examining mines in New Caledonia and Australia. For many years he was in practice as a consulting engineer, with headquarters in London, but retired from active work after the 1914–18 war. He was a director of the Poderosa Mining Co., Ltd., and of other concerns at the time of his death.

He was elected to Studentship of the Institution in 1893, and was transferred to Associateship in 1896 and to Membership in 1902.

Selwyn Gwillym Blaylock died at the age of 66 in the Trail-Tadanac Hospital, Canada, on November 19th, 1945, after a long illness. He was born in Paspobiac, Quebec, and was educated at Bishop's College School, Lennoxville, and McGill University, graduating in 1899 with the B.Sc. degree in mining and metallurgy. He obtained an appointment in British

OBITUARY—*continued.*

Columbia as assayer and junior chemist at the Trail plant of the Canadian Smelting Works, and became chief chemist at the plant in 1901 and metallurgist in 1905. Upon the acquisition of the Trail smelter in 1906 by the Consolidated Mining and Smelting Company of Canada, Ltd., Mr. Blaylock began his long association with that company, and was appointed chief superintendent at the Hall mines smelter in Nelson in 1907. He was transferred to the St. Eugene mine, Moyie, in 1908, and the following year, in company with Mr. R. H. Stewart, made an examination of the Sullivan mine which was subsequently acquired by the Consolidated Mining and Smelting Company of Canada, and he was appointed superintendent of that mine also until 1911. He was recalled to Trail as assistant general manager, and during the 1914–1918 war Mr. Blaylock's great efforts successfully increased the production of zinc at the Trail and Tadanac plants. He became general manager of the Company in 1919, director in 1922, vice-president in 1927, managing director in 1938, and president and managing director a year later. He was appointed chairman and president in 1943, but owing to ill health resigned the presidency of the company while remaining chairman of the Board. He was a director of the Bank of Montreal and several mining and metallurgical companies, and had been a director of the Canadian Pacific Railway Co.

His outstanding ability was recognized everywhere. In 1924 he was awarded the McCharles Prize of the University of Toronto, and in 1928 he received the James Douglas Medal of the American Institute of Mining and Metallurgical Engineers, of which he was elected an Honorary Member in 1944. In 1929 and in 1930 he had conferred on him the honorary degree of LL.D., first by McGill University and then by the University of Alberta. He was a member of the Board of Governors of the former. He was awarded the Platinum Medal of the International Nickel Co. of Canada, Ltd., by the Canadian Institute of Mining and Metallurgy, of which he had been a member since 1907 and on whose Council he had served for two periods. He was President of that Institute during the session 1934–35, and was designated an Honorary Member in February, 1945.

Mr. Blaylock was elected a Member of the Institution in 1925 and was awarded the Gold Medal of the Institution in 1940 in recognition of his outstanding achievements in advancing the science of metallurgy in the Dominion of Canada. With others he contributed a paper on 'A Short History of the Discovery and Development of the Sullivan Mine' to the Empire Mining and Metallurgical Congress held at Wembley in 1924 (*Transactions* I.M.M., vol. 34, Part 2, 1924–25).

Ernest William Byrde, *M.C., Croix de Guerre (France)*, died at Kaduna, Nigeria, on February 2nd, 1946, at the age of 68. He was a student at the Royal School of Mines from 1896 to 1901, and graduated in 1900 with the A.R.S.M. in Mining. He left England to join the staff of the Borneo Co.'s Bidi gold mine in Sarawak, where he remained for four years. He was on the Gold Coast from 1905–6 as assistant metallurgist at Abbontiakoon Block I, and visited Norway and Siberia professionally in 1907, joining the staff of Cie des Mines de Siguiri, Haute Guinée, in the same year. From 1909 to 1911 Mr. Byrde was assistant and acting manager at Mawchi tin

OBITUARY—*continued.*

and wolfram mines of the Shan States Syndicate, and for the following two years was employed by Naraguta Tin Mines, Northern Nigeria, where he also held the position of Honorary Secretary of the Nigerian Chamber of Mines. In 1913 he did some surveying on the Lafon river, and then spent a year in British Columbia on the Sullivan mine, Kimberley, and in 1914 became manager of a copper mine of the Anglo-Orient Syndicate in Turkey. During the 1914–18 war he served with the Royal Engineers in France and Flanders, being commissioned in 1915 and attaining the rank of major. He was mentioned in despatches, and was awarded the Military Cross and Croix de Guerre (France) with Citation. From 1920–21 he took a refresher course at the Royal School of Mines and was then appointed manager of the Kuru Syndicate (later the Jantar Company), Northern Nigeria. He subsequently operated a mine in Nigeria on his own account and was in private practice at Jos until the time of his death.

Mr. Byrde was elected a Student of the Institution in 1901 and was transferred to Associateship in 1904.

Captain Harry Stuart Duncan, F.M.S.V.R., is reported to have died of dysentry and injuries while in a Japanese prisoner-of-war camp at Kanyu on June 10th, 1943, at the age of 41. He received his professional training at the Royal School of Mines from 1919 to 1922, and was awarded a first-class Associateship of the School and the degree of B.Sc. (London) with honours in Metallurgy. He was thereupon engaged as an assistant in the works and laboratories of Messrs. Walkers, Parker & Co., Ltd., Newcastle-on-Tyne, until in May, 1925, he took up the appointment of assistant chemist in the Pulau Brani smelting works of the Straits Trading Co., Ltd., Singapore, transferring to the position of works assistant two years later. He remained with the company until 1933, when he joined the staff of Anglo-Malayan Tin, Ltd., Rawang, F.M.S., and was at Rawang Tin Fields from 1937. News was received in 1943 that he was a prisoner of war in Camp No. 4, Thailand, and the report of his death was not received until January, 1946.

Mr. Duncan was elected a Student of the Institution in 1925, and was transferred to Associateship in 1928.

Walter John Joseph Franks died on October 25th, 1945, at the age of 75. His whole career was with the firm of Messrs. D. C. Griffith & Co., assayers to the Bank of England, metallurgists and analytical and consulting chemists, which he joined in 1887. At that time the business was mainly concerned with the assaying of gold and silver bullion, but later he was partly responsible for the development of the ore and base metal side of the business, and in his capacity of technical manager did much to expand his company's activities. At his death he had been associated with the firm for 58 years.

He was elected a Member of the Institution in 1913.

Edmund William Janson died suddenly on August 18th, 1945, at Lundin Links, Fife, at the age of 77. He was educated at Uppingham School and Cambridge University, and became a pupil of Messrs. Edward Riley & Co., mining engineers and assayers, in 1891. A year later he

OBITUARY—*continued.*

began his mining training at the Camborne School of Mines, graduating with the Diploma of the School in 1895. During 1893 he had reported on the Pilleys Island, Newfoundland, pyrites deposits for Messrs. Edward Riley & Co., and in 1894, working for private interests, he visited the iron mines of Bilbao, the gold, lead and slate mines of North Wales, and the South Staffordshire coalfield. He was then appointed by the Don Pedro Gold Mines of Brazil as surveyor and assistant manager, and in 1896 he entered into partnership with Mr. E. M. Touzeau, consulting engineer, of London, subsequently representing his firm in New Zealand and Australia. In 1899 he visited the United States and reported on several mines in the Mother Lode of California, and in Montana, in conjunction with Mr. Richard Parker. From 1900 Mr. Janson was a partner of the late Mr. Percy Tarbutt, practising as Tarbutt, Son and Janson, the firm later being re-named Percy Tarbutt & Co., and he travelled extensively in his professional capacity. He retired from active mining work in 1930, but continued to act as a director of various companies, including the Associated Tin Mines of Nigeria, Ltd., the Consolidated African Selection Trust, Ltd., and the Jantar Nigeria Co., Ltd. Mr. Janson was elected to Membership of the Institution in 1909.

John Taylor Marriner died in Worcestershire on August 27th, 1944, at the age of 69. He began his career as a pupil at the Blackwall iron works of Messrs. John Stewart & Co., and then studied engineering at University College, London, for a period holding the position of demonstrator and assistant in the engineering department of the College. In 1896 he was employed for nine months in the laboratory of the Cassel Gold Extracting Co., Glasgow, leaving in 1897 for Western Australia to take up the position of metallurgist at the Eureka gold mine, Blackett's Mines, Ltd., Coolgardie. Later he was appointed metallurgist to Great Boulder Main Reef, Ltd., Kalgoorlie, and he was promoted to the position of manager in 1900. From 1903 to 1904 he took over the temporary management of British Corean Concessions, and in May, 1907, became general manager of The Duff Development Co. at Kelantan, Malaya. He left this position in 1911 on his appointment as general manager of Pahang Consolidated Co., Ltd., where he remained for many years. He retired in 1933, and resided in the Channel Islands until 1940.

Mr. Marriner was elected a Student of the Institution in 1896 and was transferred to Membership in 1907.

Lieutenant-Colonel Edgar Pam, O.B.E., died on December 20th, 1945, at his home at Virginia Water, Surrey, at the age of 63, after a long illness.

Col. Pam was educated at Harrow and received his professional training at the Royal School of Mines, from which he graduated in both mining and metallurgy in 1904. In November of that year he took up employment with Village Deep, Ltd., Johannesburg, first as assistant sampler and rising through the positions of assistant surveyor, chief sampler, chief surveyor, and shift boss to mine captain in May, 1909. During the rest of that year he studied collieries in the United Kingdom and hydraulic filling methods

OBITUARY—*continued.*

in Germany. From 1910 to 1912 he was employed as assistant to the consulting engineer, Central Mining and Investment Corporation, Johannesburg, in charge of sand-filling operations at mines controlled by that group. In 1912 he was appointed sectional manager of Geldenhuis Deep, Ltd., a few months later becoming manager. During the Great War he served first in the Royal Engineers (Tunnelling Companies) and later as Assistant Director General of Transportation, First Army and Army of the Rhine. He was mentioned in despatches and received the O.B.E. for his services. He returned to Geldenhuis Deep in 1920 and later became manager of Modderfontein East. In 1928 Col. Pam returned to England to take up an appointment as consulting engineer to the Mond Nickel Co., Ltd. When that company merged with the International Nickel Co. of Canada, Ltd., in 1929, he was appointed assistant to the delegate director of the Mond Nickel Co., and became deputy delegate director in 1939. At the time of his death he was a director of the company.

Col. Pam was elected to Studentship of the Institution in 1903, was transferred to Associateship in 1912 and to Membership in 1915. He had been a Member of Council since 1934 and held the office of Vice-President for the two periods 1936–39 and 1942–44. He was elected President of the Institution for the Session 1944–45 and re-elected for a second year of office in 1945–46. His death in office was a severe loss to the Institution and especially to his colleagues on the Council, who had hoped to have the benefit of his wise guidance in the difficult post-war period.

Major Carlos René Pullinger, *M.C.*, The Green Howards, died in November, 1943, of wounds received during the landing in Sicily. He was 25 years old. He entered the Royal School of Mines in 1935, and completed the course in mining geology in 1939, graduating with the A.R.S.M. and B.Sc. (Geology). He joined the Forces early in the war and was awarded the Military Cross. Major Pullinger was elected a Student of the Institution in 1937.

William Robert Wilson Ronald Scott died suddenly at San Jose, Costa Rica, on November 13th, 1945, at the age of 69. He was born in Scotland, and from 1894 to 1898 served an apprenticeship with Messrs. John & G. H. Geddes, mining engineers, of Edinburgh. He then entered the employment of Messrs. John Lancaster & Co., Ltd., first as assistant overman and later as assistant manager at their Blaina and Six Bells collieries, Monmouthshire. In July, 1899, he took up the position of manager of the Sabiwa mine at Granada, Rhodesia, in the service of Rice-Hamilton Exploration Syndicate, Ltd., until, in 1902, he became for two years engineer for the Gwanda district for Rhodesia, Ltd. In the autumn of 1904 he reported on properties in Mexico, later becoming general manager of the Premier Development Corporation of Mexico, Ltd., and in 1909 he returned to England where he was engaged as consulting mining engineer in London. He took up cotton planting in North Carolina in 1910, where he stayed for two years, and from 1912 to 1913 conducted a general importing business in Sydney. He resumed mining employment in 1913 at Webb's Consols mine, N.S.W., and carried out experimental work on molybdenite. He left Australia in 1916 to join the Royal

OBITUARY—*continued.*

Engineers, holding a commission in the 252 Tunnelling Coy., B.E.F. In 1923 he went to Costa Rica, Central America, where he remained until his death. He was elected an Associate of the Institution in 1917.

Frederick Percy Tremble died on December 25th, 1945, at the age of 51. He studied inorganic chemistry and metallurgy in the evenings at the South Western Polytechnic Institute, Chelsea, from 1911 to 1915 and from 1920 to 1922 while working for Messrs. Riley, Harbord and Law, of London, with whom he held the position of assistant chemist from November, 1910, to February, 1915. He then joined the Royal Navy, but resumed his employment in October, 1915, as assistant chemist and inspector. In January, 1922, he became manager of the Swansea branch of the firm, and remained there for nearly four years, leaving in December, 1925, on his appointment as senior assistant in the ore department of British Metal Corporation, Ltd., for whom he worked in England, Germany, France, Belgium and Italy. On the outbreak of war in 1939 he went to Rugby with the Non-Ferrous Metals Control, remaining there until early in 1943, when he returned to London to take up a position with Continental Metals & Minerals, Ltd., a subsidiary of British Metal Corporation. Mr. Tremble joined the British Economic Mission in North Africa on the direction of the Ministry of Supply, and worked in Algiers with the Minerals Division until his return to London in June, 1945, to resume his employment with British Metal Corporation. He was elected an Associate of the Institution in 1928.

The Council regret to report the death of **Walter Edgar Segsworth,** *Member*, on July 20th, 1945. An obituary notice will appear in a later *Bulletin*.

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

AIR COMPRESSORS: their installation, operation and maintenance. By Eugene W. F. Feller. 460 pp. New York and London: McGraw-Hill Book Company, Inc., 1944. £1 7s. ($4.50).

CANADA: GEOLOGICAL SURVEY PAPERS AND MAP: Paper 45–16: Canol geological investigations in the Mackenzie River area, Northwest Territories and Yukon (report and 3 maps), by G. S. Hume and T. A. Link. Paper 45–24: Saunders map-area, Alberta (report and map), by O. A. Erdman. Paper 45–25: Londonderry, Colchester and Hants Counties, Nova Scotia (preliminary map), by L. J. Weeks. Map 829A: Waterford, Kings and Saint John Counties, New Brunswick. Ottawa: Department of Mines and Resources—Mines and Geology Branch, 1945.

EL PARICUTIN, ESTADO DE MICHOACAN. Estudios Vulcanologicos, Instituto de Geologia, Universidad Nacional Autonoma de Mexico. 165 pp. Mexico: Imprenta Universitaria. 1945.

LIST OF ADDITIONS TO THE JOINT LIBRARY—*continued.*

GEOLOGICAL MAP OF THE DOMINION OF CANADA (scale 1 in. to 60 statute miles). Map 820A of Department of Mines and Resources, Mines and Geology Branch, 1945. 50 cents. (*Presented by the Bureau of Geology and Topography.*)

GEOLOGICAL MAP OF PALESTINE (scale 1:500,000). (*Presented by the Government Geologist, Palestine.*)

INFORME GEOLÓGICO Y MINERO DE LOS YACIMIENTOS DE COBRE DE AROA, ESTADO YARACUY. By Víctor M. López, John C. Davey, and Enrique Rubio. 138 pp. Caraças, Venezuela: Litografia del Comercio, 1944.

MANUAL OF MODERN UNDERGROUND HAULAGE METHODS FOR MINING ENGINEERS. By H. R. Wheeler. 67 pp., illus. Large crown quarto. London: Charles Griffin & Co., Ltd., 1946. 18s. (*Presented to the Institution of Mining Engineers by the Publishers.*)

MICA. By R. S. Matheson. 75 pp., with plans. Bulletin No. 2 of Western Australia Department of Mines (Mineral Resources of Western Australia). Perth: Government Printer, 1944.

MONMOUTHSHIRE AND SOUTH WALES COAL OWNERS' ASSOCIATION: SEVENTEENTH REPORT OF THE COAL DUST RESEARCH COMMITTEE—Water infusion: its application—the use of wetting agents; October, 1945. (*Presented to the Institution of Mining Engineers by the Director of Research, The Monmouthshire and South Wales Coal Owners' Association.*)

NATIONAL ASSOCIATION OF COLLIERY MANAGERS: MINUTES OF PROCEEDINGS, Vol. XLII, 1945. 268 pp. London: Published by *The Iron & Coal Trades Review*, 1946.

NEW ZEALAND MINES STATEMENT, 1945, by the Hon. J. O'Brien, Acting Minister of Mines. 38 pp. Wellington: Government Printer, 1945.

NOTES ON SPONTANEOUS COMBUSTION IN COAL MINES. By H. Price. 19 pp. Wigan: Thos. Wall & Sons, Ltd., 1945. (*Presented to the Institution of Mining Engineers by the Director, The Wigan Coal Corporation, Ltd.*)

PLANS FOR RECRUITMENT, EDUCATION AND TRAINING IN THE COAL MINING INDUSTRY. Prepared by R. W. Revans in conjunction with the Recruitment, Education and Training Committee of the Mining Association of Great Britain, with an introduction by Robert Foot. 114 pp. London: Published by the Mining Association of Great Britain, 1945.

PORTUGAL: COMUNICAÇOES DOS SERVIÇOS GEOLÓGICOS DE PORTUGAL. Tômo XXII and XXIII. 109 and 369 pp. Lisbon: Direcção Geral de Minas e Serviços Geológicos, 1941 and 1942.

REINFORCED CONCRETE DESIGN FOR ENGINEERING STUDENTS. By John Berry. 108 pp. London: Hutchinson's Scientific and Technical Publications, 1945. 10s. 6d.

LIST OF ADDITIONS TO THE JOINT LIBRARY—*continued.*

ROCK WOOL. By E. M. Guppy and James Phemister. 46 pp. Memoirs of the Geological Survey, Special Reports on the Mineral Resources of Great Britain, Vol. 34. London : H.M.S.O., 1945. 9d. (*Presented by the Director of the Geological Survey and Museum.*)

SOUTH AUSTRALIA DEPARTMENT OF MINES : MINING REVIEW for the half-year ended 30th June, 1944—No. 80. 118 pp. Adelaide : Government Printer, 1945.

SYSTEMS OF WEIGHTS AND MEASURES. By W. R. Ingalls. 51 pp. New York : Published by the American Institute of Weights and Measures, 1945. (*Presented by the Author.*)

UNITED STATES GEOLOGICAL SURVEY BULLETINS : 900–J and K : Subsurface geology and oil and gas resources of Osage County, Oklahoma—Part 10. Burbank and South Burbank oil fields, Townships 26 and 27 North, Range 5 East, and Townships 25 to 27 North, Range 6 East, by N. W. Bass, H. B. Goodrich, and W. R. Dillard ; Part 11. Summary of subsurface geology with special reference to oil and gas, by N. W. Bass. (pp. 321–42 and 343–93, with illus. 60 and 45 cents.) 924 : Supplement to catalogue of mesozoic and cenozoic plants of North America, 1919–37, compiled by R. S. LaMotte. (330 pp. 40 cents.) 928–A : Stratigraphy, structure, and mineralization in the Beaver–Tarryall area, Park County, Colo. (a reconnaissance report), by Quentin D. Singewald. (44 pp., with maps. 15 cents.) 928–B : Geology and ore deposits of the Shafter mining district, Presidio County, Tex., by Clyde P. Ross. (125 pp., with maps. 65 cents.) 928–C : Adsorbent clays, their distribution, properties, production, and uses, by P. G. Nutting. (221 pp. 20 cents.) 928–D : Manganiferous and ferruginous chert in Perry and Lewis Counties, Tennessee, by Ernest F. Burchard (with statement on concentration tests on manganese from Perry County, by H. S. Rankin). (271 pp. 25 cents.) 930–A, B, C, D : Spirit leveling in Illinois, 1896–1941—Part 1. Southern Illinois. (pp. 1–481. 65 cents.) Part 2. West-Central Illinois. (pp. 483–918. 60 cents.) Part 3. East-Central Illinois. (pp. 919–1281. 40 cents.) Part 4. Northern Illinois. (pp. 1283–1704. 45 cents.) 931–H : Tin and tungsten deposits at Silver Hill, Spokane County, Washington, by Lincoln R. Page. (pp. 177–203, with maps. 30 cents.) 931–K : The Wild Horse quicksilver district, Lander County, Nevada, by Carl H. Dane and Clyde P. Ross. (pp. 259–78, with maps. 25 cents.) 931–R : Manganese resources of the Olympic Peninsula, Washington (a preliminary report), by Charles F. Park, Jr. (pp. 435–57, with illus.. 35 cents.) 931–S : Manganese deposits in the Paymaster mining district, Imperial County, California, by Jarvis B. Hadley. (pp. 459–73, with illus. 30 cents.) 932–D : Geophysical abstracts 107, October–December, 1941, compiled by W. Ayvazoglou. (pp. 123–83. 10 cents.) 934 : Phosphate resources of Florida, by George R. Mansfield. (83 pp., with illus. $1.00.) 935–A : Chrome resources of Cuba, by T. P. Thayer. (74 pp., with illus. $1.00.) 935–B : Manganese deposits of Cuba, by Charles F. Park, Jr. (pp. 75–97. 10 cents.) 935–C : Tin deposits of the Republic of Mexico, by William F. Foshag and Karl Fries, Jr. (pp. 99–176. 40 cents.) 935–D : Tungsten deposits, Isla de Pinos, Cuba, by Lincoln R. Page and James F. McAllister. (pp. 177–246, with illus. $1.25.) 935–E : Nickel-silicate and associated nickel-cobalt-manganese-oxide deposits near Sao Jose do Tocantins, Golaz, Brazil, by William T. Pecora. (pp. 247–305, with illus. $1.00.) 935–F : Manganese deposits in part of the Sierra Maestra, Cuba, by C. F. Park and M. W. Cox. (pp. 307–55, with illus. 75 cents.) 935–G : Geology

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and manganese deposits of Guisa–Los Negros area, Oriente Province, Cuba, by W. P. Woodring and S. N. Daviess. (pp. 357–86, with illus. 45 cents.) 935–H: Manganese deposits in Costa Rica, by Ralph J. Roberts. (pp. 387–414. 15 cents.) 936–A: Muscovite in the Spruce Pine district, North Carolina, by T. L. Kesler and J. C. Olson. (pp. 1–38, with maps. 35 cents.) 936–B: Tungsten deposits of the Nightingale district, Pershing County, Nevada, by Ward C. Smith and Philip W. Guild. (pp. 39–58, with maps. 40 cents.) 936–C: Topaz deposits near the Brewer mine, Chesterfield County, South Carolina, by Carl Fries, Jr. (pp. 59–78, with maps. 30 cents.) 936–D: Chromite and quicksilver deposits of the Del Puerto area, Stanislaus County, California, by H. E. Hawkes, Jr., F. G. Wells, and D. P. Wheeler, Jr. (pp. 79–110, with map. 30 cents.) 936–E: Manganese deposits of Cedar Creek Valley, Frederick and Shenandoah Counties, Virginia, by Watson H. Monroe. (pp. 111–41, with maps. 35 cents.) 936–F: Quicksilver deposits of the Parkfield district, California, by Edgar H. Bailey. (pp. 143–69, with map. 35 cents.) 936–H: Quicksilver deposits near the Little Missouri river, Pike County, Arkansas, by David Gallagher. (pp. 189–219, with map. 55 cents.) 936–J: The tin-spodumene belt of the Carolinas (a preliminary report), by T. L. Kesler. (pp. 245–69, with illus. 50 cents.) 936–K: Tin deposits of Irish Creek, Virginia, by A. H. Koschmann, J. J. Glass, and J. S. Vhay. (pp. 271–96. 10 cents.) 936–L: The Three Kids manganese district, Clark County, Nevada, by C. B. Hunt, V. E. McKelvey, and J. H. Wiese. (pp. 297–319, with maps. 30 cents.) 936–P: Vanadium deposits of Colorado and Utah (a preliminary report), by Richard P. Fischer. (pp. 363–394. 30 cents.) 936–Q: The Coso quicksilver district, Inyo County, California, by Clyde P. Ross and Robert G. Yates. (pp. 395–416, with map. 40 cents.) 936–R: Manganese deposits in the Artillery mountains region, Mohave County, Arizona, by S. G. Lasky and B. N. Webber. (pp. 417–48, with maps. $1.25.) 937: Bibliography of North American geology, 1929–39, compiled by Emma Mertins Thom—Part 1. Bibliography. (1063 pp. $2.50.) Part 2. Index. (pp. 1065–1546. 50 cents.) 938: Bibliography of North American geology, 1940 and 1941, compiled by Emma Mertins Thom. (479 pp. 50 cents.) 939–A, B, C, D: Geophysical abstracts, compiled by W. Ayvazoglou. 108, January–March, 1942; 109, April–June, 1942; 110, July–September, 1942; 111, October–December, 1942. (138 pp. 10 cents each.) 940–A: The Rose Creek tungsten mine, Pershing County, Nevada, by Ralph J. Roberts. (pp. 1–14, with maps. 30 cents.) 940–B: Manganese deposits of the Elkton area, Virginia, by Philip B. King. (pp. 15–55, with maps. 75 cents.) 940–C: Geophysical surveys in the Ochoco quicksilver district, Oregon (a preliminary report), by E. L. Stephenson. (pp. 57–98, with maps. 50 cents.) 940–D: Vanadium-bearing magnetite-ilmenite deposits near Lake Sanford, Essex County, New York, by James R. Balsley, Jr. (pp. 99–123, with illus. 60 cents.) 940–E: Occurrence of manganese in Eastern Aroostook County, Maine, by Walter S. White. (pp. 125–61, 25 cents.) 940–F: Manganese deposits of the Lyndhurst–Vesuvius district, Augusta and Rockbridge Counties, Virginia, by Maxwell M. Knechtel. (pp. 163–98, with maps.) 940–G: Manganese deposits of the Sweet Springs district, West Virginia and Virginia, by Harry S. Ladd. (pp. 199–217, with maps. 35 cents.) 940–H: Manganese deposits of the Flat Top and Round Mountain districts, Bland and Giles Counties, Virginia, by Harry S. Ladd and Frank W. Stead. (pp. 219–45, with illus. 75 cents.) 940–I: Tungsten deposits in the Boriana district and the Aquarius Range, Mohave County, Arizona, by S. W. Hobbs. (pp. 247–64, with illus. 40 cents.) 940–J: Cobalt-bearing manganese

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UNITED STATES GEOLOGICAL SURVEY PROFESSIONAL PAPERS : 197–D : The Basin and Range Province in Utah, Nevada, and California, by Thomas B. Nolan. (pp. 141–96. 15 cents.) 197–E : Some standard thermal dehydration curves of minerals, by P. G. Nutting. (pp. 197–217. 5 cents.) 197–F : The action of some aqueous solutions on clays of the Montmorillonite group, by P. G. Nutting. (pp. 219–35. 10 cents.) 200 : Geology and ore deposits of the Magdalena mining district, New Mexico, by G. F. Loughlin and A. H. Koschmann. 168 pp. $2.00.) 201 : Geology and ore deposits of the Cottonwood-American Fork Area, Utah, by F. C. Calkins and B. S. Butler, with sections on history and production, by V. C. Heikes. (152 pp. and separate map carton. $1.75.) 202 : Geology and ore deposits of the Metaline quadrangle, Washington, by C. S. Park, Jr. and R. S. Cannon, Jr. (81 pp. $1.50.) 203 : Stratigraphy and fauna of the Louisiana limestone of Missouri, by James Steele Williams. (133 pp. 50 cents.) 204 : Geology of the Hanover-York district, Pennsylvania, by Anna Jonas Stose and George W. Stose. (pp. 84. $1.75.) 205-A : Relative abundance of nickel in the earth's crust, by Roger Clark Wells. (pp. 1–21. 10 cents.) Washington, D.C. : U.S. Government Printer. 1942–1944.

UNITS OF WEIGHTS AND MEASURES. By W. R. Ingalls. 49 pp. New York : Published by the American Institute of Weights and Measures, 1946. (*Presented by the Author.*)

WESTERN AUSTRALIA DEPARTMENT OF MINES : Report of the Department of Mines for the year 1944. 211 pp. Perth : Government Printer, 1945.

WESTERN AUSTRALIA DEPARTMENT OF MINES : Report of the Geological Survey for the years 1941 and 1942 ; Report of the Government Geologist for the year 1943. 13 and 31 pp. respectively. Perth : Government Printer, 1944.

INDEX OF RECENT ARTICLES.

NOTE.—All Articles indexed are contained in the Library of the Institution.

INDEX OF RECENT ARTICLES—*continued.*

COAL—*continued.*

COKING PRACTICE—AMERICAN COALS.—Work of the survey of carbonizing properties of American coals. J. D. Davis and D. A. Reynolds.—U.S. Bureau of Mines *Report of Investigations* 3760, June, 1944, 17 pp.

COKING PRACTICE — ANTHRACITE.—Utilization of anthracite fines in the manufacture of by-product coke. William Seymour and L. D. Schmidt.—U.S. Bureau of Mines *Report of Investigations* 3808, April, 1945, 33 pp.

DISTILLATION — BIBLIOGRAPHY. — Bibliography of Bureau of Mines investigations on the production of liquid fuels from oil shale, coal, lignite, and natural gas. Arno C. Fieldner and Paul L. Fisher.—U.S. Bureau of Mines *Information Circular* 7304, January, 1945, 18 pp.

DUST.—Dust in conveyor districts—VIII: how special types of loading chutes affect the raising of dust at loading points. IX: the dust problem at coal transfer points. W. A. Terrell.—*Colliery Engng.*, Lond., Vol. 23, 1946; January, pp. 15–17; February, pp. 38–41. 1s. each.

DUST PREVENTION.—The prevention and suppression of coal-dust in mines. H. Watson Smith and H. F. Banks.—*Trans.* Instn. Min. Engrs., Lond., Vol. 104, Part 11, August, 1945, pp. 602–21. 6s.

DUST—SUPPRESSION.—A new device for allaying dust made in boring operations, with special reference to rippings. J. C. Webb.—B.C.O.R.A. *Quart. Bull.*, Lond., No. 6, June, 1945, pp. 14–16.

DUST SUPPRESSION.—Dust in return roadways: a review of causes and available means for its suppression. (*Concluded.*) W. A. Terrell.—*Iron Coal Tr. Rev.*, Lond., Vol. 151, December 14, 1945, pp. 935–8. 9d.

DUST — SUPPRESSION.—The Daniel dust trap. J. Ivon Graham.—B.C.O.R.A. *Quart. Bull.*, Lond., No. 6, June, 1945, pp. 11–14.

ECONOMICS.—Coal in 1945. Sir Richard Redmayne.—*Engineer*, Lond., Vol. 181, 1946; January 4, pp. 4–5; January 11, pp. 29–30. 1s. each.

ECONOMICS.—Plans for the coal mining industry.—*Engineer*, Lond., Vol. 181, January 18, 1946, pp. 57–9. 1s.

ECONOMICS.—The coal trade in 1945.—*Colliery Guard.*, Lond., Vol. 172, January 4, 1946, pp. 1–8. 10d.

ECONOMICS—GREAT BRITAIN.—British coal-mining industry in 1945.—*Iron Coal Tr. Rev.*, Lond., Vol. 152, January 4, 1946, pp. 1–5. 9d.

ECONOMICS—INDIA.—Coal mining in India: organisation and conditions of service. (From notes prepared by the Council of the Association of Colliery Managers of India.)—*Iron Coal Tr. Rev.*, Lond., Vol. 152, January 18, 1946, pp. 103–4. 9d.

COAL—*continued.*

EUROPE.—Mining in Europe: wartime coal production and utilization. H. F. Yancy.—*Coal Age*, Albany, N.Y., Vol. 50, December, 1945, pp. 115–8. 35 cents.

FIFESHIRE.—Colliery modernization with special reference to The Fife Coal Company's collieries. Henry R. King.—*Trans.* Instn. Min. Engrs., Lond., Vol. 104, Part 11, August, 1945, pp. 634–59. 6s.

GEOLOGY—ANTHRACITE FORMATION.—The formation of anthracite. E. A. Rudge and S. E. A. Moon.—*Chem. Ind.*, Lond., January 26, 1946, pp. 40–41. 1s. 9d.

HAULAGE.—Lancashire man-riding haulage practice. A. Wright (Investigator to the Lancashire & Cheshire Safety in Mines Research Committee).—*Trans.* Instn. Min. Engrs., Lond., Vol. 104, Part 10, July, 1945, pp. 566–88. 6s.

HAULAGE—BELT CONVEYORS.—Underground belt haulage for panel and main entries. (Report by Committee on Conveyor Mining.) A. E. Long.—*Min. Congr. J.*, Wash., D.C., Vol. 31, October, 1945, pp. 47–9. 20 cents.

HAULAGE — CONVEYORS — CHAIN. — Chain-conveying in steep seams. R. W. Scurfield.—*Trans.* Instn. Min. Engrs., Lond., Vol. 104, Part 10, July, 1945, pp. 547–53. 6s.

HAULAGE — ELECTRIC — HAZARDS. — Hazards of the trolley-locomotive haulage system in coal mines. D. Harrington and R. G. Warncke.—U.S. Bureau of Mines *Information Circular* 7328, July, 1945, 38 pp.

HAULAGE—SAFETY MEASURES.—Haulage safety promoted by logic in eliminating hazards.—*Coal Age*, Albany, N.Y., Vol. 50, December, 1945, pp. 110–14. 35 cents.

LANCASHIRE—Clockface colliery (near Warrington, Lancs.).—*Engineer*, Lond., Vol. 180, 1945; December 21, pp. 500–501; December 28, pp. 521–3. 1s. each.

LANCASHIRE.—The Lancashire coalfield. N. Simpkin.—*Inst. Fuel War Time Bull.*, December, 1945, pp. 54–62.

LIGNITE—NORTH DAKOTA.—The Velva mine (lignite coal mine in North Dakota).—*Min. Congr. J.*, Wash., D.C., Vol. 31, October, 1945, pp. 30–34. 20 cents.

MINING METHODS—BORD AND PILLAR.—Bord and pillar mining. Parts 1 & 2 (*To be continued.*) A. Roberts.—*Colliery Engng.*, Lond., Vol. 23, 1946; January, pp. 9–10; February, pp. 29–30 and 32. 1s. each.

MINING METHODS — LANARKSHIRE.—Sixty years in mining: early methods of coal-getting, and development of machine mining in Lanarkshire. James Dalgleish. (Paper read before Scottish Branch of National Association of Colliery Managers.)—*Iron Coal Tr. Rev.*, Lond., Vol. 152, January 18, 1946, pp. 153–6. 9d.

INDEX OF RECENT ARTICLES—*continued.*

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MINING METHODS — LONGWALL. — Modified longwall cuts top trouble and boosts efficiency. J. H. Edwards.—*Coal Age*, Albany, N.Y., Vol. 50, December, 1945, pp. 107–9. 35 cents.

MINING METHODS — MECHANICAL — SHUTTLE CARS.—Tonnage raised by adoption of shuttle-car gathering.—*Coal Age*, Albany, N.Y., Vol. 50, November, 1945, pp. 88–9. 35 cents.

NEW BRUNSWICK.—The Grand Lake coal field, N.B. W. J. Wright and C. S. Clements.—*Trans.* Canad. Inst. Min. Metall., Vol. 48, 1945, pp. 747–54. (*Canad. Inst. Min. Metall. Bull.*, Montreal, No. 404, December, 1945. $1.00.)

OPENCAST MINING.—Four veins stripped in 25-ft. benches to get 2,200 tons daily. Ralph R. Richart.—*Coal Age*, Albany, N.Y., Vol. 50, December, 1945, pp. 96–100. 35 cents.

OPENCAST MINING.—Latest practices in opencut coal mining: overburden stripped to 70 ft. exposes 20-ft. seam; six-cubic-yard dragline; worked-out cuts filled. — *Chem. Engng. Min. Rev.*, Melb., Vol. 38, October, 1945, pp. 19–20. 1s.

OPENCAST MINING.—Mount outcrop strip mining at Leckie Smokeless Coal Company's Anjean mine. E. W. Haessler.—*Min. Congr. J.*, Wash., D.C., Vol. 31, November, 1945, pp. 39–43. 20 cents.

OPENCAST MINING.—The development of strip mining. R. H. Sherwood.—*Min. Congr. J.*, Wash., D.C., Vol. 31, November, 1945, pp. 31–4. 20 cents.

POLAND.—Can Polish coal save Europe?—*Min. J.*, Lond., Vol. 226, February 9, 1946, pp. 105–7. 8d.

PREPARATION—PACKAGED FUEL.—Technical and economic study of packaged fuel. (The compression and wrapping of coal slack.) V. E. Parry—U.S. Bureau of Mines *Report of Investigations* 3757, June, 1944, 45 pp.

PREPARATION — PLANT MAINTENANCE.—Getting results in maintaining coal-preparation plants. Fred W. Richart.—*Coal Age*, Albany, N.Y., Vol. 50, November, 1945, pp. 90–96. 35 cents.

PREPARATION — PLANT MAINTENANÇE.—Maintenance for more efficient coal preparation. Fred W. Richart.— *Coal Age*, Albany, N.Y., Vol. 50, December, 1945, pp. 101–6. 35 cents.

PREPARATION — SINK-AND-FLOAT. — Low-gravity float-and-sink separations of bituminous-coal samples. Thomas Fraser, William L. Crentz, H. M. Cooper, R. F. Abernethy, and O. T. Barrett.—U.S. Bureau of Mines *Report of Investigations* 3812, May, 1945. 11 pp.

COAL—*continued.*

PREPARATION — WASHING—CYCLONE WASHER.—The use of centrifugal force for cleaning fine coal in heavy liquids and suspensions with special reference to the cyclone washer. M. G. Driessen.—*J. Inst. Fuel*, Lond., Vol. 19, December, 1945, pp. 33–45. 10s.

PULVERISED COAL — NEVADA — McGILL.—The coal pulverizing plant at the McGill smelter of the Kennecott Copper Corporation. Edward Pesout.—(A.I.M.E. Tech. Pub. No. 1822) *Min. Tech.*, N.Y., Vol. 9, November, 1945, 6 pp. $1.35.

QUEENSLAND.—Mount Mulligan coal mine, N.Q.: Geological Survey report. S. R. L. Shepherd.—*Qd. Govt. Min. J.*, Brisbane, Vol. 46, September, 1945, pp. 261–5. 6d.

RESEARCH — GERMANY. — Co-operative coal research in the Ruhr: a review of the organisation and activities of the main coal research bodies in Western Germany with particular reference to working developments. —*Colliery Engng.*, Lond., Vol. 23, January, 1946, pp. 12–14 and 17. 1s.

RESEARCH—U.S.A.—Annual report of research and technologic work on coal, fiscal year 1944. A. C. Fieldner, P. L. Fisher, and R. E. Brewer.—U.S. Bureau of Mines *Information Circular* 7322, May, 1945, 79 pp.

SCOTLAND.—The growth of the mining industry in Scotland (Presidential Address). Thomas Brown.—*Trans.* Instn. Min. Engrs., Lond., Vol. 104, Part 10, July, 1945, pp. 593–601. 6s.

SOUTH AFRICA.—South African coal resources: Fuel Research Board survey.—*S. Afr. Min. Engng. J.*, J'burg, Vol. 56, Part II, December 15, 1945, pp. 379–81. 6d.

SUPPORT—CONCRETE.—Trials of reinforced concrete arches on roadways: investigations carried out by Safety in Mines Research Board, Lancashire & Cheshire Safety in Mines Research Committee. W. H. Evans and A. Wright.—*Colliery Guard.*, Lond., Vol. 172, January 11, 1946, pp. 39–44. 10d. *Iron Coal Tr. Rev.*, Lond., Vol. 152, January 25, pp. 141–6. 9d. *Colliery Engng.*, Lond., Vol. 23, February, 1946, pp. 43–8. 1s.

SUPPORT — ROADWAYS — LONGWALL. —Report of the Safety in Mines Research Committee of the North Staffordshire Institute of Mining Engineers: the application of double packing on longwall roadways.—*Trans.* Instn. Min. Engrs., Lond., Vol. 104, Part 10, July, 1945, pp. 512–36. 6s.

SUPPORT—ROOF CONTROL.—Roof control research and practical application. A. Winstanley.—*Trans.* Instn. Min. Engrs., Lond., Vol. 104, Part 9, June, 1945, pp. 430–41. 6s.

TRANSVAAL — WITBANK. — The production of both coking and steam coals from the Witbank coalfield. L. A. Bushell, S. B. Gass, F. W. Quass, and R. F. J. Teichmann.—*J. Chem. Metal. Min. Soc. S. Afr.*, J'burg, Vol. 46, July-August, 1945, pp. 45–59. 3s. 6d.

INDEX OF RECENT ARTICLES—*continued*.

COAL—*continued*.

U.S.S.R.—Coal mining in Russia: opening up new Soviet coal fields. A. Beloborodov.—*S. Afr. Min. Engng. J.*, J'burg, Vol. 56, Part II, January 12, 1946, p. 495. 6d.

U.S.S.R.—DONBAS.—A coalfield restoration problem. George H. Hanna. (Restoration and extension of the Donbas.)—*Canad. Min. J.*, Gardenvale, P.Q., Vol. 66, November, 1945, pp. 784–5. 25 cents.

VENTILATION — DUST — AIR-SAMPLING.—Filter-paper method for obtaining dust-concentration results comparable to impinger results. Carlton E. Brown,—U.S. Bureau of Mines *Report of Investigations* 3788, December, 1944, 20 pp.

YORKSHIRE.—Bullcliffe Wood colliery (Bretton, W. Yorks.): a pioneer development. —*Iron Coal Tr. Rev.*, Lond., Vol. 152, January 11, 1946, pp. 61–2. 9d.

COPPER.

ARIZONA.—Milling and smelting operations of the Magma Copper Co., Superior, Ariz. Edward J. Caldwell.—U.S. Bureau of Mines *Information Circular* 7300, December, 1944, 38 pp.

QUEENSLAND — CLERMONT. — Peak Downs copper mine, Copperfield, Clermont, Queensland: Geological Survey report. A. K. Denmead.—*Qd. Govt. Min. J.*, Brisbane, Vol. 46, September, 1945, pp. 259-60. 6d.

QUEENSLAND — MOUNT CANNINDAH. —Mount Cannindah copper: Geological Survey report. A. K. Denmead.—*Qd. Govt. Min. J.*, Brisbane, Vol. 46, October, 1945, p. 302. 6d.

ECONOMICS.

EDUCATION.—Second report of Committee (of Institution of Mining Engineers) on Qualifications and Training for Mining Engineers.—*Trans.* Instn. Min. Engrs., Lond., Vol. 104, Part 8, May, 1945, pp. 372–84. 6s.

EDUCATION — COAL MINING. — The South Notts occupational training scheme. Tudor J. Jones—*Colliery Guard.*, Lond., Vol. 172, January 25, 1946, pp. 107–110. 10d.

HEALTH OF MINERS—PHTHISIS PREVENTION.—Control of silicosis hazard by substitution of quartz-free or low-quartz material for sand under mine locomotives. Carlton E. Brown and H. H. Schrenk.—U.S. Bureau of Mines *Information Circular* 7297, October, 1944, 5 pp.

HEALTH OF MINERS—SILICOSIS.—Miners' silicosis: reports on recent research work in certain other countries. Waldemar Schweisheimer.—*Colliery Engng.*, Lond., Vol. 23, February, 1946, p. 42. 1s.

ECONOMICS—*continued*.

HEALTH OF MINERS—TROPICAL HYGIENE.—Tropical hygiene and the overseas Empire. G. Macdonald.—*J. R. Soc. Arts*, Lond., Vol. 94, December 7, 1945, pp. 35–40. 2s.

LABOUR — WESTERN AUSTRALIA. — Labour in Western Australian mines: manpower requirements of mines; types of labour; methods of training labour on leading mines.—*Chem. Engng. Min. Rev.*, Melb., Vol. 38, October, 1945, pp. 21–22. 1s.

MANAGEMENT.—The education of an engineer as an administrator. W. E. Clegg.—*J. Instn. Engrs. Aust.*, Sydney, Vol. 17, September, 1945, pp. 173–80.

MANAGEMENT.—Management. John Wishart.—*Mining, Electrical and Mechanical Engineer*, Manchester, Vol. 26, January, 1946, pp. 259–70. 2s. 6d.

MANAGEMENT—COLLIERIES. — Administration of collieries, Parts 4 & 5: the checking system; and co-ordinating administrative control. (*To be continued.*) F. Colin Swallow. *Colliery Engng.*, Lond., Vol. 23, 1946; January, pp. 4–8; February, pp. 33–7. 1s. each.

PRODUCTION — CALIFORNIA. — Wartime mineral industry of California. Spangler Ricker.—*Calif. J. Min.*, Sacramento, Vol. 41, January, 1945, pp. 37–44.

PRODUCTION—ONTARIO.—Statistical review of the mineral industry of Ontario for 1940. Maurice Tremblay.—Annual Report of Ontario Dept. of Mines, Vol. 50, Part IA, 1941, Toronto, 1945, 72 pp.

PROFESSIONAL MATTERS — METALLURGISTS.—Metallurgy as a profession: past, present, and future. (Presidential Address.) J. W. Jenkin.—*J. Bgham. Metall. Soc.*, Bgham., Vol. 25, December, 1945, pp. 220–37. 2s. 6d.

STATE IN RELATION TO MINING — U.S.A.—Principles of foreign mineral policy of the United States, as interpreted by C. K. Leith, after conferences with mineral specialists in industry and Government, November, 1945.—*Min. Metall.*, N.Y., Vol. 27, January, 1946, pp. 6–17. 50 cents.

TAXATION — MEXICO. — Mexican mining must have tax relief. Constantino Perez Duarte.—(Free translation from article appearing in *El Economista*, Mexico, D. F.)—*Engng. Min. J.*, N.Y., Vol. 146, December, 1945, pp. 96–9. 50 cents.

TAXATION — SOUTH AFRICA.—Taxation and the mining industry: analysis of the present position. Alan Stuart.—*S. Afr. Min. Engng. J.*, J'burg, Vol. 56, Part II, December 1, 1945, pp. 319–21. 6d.

TAXATION—SOUTH AFRICA.—The State and gold mining: the case for taxation revision. Parts I, II, III and IV (*To be concluded*).—*S. Afr. Min. Engng. J.*, J'burg, Vol. 56, Part II; December 29, 1945, pp. 437–41; January 5, 1946, pp. 463–9; January 12, 1946, pp. 489–93; January 19, 1946, pp. 515–7. 6d. each.

INDEX OF RECENT ARTICLES—*continued.*

GEOLOGY AND ORE DEPOSITS.

BRITISH COLUMBIA—VANCOUVER.—The coast range batholith near Vancouver, British Columbia. T. C. Phemister.—*Quart. J. Geol. Soc.*, Lond., Vol. 101, Parts 1 & 2, 1945, pp. 37–81. 7s. 6d.

DERBYSHIRE—EDALE VALLEY.—The carboniferous rocks of the Edale anticline, Derbyshire. R. G. S. Hudson and G. Cotton. *Quart. J. Geol. Soc.*, Lond., Vol. 101, Parts 1 & 2, 1945, pp. 1–36. 7s. 6d.

ENGLAND.—Evidence for a new major fault in north-east England. A. Fowler.—*Geol. Mag.*, Hertford, Vol. 82, No. 6, November—December, 1945, pp. 245–55. 5s.

GENERAL.—Economic geology (Presidential Address.) John M. Boutwell.—*Econ. Geol.*, Lancaster, Pa., Vol. 40, No. 7, November, 1945, pp. 431–48. $1.00.

MONTANA.—Stratigraphic control of ore-bodies in Montana. Eugene S. Perry.—*Engng. Min. J.*, N.Y., Vol. 146, October, 1945, pp. 96–8. 50 cents.

NEVADA.—The mineralization of Nevada. W. James Bichan.—*Canad. Min. J.*, Gardenvale, P. Q., Vol. 66, December, 1945, pp. 855–8. 25 cents.

PEMBROKESHIRE.—The geology of the Prescelly Hills, North Pembrokeshire. W. D. Evans.—*Quart. J. Geol. Soc.*, Lond., Vol. 101, Parts 1 & 2, 1945, pp. 89–107. 7s. 6d.

QUEENSLAND—DAWSON RIVER.—The Dawson River area : Geological Survey report. J. H. Reid. —*Qd. Govt. Min. J.*, Brisbane, Vol. 46, October, 1945, pp. 296–9. 6d.

QUEENSLAND — GARNANT. — Geology, Garnant, near Rockhampton : Geological Survey report. J. D. East.—*Qd. Govt. Min. J.*, Brisbane, Vol. 46, October, 1945, pp. 300–301. 6d.

SPAIN—HUELVA.—The San Telmo ore-body, Spain. J. C. Allan.—*Instn. Min. Metall. Bull.*, Lond., No. 476, January, 1946, 15 pp. 1s. 6d.

GOLD.

ARABIA.—The gold fields of Arabia : possible source of 'King Solomon's Mines.' —*S. Afr. Min. Engng. J.*, J'burg, Vol. 56, Part II, December 29, 1945, pp. 453–4. 6d.

BRITISH COLUMBIA.—The Salmon gold mine (B.C.). Evan E. Harris and C. Rutherford.—*W. Miner*, Vancouver, B.C., Vol. 19, January, 1946, pp. 38–41. 25 cents.

CONCENTRATION—JIGGING.—Gold recovery by jig : advantages for small workings. L. L. Colin.—*S. Afr. Min. Engng. J.*, J'burg, Vol. 56, Part II, November 10, 1945, pp. 231–3. 6d.

GOLD—*continued.*

GEOLOGY — BRITISH COLUMBIA.—Structure at Island Mountain mine, Wells, B.C. B. C. Benedict.—*Trans. Canad. Inst. Min. Metall.*, Vol. 48, 1945, pp. 755–70. (*Canad. Inst. Min. Metall. Bull.*, Montreal, No. 404, December, 1945. $1.00.)

NEW GUINEA—MOROBE.—The fineness of gold, with special reference to the Morobe goldfield, New Guinea. N. H. Fisher.—*Econ. Geol.*, Lancaster, Pa., Vol. 40, No. 7, November, 1945, pp. 449-95. $1.00.

ONTARIO—THUNDER BAY—LITTLE LONG LAC.—Report on gold belts in the Little Long Lac-Sturgeon River district. A. E. Tyson.—*Canad. Min. J.*, Gardenvale, P.Q., Vol. 66, December, 1945, pp. 839–50. 25 cents.

PHILIPPINE ISLANDS.—Gold mines of the Philippines : early resumption of production. —*S. Afr. Min. Engng. J.*, J'burg, Vol. 56, Part II, December 8, 1945, pp. 355–7. 6d.

IRON AND STEEL.

ALABAMA — BIRMINGHAM. — Classification and tabling of Birmingham (Ala.) red iron ores with recommendations for added recovery. Will H. Coghill, G. Dale Coe, and Homer N. McDonald.—U.S. Bureau of Mines *Report of Investigations* 3789, January, 1945, 18 pp.

CONCENTRATION — FLOTATION. — Amine flotation of gangue from magnetite concentrates. Donald W. Scott, A. C. Richardson, and Nathaniel Arbiter.—(A.I.M.E. Tech. Pub. No. 1902) *Min. Tech.*, N.Y., Vol. 9, November, 1945, 10 pp. $1.35.

FLOTATION.—Beneficiation of iron ores by flotation—Part 1. Anionic flotation of silica from calcareous red iron ores of the Birmingham district, Alabama. J. Bruce Clemmer, B. H. Clemmons, Carl Rampacek, M. F. Williams, Jr. and R. H. Stacy.—U.S. Bureau of Mines *Report of Investigations* 3799, March, 1945, 42 pp.

MINNESOTA—MESABI RANGE.—Taconite mining, past and future. R. W. Whitney. — *Excavat. Engr.*, S. Milwaukee, Vol. 40, January, 1946, pp. 14-17. 25 cents.

PRODUCTION—SPONGE IRON.—Sponge iron : a progress report. R. S. Dean.—U.S. Bureau of Mines *Report of Investigations* 3790, January, 1945, 29 pp.

LEAD AND ZINC.

CONCENTRATION. — Flexible flowsheet treats varying lead-zinc ore. C. E. Craven.—*Engng. Min. J.*, N.Y., Vol. 146, November, 1945, pp. 85–9. 50 cents.

CONCENTRATION — FLOTATION. — Amine flotation of sphalerite-galena ores. Herbert H. Kellogg and Hugo Vasquez-Rosas. —(A.I.M.E. Tech. Pub. No. 1906) *Min. Tech.*, N.Y., Vol. 9, November, 1945, 23 pp. $1.35.

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INDEX OF RECENT ARTICLES—*continued.*

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ORE DRESSING — TESTS. — Ore-testing studies on gold and gold-silver deposits. J. A. Woolf and A. P. Towne.—U.S. Bureau of Mines *Report of Investigations* 3765, August, 1944. 63 pp.

PLANT — RUBBER-LINED PIPES. — Rubber-lined pipe resists abrasion in mill circuits. James Casale.—*Engng. Min. J.*, N.Y., Vol. 146, December, 1945, pp. 100–101. 50 cents.

SCREENS.—A vibrating screen surface for the removal of flat and elongated pieces from crushed stone. Lloyd H. Banning and Frank D. Lamb.—U.S. Bureau of Mines *Report of Investigations* 3781, September, 1944, 5 pp.

SCREENS —STANDARD.— Comparison of fine-series, square-mesh-wire test sieves of different countries. R. E. Brewer.—U.S. Bureau of Mines *Report of Investigations* 3766, July, 1944, 5 pp.

MINING (General).

ACCIDENT PREVENTION—OLD WORKINGS.—Precautions to be taken when approaching old mine workings. D. Harrington and R. G. Warncke.—U.S. Bureau of Mines *Information Circular* 7288, June, 1944, 20 pp.

ACCIDENTS—IRON ORE MINES—MICHIGAN (LAKE SUPERIOR).—Accident experience, iron-ore mines—Lake Superior district, 1940–43. Frank E. Cash and Reuben D. Larsen.—U.S. Bureau of Mines *Information Circular* 7321, May, 1945, 9 pp.

BLASTING—BORE HOLES—DIAMOND DRILLING.—Diamond drilling of blast holes, Lake Superior district iron-ore mines. Ernest W. Johnson and Frank E. Cash.—U.S. Bureau of Mines *Information Circular* 7317, April, 1945, 11 pp.

BLASTING—COYOTE.—Coyote blast at Mount Morgan (Queensland). Arthur L. Lloyd.—*Chem. Engng. Min. Rev.*, Melb., Vol. 38, October, 1945, pp. 13–18. 1s.

BLASTING—DIAMOND DRILL-HOLES.—Diamond drilling as applied to ore production —symposium : summary of practice of 40 operators, and detailed account of electric core drilling in Soudan mine of Oliver Iron Mining Co. (*To be continued.*)—*Engng. Min. J.*, N.Y., Vol. 146, October, 1945, pp. 69–80. 50 cents.

BLASTING — EXPLOSIVES — BLACK POWDER.—Studies of the effect of humidity on the sensitivity and dispersion of black powder. Raymond H. Moore, with Wilbert J. Huff.—U.S. Bureau of Mines *Report of Investigations* 3782, November, 1944, 18 pp.

BLASTING — EXPLOSIVES — SAFETY MEASURES.—Safety in the handling and use of explosives at Hollinger mine (Ont.). Geo. Gedge.—*Trans.* Canad. Inst. Min. Metall., Vol. 48, 1945, pp. 771–88. (*Canad. Inst. Min. Metall. Bull.*, Montreal, No. 404, December, 1945. $1.00.)

MINING (General)—*continued.*

BLASTING—EXPLOSIVES—STORAGE.—Surface storage of explosives. D. Harrington and R. G. Warncke.—U.S. Bureau of Mines *Information Circular* 7307, February, 1945, 16 pp.

BLASTING—LONG-HOLE DRILLING.—Long-hole Leyner drilling speeds stope preparation.—*Engng. Min. J.*, N.Y., Vol. 146, November, 1945, pp. 82–4. 50 cents.

BORING—CORE DRILLS—DIAMOND.—Deviation of diamond-drill holes in the Metaline District, Washington. Edward Sampson and Allan B. Griggs.—(A.I.M.E. Tech. Pub. No. 1825) *Min. Tech.*, N.Y., Vol. 10, January, 1946, 8 pp. $1.35.

BORING—CORE DRILLS—DIAMOND.—Diamond drilling in exploration and development. R. S. Moehlman.—(A.I.M.E. Tech. Pub. No. 1858) *Min. Tech.*, N.Y., Vol. 10, January, 1946, 20 pp. $1.35.

BORING—CORE DRILLS—DIAMOND.—Some details of diamond-drilling equipment. S. V. Griffith.—*Min. Mag.*, Lond., Vol. 73, December, 1945, pp. 343–7. 1s.

BORING—CORE DRILLS—DIAMOND.—Some problems involved in the interpretation of diamond-drill-hole sampling and surveying. John J. Collins.—(A.I.M.E. Tech. Pub. No. 1842) *Min. Tech.*, N.Y., Vol. 10, January, 1946, 28 pp. $1.35.

BORING — BOREHOLE SAMPLING. — Diamond-drill sludge sampling and appraisal of a weathered ilmenite ore body, Piney River, Virginia. D. M. Davidson.—(A.I.M.E. Tech. Pub. No. 1951) *Min. Tech.*, N.Y., Vol. 10, January, 1946, 5 pp. $1.35.

BORING — BOREHOLE SURVEYING. — Surveying and controlling diamond-drill holes. Arthur B. Yates.—(A.I.M.E. Tech. Pub. No. 1950) *Min. Tech.*, N.Y., Vol. 10, January, 1946, 10 pp. $1.35.

BORING — BOREHOLE SURVEYING. — The application of oil-well surveying instruments and technical services in the mining industry. G. L. Kothny.—(A.I.M.E. Tech. Pub. No. 1964) *Min. Tech.*, N.Y., Vol. 10, January, 1946, 8 pp. $1.35.

BORING AND BOREHOLES—SURVEYING.—A method for surveying drill holes by oriented drill rods. L. A. Dahners and C. J. Cohen.—U.S. Bureau of Mines *Report of Investigations* 3773, August, 1944. 6 pp.

BRITISH COLUMBIA—HISTORY.—Vulcan in British Columbia : the life story of a mining industry. (*Concluded.*) H. G. Nichols. —*Canad. Min. J.*, Gardenvale, P.Q., Vol. 66, 1945 ; November, pp. 786–9 ; December, pp. 868–72. 25 cents each.

DEEP MINES.—Mining at depth or at great heights. W. Schweisheimer.—*Min. Mag.*, Lond., Vol. 74, January, 1946, pp. 27–8. 1s.

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MINING (General)—*continued.*

DREDGING — DRAGLINE DREDGES.—Dragline gold dredging. Glenville A. Collins.—*Trans.* Canad. Inst. Min. Metall., Vol. 48, 1945, pp. 717–25. (*Canad. Inst. Min. Metall. Bull.*, Montreal, No. 403, November, 1945. $1.00.)

DREDGING — VICTORIA. — Operations of Victoria Gold Dredging Co., N.L. (Newstead, Vic.).—*Chem. Engng. Min. Rev.*, Melb., Vol. 38, November, 1945, pp. 39–40. 1s.

FRENCH EQUATORIAL AFRICA.—Mining position in French Equatorial Africa.—*Min. Indust. Mag.*, J'burg, Vol. 33, December, 1945, p. 755. 1s.

GEOPHYSICAL PROSPECTING.—Trends in exploration of mineral deposits. Lowell B. Moon.—U.S. Bureau of Mines *Information Circular* 7312, March, 1946, 6 pp.

GEOPHYSICAL PROSPECTING — BIBLIOGRAPHY.—Geophysical abstracts 120, January-March, 1945. Compiled by W. Ayvazoglou and V. Skitsky.—U.S. Bureau of Mines *Information Circular* 7324, June, 1945, 43 pp.

GEOPHYSICAL PROSPECTING — BIBLIOGRAPHY—U.S.S.R. — Bibliography of Russian literature on geophysical exploration, 1929–41. Compiled by W. Ayvazoglou and V. Skitsky.—U.S. Bureau of Mines *Information Circular* 7323, June, 1945, 44 pp.

GEOPHYSICAL PROSPECTING — MAGNETOMETRIC METHODS.—The magnetometer as an aid in geological mapping. J. W. Ambrose.—Canada Department of Mines and Resources, Mines and Geology Branch *Geol. Surv. Bull.* No. 2, 1945, 21 pp. 10 cents.

GEOPHYSICAL PROSPECTING — MAGNETOMETRIC METHODS — ALASKA. — Magnetic exploration of the nickel-copper deposits of Bohemia Basin, South-eastern Alaska. Matt. S. Walton, Jr., and George C. Kennedy.—*Econ. Geol.*, Lancaster, Pa., Vol. 40, No. 7, November, 1945, pp. 496–502. $1.00.

GEOPHYSICAL PROSPECTING — MAGNETOMETRIC METHODS—FLORIDA.—Magnetic survey of the Florida peninsula. F. W. Lee, J. H. Swartz, and S. J. Hemberger.—U.S. Bureau of Mines *Report of Investigations* 3810, May, 1945, 49 pp.

GEOPHYSICAL PROSPECTING — MAGNETOMETRIC METHODS — OREGON. — Magnetometer surveys on black sands of the Oregon coast. E. L. Stephenson.—U.S. Bureau of Mines *Report of Investigations* 3814, June, 1945, 18 pp.

HAULAGE — DIESEL LOCOMOTIVE. — Underground haulage by diesel locomotives. C. P. Bates. (Abstract of paper read before N. Staffs Institute of Mining Engineers.)—*Colliery Guard.*, Lond., Vol. 171, 1945, December 21, pp. 757–8; December 28, pp. 785–6; Vol. 172, January 1, 1946, pp. 33–4. 10d. each.

MINING (General)—*continued.*

HAULAGE — ELECTRIC — FIRE PREVENTION.—Prevention of fires caused by electric arcs and sparks from trolley wires. F. E. Griffith, E. J. Gleim, R. T. Arz, and D. Harrington.—U.S. Bureau of Mines *Information Circular* 7302, November, 1944, 10 pp.

HOISTING—CONTROL.—Control of mine winders. Rolt Hammond.—*Mine & Quarry Engng.*, Lond., Vol. 11, January, 1946, pp. 3–10. 1s.

HOISTING — SAFETY DEVICES. — Testing safety catches on mine cages—Part II: specially designed tripping devices and permanently installed equipment featured at some mines. Clyde M. Pearce and Frank E. Cash.—*Min. Congr. J.*, Wash., D.C., Vol. 31, October, 1945, pp. 35–7. 20 cents.

HOISTING — SAFETY DEVICES. — Testing safety catches on mine cages at some eastern bituminous coal mines. H. J. Sloman.—U.S. Bureau of Mines *Information Circular* 7290, July, 1944, 20 pp.

HOISTING — SAFETY DEVICES. — Testing safety catches on mine cages, Lake Superior district. Clyde M. Pearce and Frank E. Cash.—U.S. Bureau of Mines *Information Circular* 7325, June, 1945, 12 pp.

MINE LIGHTING.—Improving the portable lamp. C. S. Chubb.—*Colliery Engng.*, Lond., Vol. 23, February, 1946, pp. 31–2. 1s.

MINE LIGHTING.—Progress in mine lighting. A. H. Bull.—*Trans.* Instn. Min. Engrs., Lond., Vol. 101, Part 8, May, 1946, pp. 408–21. 6s.

MINE SIGNALLING.—Block signal system is safe and dependable. L. D. Stewart.—*Engng. Min. J.*, N.Y., Vol. 146, October, 1945, pp. 88–9. 50 cents.

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MINE SIGNALLING.—Suggested hoisting-signal code for metal-mine shafts. D. Harrington and J. H. East, Jr.—U.S. Bureau of Mines *Information Circular* 7289, July, 1944, 13 pp.

MINE SIGNALLING.—Suggested hoisting-signal code for slope coal mines and for shaft mines having only one level. D. Harrington and J. H. East, Jr.—U.S. Bureau of Mines *Information Circular* 7291, July, 1944, 9 pp.

MINING METHODS—TRACKLESS EQUIPMENT.—Trackless mobile equipment reduces mining costs. Howard O. Gray.—*Engng. Min. J.*, N.Y., Vol. 146, November, 1945, pp. 97–9. 50 cents.

NEW MEXICO—ARRIBA COUNTY.—Reconnaissance survey of the Headstone mining district, Rio Arriba County, New Mexico. T. D. Benjovsky.—Circular No. 11 of New Mexico School of Mines, State Bureau of Mines and Mineral Resources, Socorro, N.M., November, 1945, 10 pp. and map.

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GENERAL.—The minor metals of the world: their growing use and importance. Sir Thomas Holland.—*S. Afr. Min. Engng. J.*, J'burg, Vol. 56, Part II, December 8, 1945, p. 362. 6d.

MANGANESE — METALLURGY. — Manganese investigations: 19. Studies in the electro-deposition of manganese. F. W. Woodman and J. H. Jacobs.—U.S. Bureau of Mines *Report of Investigations* 3681, February, 1943, 26 pp.

MANGANESE—NEVADA—CLARK COUNTY.—Utilization of Three Kids manganese ore in the production of electrolytic manganese. 1.—Mining and electrolytic reduction of ore from Three Kids manganese mine, Clark County, Nev., by Glenn L. Allen, J. H. Jacobs, and J. W. Hunter. 2.—Acceptability of electrolytic manganese to the steel industry, by R. T. C. Rasmussen and F. Sillers, Jr.—U.S. Bureau of Mines *Report of Investigations* 3815, June, 1945, 78 pp.

MERCURY — CALIFORNIA. — Concurrent firing at the Sulphur Bank (Lake County) and Reed (Yolo County) quicksilver plants (Calif.). R. G. Hall and Worthen Bradley.—(A.I.M.E. Tech. Pub. No. 1889) *Metals Tech.*, N.Y., Vol. 12, December, 1945, 11 pp. $1.35.

MERCURY — CALIFORNIA. — Quicksilver deposits of Central San Benito and Northwestern Fresno Counties, California. Robert G. Yates and Lowell S. Hilpert.—*Calif. J. Min.*, Sacramento, Vol. 41, January, 1945, pp. 11–35.

MERCURY—U.S.A.—Bureau of Mines exploration of mercury deposits to June 30, 1944. McHenry Mosier.—U.S. Bureau of Mines *Information Circular* 7299, November, 1944, 16 pp.

TANTALUM.—World survey of tantalum ore. James S. Baker.—U.S. Bureau of Mines *Information Circular* 7319, March, 1945, 48 pp.

TANTALUM AND NIOBIUM.—Occurrence, extraction and uses of niobium and tantalum. Graham Oldham.—*Min. J.*, Lond., Vol. 226, January 12, 1946, pp. 29–30. 8d.

URANIUM.—The world's uranium supplies: difficulties of effective control.—*S. Afr. Min. Engng. J.*, J'burg, Vol. 56, Part II, November 10, 1945, p. 239. 6d.

ZIRCONIUM—AUSTRALIA.—Titanium and zirconium minerals: a war-developed Australian industry.—*Chem. Tr. J.*, Lond., Vol. 118, February 8, 1946, p. 169. 8d.

NON-METALLIC MINERALS.

ASBESTOS — PALIGORSKITE. — Paligorskite—a possible asbestos substitute. Robert B. Fisher, Robert L. Thorne, and Corbin Van Cott.—U.S. Bureau of Mines *Information Circular* 7313, March, 1945, 5 pp.

NON-METALLIC MINERALS—*continued.*

ASBESTOS—QUEBEC.—Asbestos-fiber exploration and production forecasts by core drilling, Jeffrey mine, Asbestos, Quebec. George K. Foster and Charles D. Borror.—(A.I.M.E. Tech. Pub. No. 1952) *Min. Tech.*, N.Y., Vol. 10, January, 1946, 9 pp. $1.35.

BUILDING STONE.—The wire saw in quarrying dimension stone. P. De Vitry and Willis P. Mould.—(A.I.M.E. Tech. Pub. No. 1949) *Min. Tech.*, N.Y., Vol. 10, January, 1946, 7 pp. $1.35.

CALCITE—MONTANA.—Methods of prospecting and mining optical calcite in Montana. E. W. Newman.—(A.I.M.E. Tech. Pub. No. 1896) *Min. Tech.*, N.Y., Vol. 9, November, 1945, 10 pp. $1.35.

FLOTATION—CEMENT ROCK.—Cationic flotation of cement rock. J. C. Williams.—(A.I.M.E. Tech. Pub. No. 1901) *Min. Tech.*, N.Y., Vol. 10, January, 1946, 5 pp. $1.35.

FLUORSPAR — DURHAM. — A projected central mill for the Durham fluorspar industry. Andrew Pearson.—*Instn. Min. Metall. Bull.*, Lond., No. 476, January, 1946, 19 pp. 1s. 6d.

FLUORSPAR — FLOTATION. — Filtering fluorspar flotation concentrates. W. H. Reck. —*Deco Trefoil*, issued by Denver Equipment Co., Denver, Colorado, November-December, 1945, 1 p.

FLUORSPAR—QUEENSLAND.—Fluorspar deposits, Emuford, Almaden, and Chillagoe districts: Geological Survey report. J. E. Ridgway.—*Qd. Govt. Min. J.*, Brisbane, Vol. 46, November, 1945, pp. 326–33. 6d.

GRAPHITE — ONTARIO — RENFREW COUNTY.—Black Donald graphite: Canada's most important graphite deposit. D. C. McLaren.—*Canad. Min. J.*, Gardenvale, P.Q., Vol. 66, November. 1945, pp. 763–72. 25 cents.

KAOLIN—MYSORE.—Scope for developing china clay industry in Malnad (Mysore, India). B. N. Raghunatha Rao.—Mysore Geological Department Popular Studies No. 4, Bangalore, 1945, 9 pp. 8 annas.

SILICA — AGATE — QUEENSLAND. — Re agate—Agate Creek, Percyville: Geological Survey report. J. E. Ridgway.—*Qd. Govt. Min. J.*, Brisbane, Vol. 46, October, 1945, pp. 299–300. 6d.

SILICA—GLASS SAND—PORTO RICO.—Glass sand and a glass industry in Puerto Rico. Howard A. Meyerhoff and J. Earl Frazier.—(A.I.M.E. Tech. Pub. No. 1939) *Min. Tech.*, N.Y., Vol. 9, November, 1945, 9 pp. $1.35.

SILICA—MOULDING SANDS.—The composition and properties of molding sands—Part 1. The nature of the A.F.A. clay fraction removed from natural molding sands. Louis H. Berkelhamer.—U.S. Bureau of Mines, *Report of Investigations* 3774, September, 1944, 58 pp.

INDEX OF RECENT ARTICLES—*continued.*

NON-METALLIC MINERALS—*continued.*

SILICA — QUARTZ CRYSTAL. — Quartz crystal as a mineral resource. Robert B. McCormick.—(A.I.M.E. Tech. Pub. No. 1916) *Min. Tech.*, N.Y., Vol. 9, November, 1945, 4 pp. $1.35.

TALC—USES.—Survey of the suitability of domestic talcs for high-frequency insulators. Theron A. Klinefelter, Sidney Speil, and Sidney Gottlieb.—U.S. Bureau of Mines *Report of Investigations* 3804, April, 1945, 58 pp.

TITANIUM — ARKANSAS. — Mining and milling operations of rutile mine of the Titanium Alloy Co. of Arkansas, Hot Spring County, Ark. Felix A. Vogel, Jr.—U.S. Bureau of Mines *Information Circular* 7293, September, 1944, 7 pp.

PETROLEUM.

ALASKA.—Oil seepages on the Alaskan Arctic slope: numerous evidences of petroleum found in latest reconnaissance. Norman Ebbley, Jr.—*Min. Metall.*, N.Y., Vol. 25, September, 1944, pp. 415–9. 50 cents.

ASPHALT.—The chemistry and constitution of asphalts. B. A. Murphy.—*J. Inst. Petrol.*, Lond., Vol. 31, December, 1945, pp. 475–85. 7s. 6d.

GEOLOGY — RUMANIA. — Notes on the geology of the Southern Rumanian oil district with special reference to the occurrence of a sedimentary laccolith. J. F. M. De Raaf.—*Quart. J. Geol. Soc.*, Lond., Vol. 101, Parts 1 and 2, 1945, pp. 111–30. 7s. 6d.

OKLAHOMA.—The West Edmond oil field of Oklahoma. E. G. Dahlgren and Dan O. Howard.—*Min. Metall.*, N.Y., Vol. 26, December, 1945, pp. 607–10. 50 cents.

PROSPECTING — STRUCTURE DRILLING.—Structure drilling in oil exploration, Southern Alberta. G. M. Furnival and W. M. Tovell.—*Trans.* Canad. Inst. Min. Metall., Vol. 48, 1945, pp. 726–44. (*Canad. Inst. Min. Metall. Bull.*, Montreal, No. 403, November, 1945. $1.00.)

PETROLEUM—*continued.*

SHALE OIL.—Shale oil.—*Engineer*, Vol. 180, 1945; December 21, pp. 49 December 28, pp. 520–21. 1s. each.

WELL DRILLING—PENNSYLVA Horizontal drilling for oil in Pennsylv preliminary report. C. W. Elder, Jr Bureau of Mines *Report of Investigation* September, 1944, 68 pp.

PLANT AND POWER.

COMPRESSED AIR.—Compressed ai lopments in coal mining—III: a re recent progress. J. I. Davies.—*Com Engng.*, Lond., Vol. 10, December, 19 147–50. 1s.

ELECTRIC POWER—DISTRIBUT Mining type plugs, sockets and coup flexible cables. G. Denholm.—*Minin trical and Mechanical Engineer*, Man Vol. 26, January, 1946, pp. 247–52.

PRECIOUS STONES.

DIAMOND — ECONOMICS.—The d industry in 1944, 20th annual review. H. Ball. (Abstract from *Jewelers' Ci Keystone.*)—*Min. J.*, Lond., Vol. 226 January 12, pp. 26–9; January 19, pp January 26, pp. 66–7; February 2, p 8d. each.

DIAMOND—INDUSTRIAL DIAMO Selection of industrial diamonds Grodzinski.—*Min. J.*, Lond., Vol. 226, 5, 1946, pp. 5–7. 8d.

RUBY.—The ruby—gem of the East notes on the history, occurrence and of the ruby. D. C. McLaren.—*Cana J.*, Gardenvale, P.Q., Vol. 66, Decembe pp. 852–4. 25 cents.

TIN.

ASIA—EAST.—The tin industry c Asia.—*Min. J.*, Lond., Vol. 225, Decen 1945, pp. 886–8. 8d.

CONCENTRATION.—The dressing ores: examples in practice. F. B. Mi *Mine & Quarry Engng.*, Lond., Vol. 11 January, pp. 15–19; February, pp. 1s. each.

[redacted]

Subject to revision.] [*Paper issued on March 14th,* 1946, *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, Burlington House, Piccadilly, London, W.1, on Thursday, March 21st,* 1946, **at 5.30 o'clock p.m.**

A Study of Sizing Analysis by Sieving.

By Harold Heywood, Ph.D., M.Sc., A.C.G.I., M.I. Mech.E.[*]

Introduction

Sieving is applied in industry to a very large tonnage of powdered materials and is the most frequently used procedure for the sizing analyses of such materials in the laboratory, both for research and for the control of industrial processes. The importance of sub-sieve particle-size analysis has recently received great prominence, and rightly so, but the possibility of improving the technique of sieving should not on that account be neglected. The following paper, describing researches made by the author on sieving, shows that with proper attention to standardization of procedure, sieving may attain the accuracy and precision required by modern methods for the control of industrial and mining processes.

Standard Sieves and Perforated Plates

In 1907 the Institution of Mining and Metallurgy recognised the need for standardization of sieve apertures and introduced the I.M.M. sieve series. These sieves covered the range from 5 to 200 meshes per lineal inch, and in all cases the aperture was equal to the diameter of the wire, giving an effective sieving area of 25 per cent of the area of the cloth. The American Society for Testing Materials adopted a series using finer wires, these sieves thus having a larger effective sieving area. The lower limit of this series is 400 meshes per lineal inch, aperture 37 microns (1 mm. = 1,000 microns), but a liberal tolerance has to be allowed in the weaving of such fine meshes. The British Standards Institution introduced in 1931 a sieve series which, using standard gauges of wire, conformed more

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closely than the I.M.M. series to the American and Continental sieve series. A revised British Standard series, issued in 1943, divided test sieves into special and normal grades, the nominal aperture being the same for a given sieve mesh in each grade, but with the tolerances reduced for the special grades. In this B.S. series fine mesh sieves extend over the range 5 to 300 meshes per lineal inch: medium mesh sieves extend from $\frac{1}{32}$-in. to $\frac{1}{2}$-in. aperture, and plates perforated with square holes from $\frac{3}{16}$-in. to 4-in. aperture. The introduction of the perforated plates was largely due to the investigations of Markwick,(1)* who showed that apertures greater than $\frac{3}{16}$ in. could be punched in plate much more accurately than sieves of corresponding aperture could be woven.

The procedure recommended for the optical examination of test sieves is described in B.S. 410, 1943. The average and maximum apertures are restricted to definite tolerances, the latitude being greater with the finer sieves, and in addition intermediate apertures are specified with the proviso that not more than six per cent of the apertures shall be beyond the limits of either of these intermediate apertures. Extensive researches by MacCalman(2) failed to determine any method by which the effective aperture of a sieve could be calculated from the results of optical measurements of the apertures. It is clear, however, that oversize apertures are more undesirable than undersize apertures, since the latter are merely ineffective, whilst with prolonged sieving the former result in an appreciable increase in the size of particle passed by the sieve. The effective aperture of a sieve is thus likely to be greater than the mean aperture determined by optical examination and the system of intermediate aperture tolerances was introduced in order to reduce this difference.

Effect of Irregular Shape of Particles

Sieving has the object of sizing particles whose relative dimensions of length, breadth, and thickness may vary widely. It is thus important to determine which dimension of a particle is most closely related to the sieve aperture through which it will just pass. The cross section of the particle passing must obviously lie within the square area representing the sieve aperture, but the length is not so restricted: for example, pieces of wire up to eight times the sieve aperture in length may easily be made to pass by normal sieve shaking.

*Figures in parentheses refer to references given at the end of the paper.

The diagrams in Fig. 1 show how the shape of a particle affects the apparent size corresponding to a given square aperture. If the particle is approximately square in cross section, then the breadth is equal to the aperture dimension; if the particle is thinner, but has an approximately rectangular cross section, then the breadth is greater than the side dimension of the aperture but less than the diagonal of the aperture. If, however, the edges of the particle are sharp, or the particle is very thin, then the breadth is equal to the diagonal of the aperture.

Thus whilst the thickness of the particle may vary from a negligible figure to the aperture side dimension, the breadth is

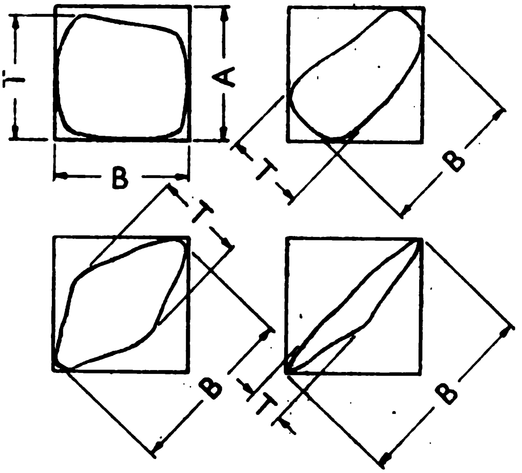

Fig. 1.—Effect of shape on relationship between breadth of particle and sieve aperture.

restricted to the limits of 1 to 1.414 times the aperture side dimension, and, therefore, it is the breadth of the particle that is most closely related to the aperture size.

The extent of the variation in length, breadth, thickness, and volume occurring in practice was determined by measurements on 142 crushed sandstone particles that had been graded by hand placing to be retained on a 1-in. square aperture and to pass a $1\frac{1}{8}$-in. square aperture. The breadth, thickness, length and volume of each particle were measured, and the mean values, the standard deviations* and the maximum variations are shown in Table I.

These results show clearly that there is least variation in the breadth of the particles, whereas the extreme values of volume have a ratio of 5 to 1 in spite of the very close grading of the test

*Standard deviation is defined in Appendix I.

TABLE I

VARIATIONS IN PARTICLE DIMENSIONS

Dimension	*Limiting Values, cm. or cu. cm.*	*Mean in cm. or cu. cm.*	*Standard Deviation, per cent of Mean*	*Maximum Variation, per cent of Mean*
Sieve aperture	2·54— 2·86	2·70	—	— 6 ; + 6
Breadth	2·50— 4·00	3·23	9·6	—23 ; + 24
Thickness ...	0·71— 2·92	1·92	23·7	—63 ; + 52
Length	2·75— 9·0	4·72	24·0	—42 ; + 91
Volume	4·7 —24·0	11·31	32·8	—59 ; +112

particles. Frequency or distribution curves of the variations in length, breadth, and thickness are shown in Fig. 2, and it may be

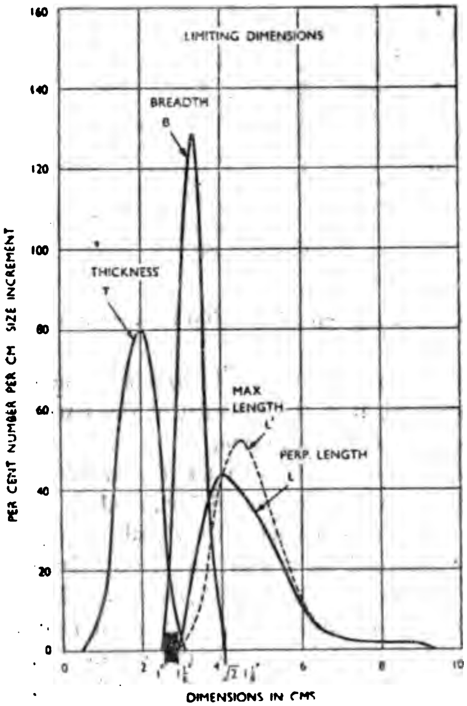

FIG. 2.—Variation in particle dimensions.
L measured perpendicular to B.
L′ maximum dimension.

deduced from these that the average diameter of particles as seen under the microscope, which is usually intermediate between the length and the breadth, is greater than the size of sieve aperture though which they have passed. This ratio between mean projected diameter and sieve aperture is usually about 1.4, but more detailed figures have been given in a previous publication by the author.[3]

Standardization of Sieving Procedure

The permissible tolerances of test sieves have now been reduced to the minimum commensurate with the technique of commercial weaving, hence further efforts to reduce variations between sieving analyses made in different laboratories must be applied to the actual process of sieving.

Sieving is a statistical process—that is, there is always an element of chance as to whether a particular particle of 'near mesh' dimensions will or will not pass through the sieve. Thus there is no definite 'end point' of a sieving analysis, and this must be fixed by means of some conventional agreement.

Specifications have been issued for defining the 'end point' by the following methods :—

(*a*) By means of a standard time of sieving.

(*b*) By sieving until the weight of material passing the sieve per minute is less than a specified percentage of the weight of sample taken.

(*c*) By sieving until the weight of material passing the sieve per minute is less than a specified percentage of the weight of residue on the sieve concerned.

Method (*a*) is applied in B.S. No. 12, 1940, for Portland cement, which states that sieving on the 170-mesh sieve is to be continued for 15 minutes. Method (*b*) is specified by the A.S.T.M.[4] for the sieving analysis of pulverized coal, the limiting rate of passing being 0.1 per cent of the sample weight per minute. Method (*c*) is adopted by A.S.T.M. Standard C41-33 for the grading of road aggregates, the limiting rate being 1 per cent of the weight of material on the sieve per minute. Definition of the 'end point' by means of a specified sieving rate would appear to be fundamentally more accurate, but is clearly more tedious to apply in practice, and for most purposes a specified sieving time would be sufficiently accurate.

The process of sieving occurs in two stages. First, the elimination of the fine dust, which usually takes place fairly rapidly, and, secondly, the elimination of the 'near mesh' particles, which occurs at a gradually diminishing rate. It is this second elimination which creates the difficulty in defining the 'end point,' for it is affected by the method of shaking the sieve, the weight of sample used for the test, the shape of the particles, and the moisture content of the material and even of the atmosphere.

The author has very definite opinions on one feature of sieve analyses—namely, that a preliminary sieving should be conducted on the finest sieve of the series to be used. This removes the bulk of the fine dust and avoids clogging of the coarser sieves in succession, although some fine dust frequently adheres to the coarser particles in spite of prolonged sieving.

The extent of this effect was shown by experiments made on the dry sieving of ground silica on a 200-mesh B.S. sieve, in which a detailed examination was made of the material passing the sieve after successive periods of sieving. A 50-gram sample was shaken on the sieve by hand and the amount passing after each minute was weighed; complete particle size determinations were made on some of these fractions by means of an apparatus of the turbidimeter type, in which the rate of settlement of the particles in water was determined by photo-electric measurements of the opacity of the suspension. The results of the tests with dry sieving are given in Table II.

TABLE II

DRY SIEVING TEST ON GROUND SILICA, 200-MESH B.S. SIEVE

Sieving Period, minutes	*Percentage of Sample passing Sieve*	
	Per Minute of Sieving Period	*Total at end of Period*
0 — 1	42·90	42·90
1 — 2	15·48	58·38
2 — 3	9·84	68·22
3 — 4	4·40	72·62
4 — 5	2·56	75·18
5 — 6	1·10	76·28
6 — 7	0·82	77·10
7 — 8	0·61	77·71
8 — 9	0·42	78·13
9 —10	0·34	78·47
10 —11	0·26	78·73
11 —13	0·28	79·29
13 —15	0·21	79·71
15 —17	0·21	80·14
17 —20	0·15	80·59

Curves of percentage undersize against particle size calculated from Stokes' equation for the terminal velocity of fall are given in Fig. 3 for some of these fractions. During the first minute of sieving the undersize consisted mainly of particles ranging from 0 to 50 microns, thus representing the fine dust present in the sample. Separation of the fine dust and the ' near mesh ' particles began to take place in the fourth minute of sieving, as shown by the double inflexion in the grading curve. The particles passing during the seventh minute were clearly separated into fine dust below 20

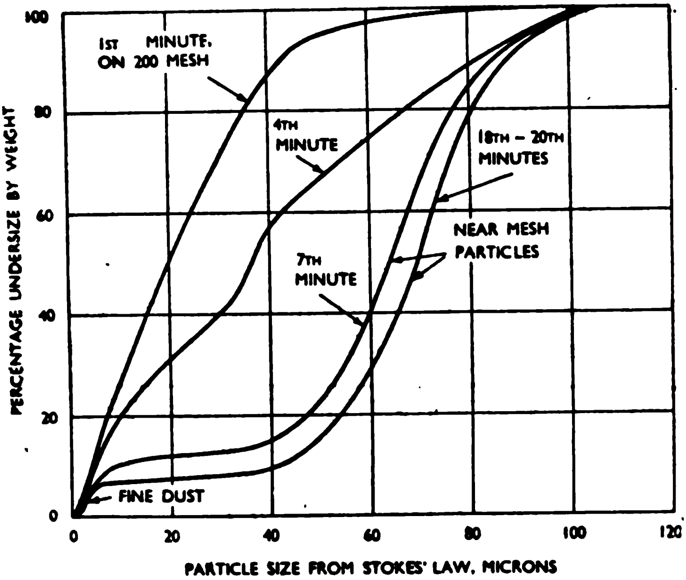

FIG. 3.—Variation in particle size with sieving time—dry sieving test.

microns and amounting to 12 per cent by weight and ' near mesh ' particles ranging from about 40 to 100 microns, with very few intermediate sized particles. During the period 17 to 20 minutes the adherent fine dust was reduced to 7 per cent by weight and to less than 10 microns in size.

A similar test was made for comparison in which the fine dust was eliminated by wet sieving. The 50-gram sample of ground silica was first washed on a 300-mesh sieve, the residue dried, and then sieved on the 200-mesh sieve. The results of this test are given in Table III and selected undersize curves in Fig. 4.

TABLE III

WET SIEVING TEST ON GROUND SILICA, 200-MESH B.S. SIEVE

Sieving Period, minutes	*Percentage of Sample passing Sieve*	
	Per Minute of Sieving Period	*Total at end of Period*
Wet sieving on 300-mesh	—	67·36
Dry sieving on 200-mesh		
0 — 1	11·94	79·30
1 — 2	1·30	80·60
2 — 3	0·51	81·11
3 — 4	0·50	81·61
4 — 5	0·27	81·88
5 — 7	0·20	82·27
7 — 9	0·13	82·53
9 —12	0·08	82·77

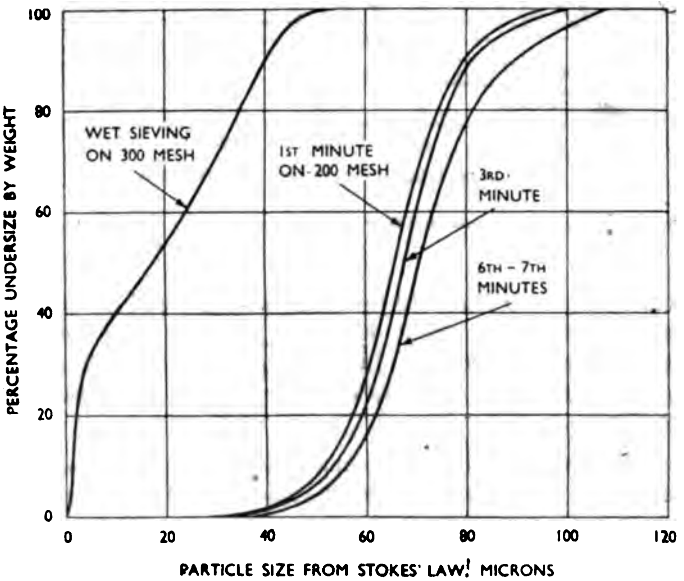

FIG. 4.—Variation in particle size with sieving time—wet sieving test.

This test shows the very rapid approach to completion when the fine dust has previously been removed by wet sieving, and the complete absence of fine dust adhering to the coarser particles. The curves also show the influence of oversize sieve apertures when sieving is prolonged.

Not only does fine dust adhere to coarser particles, but aggregates of very fine particles may occur in samples of dry powders which are strong enough to resist the abrasive action of sieving but are dispersed when the powder is wetted. This effect is particularly marked if the material contains clay—e.g., as in moulding sands. The test results shown in Table IV demonstrate the magnitude of the difference that may be obtained between the results of dry and wet sieving on the same material.

TABLE IV

COMPARISON OF DRY AND WET SIEVING

Material	*Per cent through 100-mesh Sieve*		*Per cent through 200-mesh Sieve*	
	Dry	*Wet*	*Dry*	*Wet*
Moulding sand	74·9	80·5	8·5	22·7
,, ,, ...	90·5	96·4	16·0	42·5
Moulding loam	96·8	99·4	11·6	31·4
,, ,, ...	94·2	99·9	15·9	48·2
,, ,, ...	98·4	99·8	30·6	56·9
Ground silica	98·0	98·3	81·0	83·5
,, ,, ...	—	—	77·9	81·7
,, ,, ...	—	—	91·6	95·4

The proportion of material passing the sieve is increased in all cases by wet sieving and this method would obviously be used for the grading of mineral ground in the wet condition, but dry ground mineral to be used dry should if possible be sieved dry, since the analysis thus obtained represents the effective particle size distribution. Any system of standardization of sieving methods must therefore distinguish between the two methods and define the conditions to which they are respectively applicable.

The effect of sample weight on the time required to complete a sieving analysis and on the result was investigated by experiments in which coal dust was dry-sieved on British Standard sieves 8 in. in diameter. Samples of 20, 50, and 100 grams weight were sieved until the rate of passing per minute was less than 0.1 per cent of the sample weight. The results, shown in Table V, show that neither the time required to attain the above rate of sieving nor the total percentage passing the sieve at this time were appreciably affected by the weight of sample.

TABLE V

EFFECT OF SAMPLE WEIGHT ON SIEVING

Sieve Mesh	*Weight of Sample, grams*	*Time to attain rate of 0·1 per cent per minute, minutes*	*Percentage passing at time to attain 0·1 per cent per minute*
60	100	2·8	96·9
	50	3·3	96·8
	20	3·0	96·9
100	100	6·8	89·3
	50	6·7	89·1
	20	6·6	89·0
150	100	13·6	76·3
	50	14·0	76·3
	20	13·8	76·6
200	100	18·2	60·4
	50	18·0	61·6
	20	17·2	61·3

The following sample weights are suggested as being the maximum suitable for the sieving analyses of various minerals on 8-in. diameter sieves :—

Coal dusts and pulverized coals.................. 50 grams.
Other minerals of specific gravity less than
3.1 .... 50 to 100 grams.
Minerals of specific gravity 3.1 or greater ... 100 grams.

A minimum weight of 20 grams is suggested in order that the sample may represent the bulk, but there may of course be occasions on which this amount of material is not available.

Correlation of Sieving Analyses

The foregoing sections of the paper have shown how the results of a sieving analysis are affected by the particle shape, the method of sieving, and the weight of sample. The experiments described below were devised to show the extent of the variation in the analyses made at different laboratories when the same material and weight of sample was used throughout.

A system of sieve calibration for the analyses of Portland cement was devised by the U.S. Bureau of Standards ([5]) by which samples of a standard cement that had been graded on a master sieve were supplied to laboratories for grading on their own sieves. By this means a residual weight correction was obtained, which could be

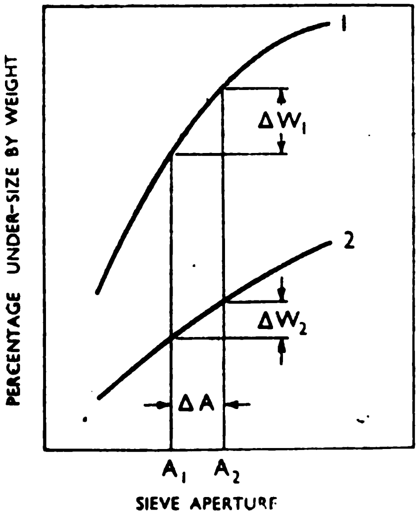

Fig. 5.—Variation with fineness of weight corrections for sieve.

applied to subsequent analyses to convert these to results which would have been obtained with the master sieve of the Bureau of Standards.

A consideration of the curves in Fig. 5 shows, however, that such weight corrections can only be constant for a fixed fineness of powder. Let A_1 be the mean aperture of the sieve which is assumed standard and A_2 the aperture of another sieve termed the user's. If the curve (1) represents part of the grading curve for a particular powder as determined by the standard series of sieves, and if it is assumed that the mean aperture A_2 is greater than A_1, then more of the powder will pass through the user's sieve and the weight correction to bring the analyses into agreement is ΔW_1 for an aperture error of Δ A. If, however, another powder of different fineness is being analysed, as shown by the curve (2), then the weight correction ΔW_2 will also be different.

Samples of coal dust were submitted to seven different laboratories and a 50-gram sample sieved on B.S. sieves of 72, 100, 150, and 200 mesh. The average of the seven sieving analyses was plotted against the nominal aperture of the corresponding sieves, and from this curve the aperture error of each individual sieve was determined for three finenesses of coal dust.* The standard deviation of the seven aperture errors for each sieve was then calculated and expressed as a percentage of the nominal sieve aperture, thus giving an indication of the probable variation in the sieving analyses for the conditions of that particular test. The average fineness of the various samples of coal dust is shown by the figures in Table VI.

TABLE VI

AVERAGE SIEVING ANALYSES OF COAL DUST SAMPLES

Sample No.	*1st Series*			*2nd Series*			*3rd Series*	
	1	2	3	4	5	6	8	9 to 15
Cumulative per cent passing sieve :								
52 mesh	76·9	99·4	99·6	71·6	99·1	99·6	84·2	—
72 mesh	60·8	93·8	97·4	57·1	95·9	98·8	—	99·3
100 mesh	49·1	83·1	90·7	46·7	87·8	96·3	51·8	97·5
150 mesh	36·6	64·9	75·2	35·4	71·4	86·9	—	91·0
200 mesh	27·2	51·2	58·3	27·5	56·9	72·7	25·7	82·2

The same sample of dust cannot be re-sieved, owing to the slight loss of fines and also because of possible degradation during the sieving process ; hence a separate sample from the bulk was sent to each laboratory and the sieving errors included also errors due to sampling. This was, however, no disadvantage, for in practice these two effects are necessarily combined and three different devices for sub-dividing the bulk sample were tested.

Sub-divider No. 1 consisted of 12 sector-shaped receptacles mounted on a rotatable plate, the bulk sample being delivered

*An example of the method of determining the aperture errors and the standard deviation is described in Appendix I. For the purpose of interpreting the results of these tests, the standard deviation may be regarded as a relative measure of the probable error involved by the method of test.

through a chute, so that a representative portion was collected by each receptacle as it passed underneath. This device is described fully in B.S. No. 735—1937, p. 76.

Sub-divider No. 2 consisted essentially of a small plate pivoted horizontally at the top edge and oscillated through an angle of approximately 30° by means of a link connected to a crank rotating at 240 r.p.m. The bulk sample was delivered through a chute mounted centrally above the top edge of the plate and was thus divided into two portions which were collected in separate receptacles.

Sub-divider No. 3 was a micro-riffler made according to dimensions specified in the *Engineering and Mining Journal*(6).

In the first series of comparative sieving tests 50-gram samples of coal dusts Nos. 1, 2, and 3, which had been prepared by sub-divider No. 2, were sent to the seven co-operating laboratories. These samples were sieved by the procedure normally used at each respective laboratory—i.e., there was no restriction as to the time of sieving, or as to the method of shaking the sieves. In the second series of tests 50-gram samples of coal dusts Nos. 4, 5 and 6 were prepared in triplicate by sub-dividers Nos. 1, 2, and 3. Each laboratory agreed to use a sieve-shaking machine and to adopt the following method of sieving. The sample of 50 grams weight was first sieved for a preliminary period of five minutes on the 200-mesh sieve to remove the greater portion of the fine dust. This sieve was then brushed to remove adherent dust and the residue placed on the nest of sieves 72, 100, 150 and 200 mesh, in this order. The nest of sieves was vibrated for a period of 10 minutes, when each sieve was brushed, the residue replaced, and sieving continued for another period of 10 minutes. The residues on each sieve were then weighed, the percentage passing 200-mesh being obtained by difference from the sample weight.

A slight loss in weight during sieving is inevitable and this is assumed to consist of fine dust, but the loss should not exceed 0.5 per cent of the weight of the sample.

The third series of tests was made by sieving eight 50-gram samples of pulverized coal on the same sieves and using the method described above. The test was repeated with several operators and different makes of sieving machine and hand shaking was used for one set of tests. Table VII gives the standard deviation of the aperture errors expressed as a percentage of the nominal sieve aperture for each set of experiments described above. The efficacy of the three different sub-dividers was substantially equal, although No. 1 gave slightly more uniform results.

Table VII

Standard Deviations of Aperture Errors Expressed as Percentage of Nominal Sieve Aperture

Test Series	*Sub-Divider No.*	*Sample No.*	*Sieve Mesh* 52	72	100	150	200	*Average for all Sieves*
1st Series	2	1 (coarse)	3·93	7·26	8·22	6·06	12·23	8·44*
	2	2 (med.)	—	8·54	7·96	6·16	12·90	8·89
	2	3 (fine)	—	6·64	7·70	7·31	8·68	7·58
	2	1, 2 and 3	3·93	7·48	7·96	6·51	11·27	8·30*
2nd Series	1	4 (coarse)	2·92	3·13	4·15	2·88	3·03	3·30*
	1	5 (med.)	—	4·62	4·51	3·22	2·80	3·79
	1	6 (fine)	—	—	5.00	2·65	1·91	3·19
	1	4, 5 and 6	2·92	3·87	4·55	2·92	2·58	3·48*
	2	4 (coarse)	3·22	4·26	5·26	6·24	3·95	4·93*
	2	5 (med.)	—	4·55	4·47	2·88	1·28	3·30
	2	6 (fine)	—	—	5·33	2·47	1·23	3·01
	2	4, 5 and 6	3·22	4·40	5·02	3·86	2·15	3·86*
	3	4 (coarse)	2·03	2·75	4·93	3·36	3·95	3·75*
	3	5 (med.)	—	5·60	4·31	2·61	3·01	3·88
	3	6 (fine)	—	—	5·20	2·77	2·74	3·57
	3	4, 5 and 6	2·03	4·17	4·81	2·91	3·23	3·78*
	1, 2 and 3	4, 5 and 6	2·72	4·15	4·79	3·23	2·65	3·71*
3rd Series	1	8 (med.)	0·44	—	0·46	—	0·62	0·54*
(Machine sieving)	3	9 (fine)	—	1·20	0·62	0·50	0·56	0·56†
	3	10 (fine)	—	0·75	0·72	0·63	0·63	0·66†
	3	11 (fine)	—	1·40	0·36	0·35	0·42	0·38†
	3	12 (fine)	—	1·30	0·79	0·90	1·02	0·90†
	3	13 (fine)	—	1·70	0·40	0·78	0·65	0·61†
	—	8 to 13	0·44	1·27	0·56	0·63	0·65	0·61*†
(Hand sieving)	3	15 (fine)	—	0·40	0·38	0·66	1·37	0·80†

*Excludes 52-mesh results, as only one sample tested on this mesh.
†Excludes 72-mesh results, as residues were very small.

The average results for all samples, sub-dividers and sieves are summarised in Table VIII.

TABLE VIII

SUMMARY OF COMPARATIVE SIEVING TESTS

Test Conditions	*Standard Deviation of Aperture Errors as Percentage of Nominal Sieve Aperture*
1st Series : Different sieves, different methods...	8·30
2nd Series : Different sieves, same method	3·71
3rd Series : Same sieves, same method :	
(a) Machine sieving..................	0·61
(b) Hand sieving.....................	0·80

The figures in Table VIII show that if machine sieving on the same sieves and by the same method is taken as a standard of reference, then the probable errors are increased about 6 times when different sieves are used but the same method of sieving is employed, and about 14 times when the method of sieving is not controlled. Thus the discrepancies between analyses made in different laboratories could be halved if a standard method of sieving were adopted. Aperture errors determined in the manner described above enabled sieving analyses made in different laboratories to be correlated with a high degree of precision. This section of the research was conducted by the British Coal Utilisation Research Association with the assistance of a number of co-operating laboratories, and is published by permission of the Director.

CONCLUSIONS

The preceding sections of the paper have shown that the process of sieving is of a statistical nature and that the 'end point' of a sieving analysis can only be defined by the adoption of a convention —such as a specified time or rate of sieving. The greatest practical accuracy in the weaving of sieve cloth will have been attained with the revised British-Standard for 'special' test sieves in operation and further improvements in the technique of sieving can only be effected by standardization of the method of conducting the test. The comparative sieving tests described in the paper have shown that the probable error of a sieving analysis may be halved by the adoption of a standard method of sieving. It is clear, however, that one standard method cannot be applied to all materials ; provision must be made for dry and wet sieving, and probably the screening of coarse material will need a different method of treatment than the sieving of finely-ground powders.

The writer considers that it would not be unduly difficult to devise alternative specifications, with a guide to their applicability, which would enable a standardized method to be applied to sieving analyses of the majority of powdered materials used in industry. The first stage in any such standardization would be to collate data on sieving methods from all scientific and industrial research

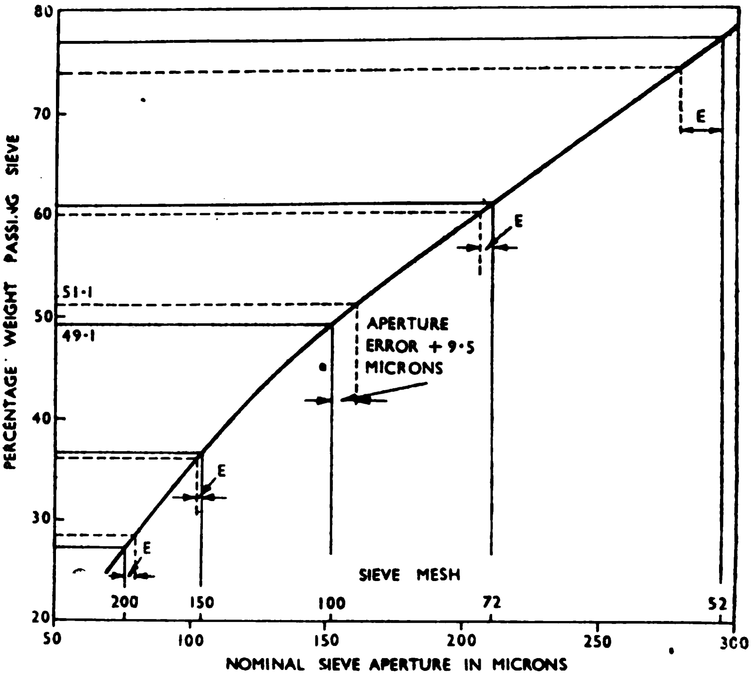

Fig. 6.—Determination of sieve aperture errors.

associations that are concerned with powdered materials, and it is to be hoped that in the near future this will be put into operation by some official organization.

Appendix I

The procedure used to calculate the standard deviation of the aperture errors is illustrated by the following examples. Table IX shows for a particular sample the percentage passing the various sieves, both the average of the analyses from the seven laboratories and the results obtained at a particular laboratory 'X'. The averaged results are plotted against the nominal apertures of the sieves as in Fig. 6, the curve being drawn through the intersection points of the unbroken lines. Horizontal dotted lines are drawn on the same diagram through the percentages passing the sieves as determined at laboratory ' X ', and assuming that the sample is the same (or alternatively including sampling errors with sieving errors), then the effective apertures of the sieves used at laboratory ' X ' are defined by the intersection of these horizontal dotted lines and the curve. The difference between the vertical unbroken and dotted lines is thus the aperture error, marked E on the figure—e.g., for 100-mesh sieve this is +9.5 microns. The effective apertures and the aperture errors for the sieves at laboratory ' X ' are also given in Table IX.

TABLE IX

SIEVING ANALYSES AND APERTURE ERRORS

Sieve Mesh B.S.	*Nominal Aperture, Microns*	*Per Cent Passing Sieve ; Average of Analyses at Seven Laboratories*	*Results at Laboratory ' X '*		
			Per Cent Passing Sieve	*Effective Aperture, Microns*	*Aperture Error, Microns*
52	295	76·9	74·0	280	—15
72	211	60·8	60·0	207	— 4
100	152	49·1	51·1	161·5	+ 9·5
150	104	36·6	36·0	102	— 2
200	76	27·2	28·5	80	+ 4

TABLE X

SIEVE APERTURE ERRORS IN MICRONS FOR VARYING FINENESS OF COAL DUST

Sample No.	*Sub-Divider No.*	*Sieve Mesh, B.S.*			
		72	100	150	200
4 (Coarse)	1	— 6·5	+ 2	+ 6·0	+ 4·0
	2	— 7·5	+ 6	+ 10·5	+ 5·0
	3	— 6	+ 3·5	+ 6·0	+ 1·5
5 (Medium)	1	— 6	+ 4	+ 5·7	+ 1·8
	2	— 4·5	+ 6	+ 5·2	+ 2·0
	3	— 7	+ 2	+ 4·9	+ 0·5
6 (Fine)	1	Residues too small		+ 5·6	+ 1·5
	2			+ 4·3	+ 1·0
	3			+ 5·0	+ 2·0
Mean error, microns..		— 6·2	+ 3·9	+ 5·9	+ 2·1
As percentage of nominal aperture		— 2·1	+ 2·6	+ 5·7	+ 2·8

The figures in Table X show the aperture errors as determined at laboratory ' Y ' for the three grades of coal dust. The agreement for a particular sieve is reasonably good, considering the statistical nature of the processes of sampling and sieving.

The aperture errors for a particular sieve mesh were determined as already described for each laboratory and the standard deviation calculated as shown below by the following example for 100-mesh sieve (Table XI).

TABLE XI

Aperture Error, E, in Microns	E^2
— 8	64·0
— 0·5	0·2
—11·5	132·2
—14·5	210·1
+10	100·0
+18·5	342·2
+ 9·5	90·3
Total of E^2, ($E\Sigma^2$) ...	... 939·0

The standard deviation is defined as $\sqrt{\Sigma E^2/(n-1)}$,(7) where n is the number of observations, and applied to this example the standard deviation is 12.5 microns, which is 8.22 per cent of the nominal sieve aperture. The figure thus determined forms one of the results given in Table VII (top line, 100 mesh), and the other figures were calculated in a similar manner. Hence the final averages represent a very large number of individual tests.

REFERENCES

(1) MARKWICK, A. H. D. ' A Study of Sieves for Coarse Aggregates '. *Journ.* Soc. of Chem. Ind., 1940, Vol. 59, p. 88.

(2) MACCALMAN, D. ' The Accuracy of Sieving Tests '. *Industrial Chemist*, 1937, Vol. 13, pp. 464, 507 ; 1938, Vol. 14, pp. 64, 101, 143, 197, 231, 306, 363, 386, 498 ; 1939, Vol. 15, pp. 161, 184, 247, 290.

(3) HEYWOOD, H. ' Measurement of the Fineness of Powdered Materials '. *Proc.* Inst. Mech. E., 1938, Vol. 140, p. 257.

(4) *Proc.* Amer. Soc. Testing Mat., 1926, Vol. 26, part I, p. 898.

(5) WIG, R. J., and PEARSON, J. C. ' Standardization of No. 200 Cement Sieves '. U.S. Bur. Standards, Tech. Paper No. 42, 1914.

(6) ' Splitting Small Samples Accurately with the Microsplitter '. *Eng. Min. Journ.*, 1937, Vol. 138, p. 185.

(7) FISHER, R. A. : ' Statistical Methods for Research Workers ', p. 47.

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A Comparison of Methods of Measuring Microscopical Particles.

By Harold Heywood, Ph.D., M.Sc., A.C.G.I., M.I.Mech.E.*

The measurement of irregularly-shaped particles by means of the microscope is not an easy process, nor is it likely to be accurate if there is an extensive size range, unless a very large number of measurements is made. The present tendency is to use one of the various sedimentation procedures for determining the size distribution of sub-sieve particles, but there are certain circumstances in which only microscopic measurement can be applied. Such cases occur either because of some peculiar characteristic of the particles which prohibits satisfactory dispersion in a liquid, or because only a minute sample of the dust is available. For the latter reason measurements of atmospheric or industrial dusts extracted from the air by means of a suitable collector often have to be made directly by means of the microscope. On the contrary, samples consisting of graded particles—i.e., with a restricted size-range—may be measured comparatively easily and accurately by means of the microscope.

The particle size distribution of microscopical dust particles is frequently determined by means of the Patterson-Cawood graticule, placed in the eyepiece of the microscope, and consisting of a series of circles of various diameters with which the particles observed are compared in size ([1])†. In the original arrangement of this graticule, shown in Fig. 1, there are two series of circles, respectively opaque and transparent, and presumably designed for comparison with particles of similar appearance. The graticule system of measurement has been extensively studied by G. L. Fairs ([2]), and certain modifications were proposed in the arrangement of the

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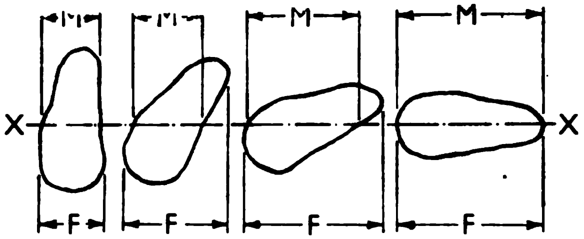
†Figures in parentheses refer to references given at the end of the paper.

which the areas had been determined by planimeter and the corresponding mean projected diameters calculated. The mean projected diameter is defined as the diameter of the circle enclosing an area equal to that of the profile of the particle when viewed normally to the position of greatest stability.

An alternative to the graticule comparison system of measurement is to adopt a single dimension which statistically represents the profile size—i.e., although the error of an individual measurement may be large, the average of a number of measurements will be truly representative. A definition of statistical diameter by Martin (5) is the length of a line intercepted by the profile boundary which approximately bisects the area of the profile and which is drawn in a random direction maintained constant for all the profile measurements. H. Green (6) adopted a similar method, measuring the horizontal intercept through the centre of the particle images, which had been photographed and projected on a screen. A



M MARTIN'S STATISTICAL DIAMETER

F FERET'S STATISTICAL DIAMETER

FIG. 3.—Definitions of statistical diameters.

different statistical diameter has been defined by Feret (7) as follows :—

The most suitable measure is the mean, for a sufficient number of particles of about the same size, of the distance between two tangents on opposite sides of the apparent outline of the particle, parallel to an arbitrary fixed direction and irrespective of the orientation of each particle with respect to this direction.

These definitions of statistical diameters are illustrated in Fig. 3, the line XX being the random direction of measurement. This diagram shows that although the statistical diameter of an individual particle of elongated shape may depart considerably from the mean projected diameter in either a positive or negative direction, the average of a large number of measurements should approximate to the mean projected diameter. It is also apparent that although the upper and lower limiting values of the statistical diameter for

TABLE I

MEASUREMENTS ON ALL PROFILES

	Observer No.	Circle No. and Diameter in Cm.									Mean in Cm.	No.		
		1	2	3	4	5	6	7	8	9		Correct	Over	Under
		2·5	2·9	3·3	3·7	4·1	4·5	4·9	5·3	5·7				
Opaque circles.........	1	0	12	46	39	32	8	2	3	0	3·683	88	9	45
	2	0	8	34	51	27	17	4	1	0	3·777	92	22	28
	3	1	11	33	36	35	15	6	3	2	3·832	102	26	14
	4	0	7	18	47	35	26	5	4	0	3·940	72	61	9
	5	0	3	20	34	36	28	10	9	2	4·100	54	87	1
	6	1	3	9	40	41	30	11	4	3	4·113	41	96	5
	7	0	2	13	38	36	33	14	6	0	4·126	37	102	3
	8	0	1	14	34	37	38	11	7	0	4·145	28	114	0
	9	0	1	14	36	39	31	13	5	3	4·148	34	108	0
	10	0	1	12	21	42	38	16	9	3	4·272	18	124	0
Transparent circles...	3	1	14	36	37	33	11	4	4	2	3·773	111	11	20
	1	0	7	35	48	28	16	6	2	0	3·805	116	14	12
	6	1	11	28	40	37	15	5	3	2	3·843	106	26	10
	2	0	7	33	42	37	13	4	5	1	3·850	117	22	3
	8	0	6	35	42	32	18	4	4	1	3·852	116	23	3
	10	0	7	28	50	32	16	5	3	1	3·852	116	23	3
	4	0	4	24	49	40	15	5	4	1	3·908	101	41	0
	7	0	2	30	46	35	19	4	5	1	3·913	99	42	1
	5	0	7	22	52	26	21	9	3	2	3·928	99	43	0
	9	0	4	19	51	34	22	7	5	0	3·959	83	59	0
Mean for opaque circles.........		0·2	4·9	21·3	37·6	36·0	26·4	9·2	5·1	1·3	4·014	56·6	74·9	10·5
Mean for transparent circles...		0·2	6·9	29·0	45·7	33·4	16·6	5·3	3·8	1·1	3·868	106·4	30·4	5·2
Mean projected diameter		0	8	34	46	37	9	4	3	1	3·796	142	0	0
Martin's statistical diameter...		3	21	35	35	28	11	4	3	2*	3·700	—	—	—
Feret's statistical diameter......		0	4	19	31	30	22	16	9	11†	4·261	—	—	—

*Includes measurements up to 6·5 cm. †Includes measurements up to 8 cm.

a given particle are approximately the same for both systems of measurement, the Feret diameter is the greater for intermediate positions of the particle with respect to the reference line.* The statistical diameters of all the sandstone profiles were measured and the results are compared later with the other systems of measurement.

Table I gives the results of profile comparisons for each observer, arranged in order of increasing mean values. Only two observers estimated below the correct average, using the opaque circles, the remainder over-estimating to an increasing degree. The results with transparent circles are also arranged in order of increasing mean values, the individual observers retaining the same reference number as in the first part of the Table. The results with open circles transposed over the profiles are clearly more accurate, though longer time is required for the comparison. Although the order of accuracy of the observers differs in the two sets of measurements, in general the same observers attained the highest degree of accuracy in both cases. The last three columns in Table I show the number of correct estimates and of those over and under-estimated; it is important to consider these in conjunction with the mean values, since a number of over and under-estimates in equal proportions would still give a correct mean value.

In order to analyse the effect of particle shape the profiles were sub-divided into groups with restricted limits of the ratio of length to breadth, this factor being termed the elongation ratio—symbol n. The limiting values of the elongation ratio for the five groups thus formed are stated in Table II, which gives a summary of the observations. This Table shows for each elongation group the number of profiles corresponding to each circle number averaged from the ten sets of observations; also the calculated mean diameters and the number of correctly, over, and under-estimated profiles. The meaured mean projected diameters and the statistical measurements are also recorded in this Table. The data is summarised in a form convenient for comparison in Table III, which shows the number of correctly, over, and under-estimated profiles on a percentage basis, and the mean diameters as determined by graticule comparison, by measurement of profile area and by statistical measurement. This Table also shows the percentage error based

*With due respect to the originators of the various mean diameters the author considers that it would be more satisfactory to designate these by a descriptive name. Thus the diameter referred to in the paper as Martin's statistical diameter could be termed the 'statistical intercept diameter', and Feret's statistical diameter could be termed the 'statistical tangential diameter'.

TABLE II

AVERAGES OF MEASUREMENTS FOR VARIOUS ELONGATION RATIOS

Range of Elongation Ratio	*Method of Measurement*	*Circle No.*									*Mean in Cm.*	*No.*		
		1	2	3	4	5	6	7	8	9		*Correct*	*Over*	*Under*
1·00 to 1·20	Opaque circle comparison......	0·2	2·9	9·1	12·4	7·3	2·0	0·1	—	—	3·655	15·8	16·2	2·0
	Transparent circle comparison	0·2	4·1	12·4	13·3	3·7	0·3	—	—	—	3·502	27·1	5·0	1·9
	Mean projected diameter......	0	4	15	12	3	0	—	—	—	3·465	34	0	0
	Martin's statistical diameter	1	5	14	9	5	0	—	—	—	3·442	—	—	—
	Feret's statistical diameter ...	0	2	5	15	9	2	1	—	—	3·783	—	—	—
1·21 to 1·40	Opaque circle comparison......	0	1·6	7·8	11·8	9·9	6·4	0·5	—	—	3·841	15·7	19·7	2·6
	Transparent circle comparison	0	2·2	10·6	14·5	8·8	1·8	0·1	—	—	3·676	30·9	5·4	1·7
	Mean projected diameter......	0	3	11	14	9	1	0	—	—	3·640	38	0	0
	Martin's statistical diameter	1	7	8	10	9	3	0	—	—	3·594	—	—	—
	Feret's statistical diameter...	0	2	7	8	11	7	3	—	—	3·961	—	—	—
1·41 to 1·70	Opaque circle comparison......	0	0·4	3·4	9·7	12·1	10·3	3·3	0·8	—	4·116	13·8	23·5	2·7
	Transparent circle comparison	0	0·6	5·1	12·4	13·7	6·7	1·5	0	—	3·953	28·7	10·7	0·6
	Mean projected diameter......	0	1	7	12	17	2	1	0	—	3·850	40	0	0
	Martin's statistical diameter	1	4	8	9	10	5	3	0	—	3·800	—	—	—
	Feret's statistical diameter...	0	0	5	6	7	9	7	4	2	4·370	—	—	—
1·71 to 2·00	Opaque circle comparison......	—	—	1·0	2·8	4·7	5·5	3·7	2·1	0·2	4·404	7·1	10·7	2·2
	Transparent circle comparison	—	—	0·9	4·7	5·1	5·7	1·8	1·8	0	4·264	14·7	4·7	0·6
	Mean projected diameter......	—	—	1	6	5	5	2	1	0	4·180	20	0	0
	Martin's statistical diameter	—	3	4	5	2	2	1	2	1*	3·960	—	—	—
	Feret's statistical diameter ...	—	0	2	2	2	4	3	3	4†	4·810	—	—	—
2·01 to 2·75	Opaque circle comparison......	—	—	0	0·9	2·0	2·2	1·6	2·2	1·1	4·720	4·2	4·8	1·0
	Transparent circle comparison	—	—	0	0·8	2·1	2·1	1·9	2·0	1·1	4·720	5·0	4·6	0·4
	Mean projected diameter......	—	—	0	2	3	1	1	2	1	4·540	10	0	0
	Martin's statistical diameter	—	2	1	2	2	1	0	1	1*	4·060	—	—	—
	Feret's statistical diameter ...	—	0	0	0	1	0	2	2	5‡	5·450	—	—	—

*Includes measurements up to 6·5 cm. †Includes measurements up to 7 cm. ‡Includes measurements up to 8 cm.

on the mean projected diameter for each elongation group and for the total number of profiles.

TABLE III

SUMMARY OF MEASUREMENTS

Range of Elongation Ratios	1·00 *to* 1·20	1·21 *to* 1·40	1·41 *to* 1·70	1·71 *to* 2·00	2·01 *to* 2·75	*All Profiles*
Opaque circle comparisons, per cent :						
Correct	46·5	41·3	34·5	35·5	42·0	39·9
Over-estimated	47·6	51·9	58·8	53·5	48·0	52·7
Under-estimated	5·9	6·8	6·7	11·0	10·0	7·4
Transparent circle comparisons, per cent :						
Correct	79·7	81·3	71·8	73·5	50·0	74·9
Over-estimated	14·7	14·2	26·7	23·5	46·0	21·4
Under-estimated	5·6	4·5	1·5	3·0	4·0	3·7
Mean sizes in cm. :						
Opaque circle comparisons...	3·655	3·841	4·116	4·404	4·720	4·014
Transparent circle comparisons	3·502	3·676	3·953	4·264	4·720	3·868
Mean projected diameter ...	3·465	3·640	3·850	4·180	4·540	3·796
Martin's statistical diameter	3·442	3·594	3·800	3·960	4·060	3·700
Feret's statistical diameter...	3·783	3·961	4·370	4·810	5·450	4·261
Error of the means relative to the mean projected diameter, expressed as per cent :						
Opaque circle comparisons...	+ 5·5	+ 5·5	+ 6·9	+ 5·4	+ 4·0	+ 5·7
Transparent circle comparisons	+ 1·1	+ 1·0	+ 2·7	+ 2·0	+ 4·0	+ 1·9
Martin's statistical diameter	— 0·7	— 1·2	— 1·3	— 5·3	—10·6	— 2·5
Feret's statistical diameter...	+ 9·2	+ 8·8	+13·5	+15·1	+20·0	+12·3

Considering the results of comparisons with opaque circles, there does not appear to be any significant variation of the error according to elongation; indeed, the results for the last groups are more accurate than the mean for all the profiles. This may, however, be accidental, owing to the small number of observations in this group,

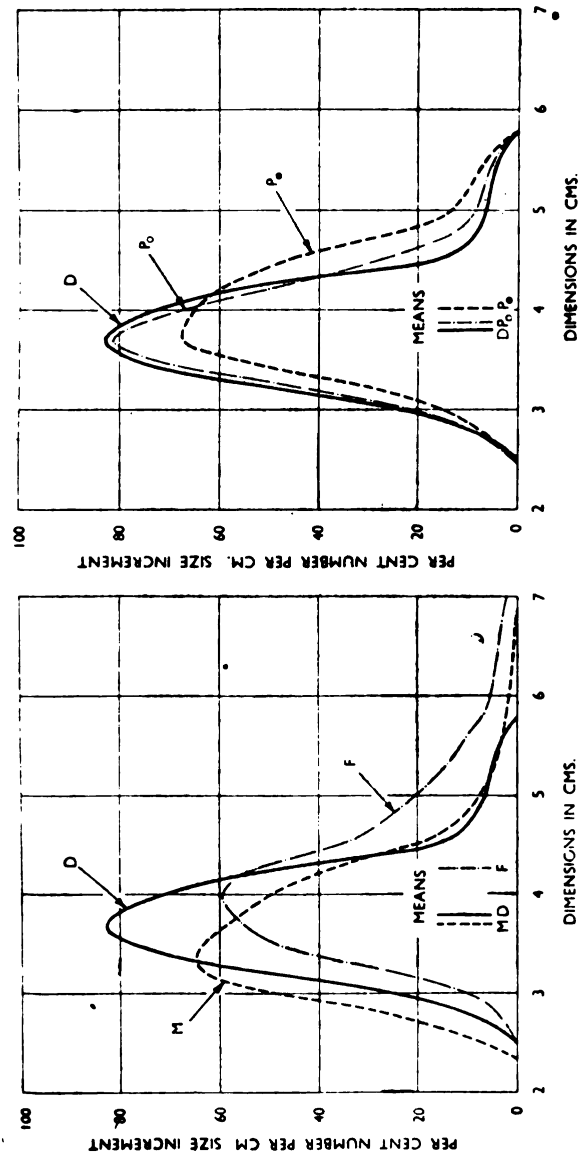


FIG. 4.—Frequency curves of particle measurements.

D Mean projected diameter.

M Martin's statistical diameter.

F Feret's statistical diameter.

$P_\bullet$ Comparison method using opaque circles.

P_0 Comparison method using transparent circles.

major axis than to the minor axis, and proportionately spaced at intermediate positions.

An arrangement of radii spaced in this manner is shown by the right-hand side of Fig. 5, and the calculated values in the final column of Table IV show that the average gives the correct mean radius of the ellipse. In practice it would be impossible to arrange such a system of spacing, nor is it practical to calculate the root mean square value of the observations ; indeed it would be incorrect to apply this method to any except a system of identical profiles.

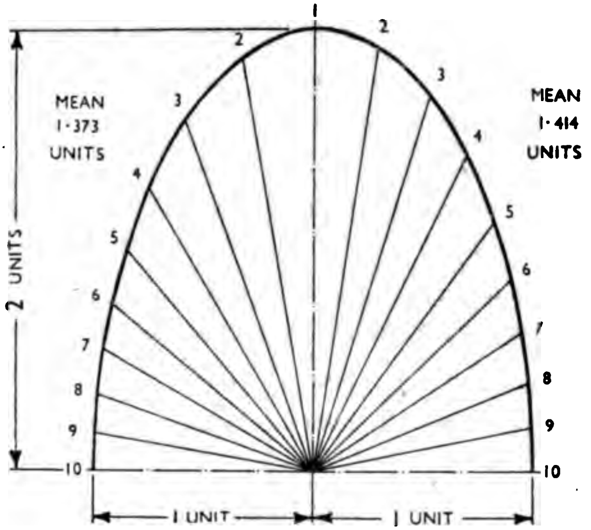


FIG. 5.—Mean radius of an ellipse.

(Left-hand side, uniform angular spacing. Right-hand side, spacing closer adjacent to semi-major axis.)

The following equations have been deduced for the values of Martin's and Feret's statistical diameters, in terms of the breadth or minor axis of the ellipse, B, and the ratio of length to breadth, or elongation n.

Martin's statistical diameter =

$$B\left\{1-\left(\frac{n^2-1}{n^2}\right)\cos^2\theta\right\}^{-\frac{1}{2}} \quad \ldots \quad Eq.\ 1$$

Feret's statistical diameter=

$$B\left\{1+(n^2-1)\cos^2\theta\right\}^{\frac{1}{2}} \quad \ldots \quad Eq.\ 2$$

In Equation (1) θ is the angle between the random line of intersection and the major axis of the ellipse ; in Equation (2), θ is the angle between the perpendicular joining the tangents to the profile and the major axis of the ellipse. The integration of these equations between the limits O and $\pi/2$ is divided by $\pi/2$ to obtain the mean values of the respective statistical diameters. Integration of these expressions involves elliptic functions, but a solution may be obtained by expanding the expressions into a series of converging terms. The mean projected diameter of an ellipse is equal to $\sqrt{n}$ times B and the figures in Table V show the calculated errors of the statistical diameters for ellipses having a ratio of length to breadth up to 10.

TABLE V

ERRORS OF STATISTICAL DIAMETERS FOR ELLIPSES

Value of n	*Error of Statistical Diameter as Percentage of Mean Projected Diameter*	
	Martin	*Feret*
1	0	0
1·5*	— 1·01	+ 3·10
2	— 2·83	+ 9·83
3	— 7·04	+ 22·8
4	—10·8	+ 36·5
10	– 25·7	+ 104·5

Thus it is shown that the errors of Martin's statistical diameter are negligible for normal mineral particles, for which n has an average value of about 1.5 and rarely exceeds 2.5, but the error of Feret's diameter is much greater. Neither of these statistical diameters is suitable for use with greatly elongated or acicular particles, which are probably best measured in terms of the breadth or length.

SUMMARY AND CONCLUSIONS

The Patterson-Cawood or other graticule systems of particle-size measurement and the use of Martin's statistical diameter give results that are within the general limits of practical accuracy. Martin's statistical diameter is approximately one per cent below

[redacted]

Subject to revision.] [*Paper issued on March 14th, 1946, 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, on Thursday, March 21st, 1946, at* **5.30 o'clock p.m.**

Application of Sizing Analysis to Mill Practice.

By H. Heywood, Ph.D., M.Sc., A.C.G.I., M.I. Mech. E.,*
and E. J. Pryor, *Member.*

Introduction

For the purpose of control in mineral dressing particle size of ground material is commonly reported, or considered, in terms of a limiting and a retaining percentage—such as, 5 per cent + 65 mesh, 80 per cent—200 mesh. This provides a simple working instruction in connection with the setting of the classifiers used in the closed circuit of a wet-grinding ball-mill. It must, however, be recognized as merely an extreme simplification in terms of operational control, which serves as a rough indicator only. In the great majority of modern mills froth flotation plays an important part and this entails physical and chemical control of the solid surfaces and volumes present in the mill pulp. This is one important reason why the trend in current practice is toward a greater precision in the definition of particle sizes and the investigation of their distribution far below 'meshes' which can be arrested on even the finest screens. The term 'mesh', even when the screening system in use is specified and a consistent method of screening is carefully applied, has certain shortcomings. Used without qualification it gives the controlling technician too little information concerning the shape of the particles undergoing treatment and, therefore, inadequate information with regard to the total surface area of a given weight of solids. Whether the concentrating plant employs gravity separation, flotation, or a chemical method the surface exposed is of considerable technical importance.

When screening is used in the circuit physical constraint is applied to the feed under 'stop-go' gauging conditions determined

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by the geometrical shape and area of the retaining apertures and by the manner in which the crimped boundaries of these apertures react toward the material presented to them. In this reaction the shape of the particle is also one of the factors determining screen efficiency. A commercial screen handling heavy tonnage usually slopes downward and arrangements should be made to distribute the incoming feed over the width of the screen at its head end. After this each particle has a series of opportunities to present its minimum cross-section normally to rectangular openings bounded by vibrating wires curved more or less severely in the vertical plane. A particle tends to slide on its broadest area, rather than to present its minimum cross-section to the mesh opening. The crimp of the wires and the applied vibratory movement of the screen-cloth both upset this attempt to travel in the most-stable position, but are hindered by two other factors which militate against good screening—that is, stratification of the feed and departure in particle-shape from cubic toward either the acicular or tabular. The nearer the cross-sectional area of a particle approaches to that of a screen opening the more markedly does divergence of its shape from spherical or cubic reduce screening efficiency.

Laboratory screening is performed in a manner which is by no means reproduced in the mill and it is important to appreciate the differences. Commercial screens must withstand rough usage and must handle run-of-mine tonnages. Laboratory screening is solely concerned to obtain accurate information simply and expeditiously. It is used to test and control fluid suspensions from the wet-grinding plant more often than to deal with the conditions in the dry-crushing plant. This paper is concerned with laboratory screening and the position revealed by sizing analysis. The authors have collected in Table I (Plate I) some fundamental data relating to these matters. Its columns can be used to aid the interpretation of control tests and to facilitate comparison between the principal screening systems in laboratory use today. In order to save space, and for convenience of reference, an explanation of the numbered columns in Table I has been set out in Table II (Plate I also).

Average Particle Shape

Surface areas of crushed mineral particles are usually calculated from an assumed shape—e.g., that the particles concerned are cubes or spheres—although such assumptions are clearly far removed from ordinary working conditions. The question then arises whether or not an average particle shape exists. Observation of any graded fraction of crushed mineral particles shows a great variety of shapes, ranging from elongated particles to thin flakes, with a proportion of approximately equi-dimensional grains. When a piece of rock approximately cubic in shape is fractured, a proportion of the resulting fragments will probably be elongated or flaky. When such shapes are further broken down by crushing or grinding there is a tendency for the newly-formed particles to revert to a more nearly equi-dimensional shape and thus a state of equilibrium is gradually attained in which the mean particle shape becomes approximately constant, in spite of great individual variation. As a result of many shape measurements on particles of crushed siliceous rocks, limestones, etc., one of the authors has found a remarkable agreement in the mean shape of the finely-crushed material, although exceptional minerals with marked cleavage—such as, mica or schistose rock—have a characteristic shape of particle which differs from the mean for other rocks.

It is therefore permissible to assume that many mineral particles may be represented by a mean shape and although an error will occur in some cases this will in general be considerably less than if the particles are assumed to be spherical or cubic. The proportions of this mean particle are such that the ratio of length to breadth is from 1.2 to 1.5 and the ratio of breadth to thickness is 1.6 to 1.8. The shape approximates to a truncated tetrahedron, and the surface in sq. cms. per gram at unity specific gravity is 82/A, where A is the aperture in mm. of the sieve through which the particle will just pass. The figures in columns 11 and 15 in Table I have been calculated using this relationship.

The photographs in Fig. 1 demonstrate that the impression of shape obtained by microscopical observation of particles lying in the most stable position may be misleading. This figure shows plan and elevation views of particles produced by the compression to fracture of cubical blocks of shale, the upper set of profiles being of particles produced by compressing the cube in a direction parallel to the bedding plane, and the lower set by compression perpendicular to the bedding plane. These profiles clearly illustrate that the plan view of a particle is no guide to its thickness, and that the breaking of stratified rocks is influenced by the direction of the crushing forces.

Surface and Energy

Column 16 of Table I shows the surface area per gram of material expressed relatively to the value for the 150 to 200-mesh screened fraction. On the assumption that Rittinger's Law holds, these figures represent the relative amounts of energy required for the production of any size of particle. Actual power consumption always exceeds this considerably, the losses being in :—

(*a*) Heat used in warming the circulating system.

(*b*) Mechanical loss in the machinery.

(*c*) The overgrinding inevitable when bringing most or all of the product to a given maximum mesh.

(*d*) Unrecorded development of surface by internal cracks, compaction into aggregates, and adhesion.

Table I presents the facts in a manner which may be used for immediate mill calculations. Although they state nothing new, they focus attention on the specific results of power application in the production of fresh surface and show clearly what proportion of that power has been usefully expended and what has been

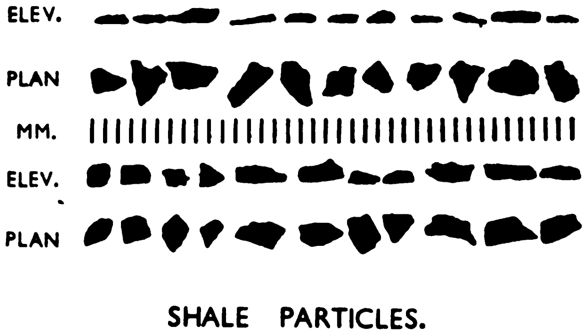

FIG. 1.—Profiles of shale particles.
Upper series produced by compression parallel to bedding plane.
Lower series produced by compression perpendicular to bedding plane.

wasted in unwanted, though today unavoidable, overgrinding. In Table III some sizing analyses have been computed in terms of percentage surface represented by each screen fraction. They show what percentage of the grinding effort has been applied to the correct end-point, and what percentage has for all practical purposes been wasted, It is frequently observed that the ball-mill works with a very low efficiency. The authors consider that a correct approach to the problem of efficient wet-grinding must take into account the whole new surface developed, and that the recording of percentage weight in the screenable fractions is no longer adequate. The major part of the grinding effort is wasted, and will continue to be wasted until we are awake to the grinding losses made in the sub-sieve sizes.

TABLE III

DISTRIBUTION OF WEIGHT AND SURFACE IN GROUND SILICA

Size	*Medium Grade*		*Fine Grade*	
	Per cent Weight in Undersize	*Per cent Surface in Undersize*	*Per cent Weight in Undersize*	*Per cent Surface in Undersize*
100-mesh B.S.......	97·8	99·8	99·9	100·0
150 " "	90·0	98·7	—	—
200 " "	75·5	95·6	99·7	100·0
240 " "	69·2	93·9	—	—
300 " "	60·3	91·0	99·4	100·0
40 microns	49·0	86·5	94·4	99·8
30 "	37·6	80·3	87·1	99·3
20 "	24·8	70·5	74·5	98·2
10 "	10·8	52·8	50·2	94·8
5 "	4·3	36·3	32·3	89·8
2 "	1·0	18·5	16·0	79·8
1 "	0·25	8·9	9·1	70·1
0·5 "	0·05	3·8	4·9	58·2
0·2 "	—	—	1·7	38·9
0·1 "	—	—	0·6	23·4
0·05 "	—	—	0·15	10·6

Note.—Percentage weight of undersize measured down to 2 microns size; subsequent percentages obtained by extrapolation of curve.

Many improvements have been made in grinding technique during the past few years, but much fundamental research is still needed. Ball-ratio, the closed circuit, control of classifier and mill density, sizing of new feed, provision of spherical balls and well-designed liners, speed of mill, and dwelling-time in the mill—all these are efficiency-promoting factors now understood and widely applied. They have greatly improved grinding efficiency, but they lack the hall-mark of the fundamental approach to the study of grinding. Though they enable the operator to increase the capacity of his mill and reduce his grinding cost, they fail to answer the question of the physical nature of disruption of a particle of ore by an applied grinding force. Until this question is solved, it is not possible to know to what extent over-grinding is inevitable or what is the ideal mode of application of grinding force—questions which profoundly affect grinding economics.

Losses due to over-grinding are not necessarily confined to the mill. Slimes sent to the concentrator may be too fine to concentrate, settle, or filter ; they may form hydrophilic coatings on otherwise floatable heavy sulphides ; their first cost is not necessarily their last cost.

Sizing Methods

One criterion of experimental work is its reproducibility. Screening, followed by estimation of sizes in the sub-sieve fraction, is a specialized process. The techniques chiefly used are :—

Screening Wet-and-Dry.—The limitations of laboratory screening arise chiefly in regard to particle shape and the difficulty met with when particles are nearly of retaining mesh size. Since screen cloth is woven with round wires near-mesh particles must be repeatedly presented to ensure that they neither rebound nor fail to drop with their true mean dimension centrally presented. Too long a screening-time may cause some degradation and so an arbitrary time-limit is used.

The effects of flocculation, electrostatic aggregation, adherence of slime, moisture, or dried-on soluble salts are minimized by wet-screening with a suitable dispersant before shaking in a nest of screens. Methods are given by Work (1)* and hints on sieving and size analysis by Dasher(2). A suitable quantity of sample should be used. For close work no fraction at completion of the sieving run should be of such a weight that it lies more than one to two particles deep over the effective sieve area. For routine control this could be increased to as much as four particles deep. With some mechanical shaking systems the whole sieve area is not used, material under test becoming ' bunched ' in accordance with the applied harmonics. Samples should be adjusted accordingly.

The figures in Col. 17 (Table I) give the approximate weights at which materials of unit specific gravity fully cover a round screen 8 in. in diameter one particle deep.

Equivalent Sieves.—Fine particles are sometimes designated by reference to an equivalent sieve mesh even though such a sieve is far beyond any possibility of manufacture and its existence is

*Figures in parentheses refer to references given at the end of the paper.

purely fictitious. There are objections to this practice and it would be preferable to train operators to think in terms of particle dimensions in microns, but in cases where this system is thought to have advantages an improvement would be effected by a recognized scheme of correlation.

Considering sieves of 150-mesh and finer, the product of meshes per inch and aperture in microns was found to average 15,500 for B.S. sieves and 14,900 for Tyler sieves. Table IV gives the apertures of hypothetical sieves if constructed to have the same proportions as the two sieve series named, and in the final column the suggested equivalent apertures are given.

TABLE IV

EQUIVALENT SIEVES AND APERTURES

Hypothetical Mesh per Inch	*Apertures in Microns*		
	B.S. Proportions	*Tyler Proportions*	*Proposed Equivalent*
500	31	30	30
600	26	25	25
800	19·4	18·6	20
1,000	15·5	14·9	15
1,500	10·3	9·9	10
2,000	7·7	7·4	7·5
2,500	6·2	6·0	6
3,000	5·2	5·0	5
4,000	3·9	3·7	4
5,000	3·1	3·0	3

If such a system is used it should be made explicit that the meshes quoted are hypothetical and only equivalent, otherwise non-technical persons may be misled into thinking that such sieves actually exist.

SUB-SIEVE PARTICLE SIZE AND SURFACE MEASUREMENT

While the importance of sub-sieve particle size analysis is stressed it is not within the scope of this paper to describe in detail the many experimental procedures available. All methods of sub-sieve analysis are based on the rate of fall of a particle in a fluid, as deduced from the well-known Stokes equation. The actual determination of the relative weights of the various sizes may be by elutriation with water or air or by the rate of sedimentation in water or some other more suitable liquid. The amount of settled or unsettled material may be determined gravimetrically, using either the sedimentation balance or the Andreasen pipette method, but photo-electric methods of measuring the surface area of particles still in suspension after various times of settlement are increasingly employed. In an apparatus of this type, called the turbidimeter in America, a narrow beam of light is directed through a dilute suspension of the particles in a liquid and then caused to impinge upon a photo-electric cell. The obstruction offered to the light-beam by the suspension depends on the amount of material present at the level of the light beam and the size of the particles concerned. From a series of readings of the output of the photo-electric cell made during the course of settlement the size distribution of the particles can be calculated([3]).

The specific surface, or surface area per unit weight, of particles dispersed in a liquid can also be calculated directly from readings of the photo-electric cell if the length of the light path through the suspension and the weight concentration of material per unit volume of suspension are known. In this case the particles are maintained in turbulent suspension, either by stirring or by pumping a flow of suspension round a circuit([4]). There are other methods of determining specific surface directly—such as, by rate of solution([5]) ([6]), by adsorption to the solid surface of a mono-molecular layer of gas([7]) ([8]) or dye ([9]), or by measuring the permeability to fluid flow of a bed of the powdered material([10]) ([11]).

Practical Application

In the mill's laboratory controls are needed which give consistency of production from day to day as the short-term goal. A master set of screens should be kept for reference, and used from time to time to check those in regular use for distortion, wear, and perforation of screen-cloth. The weakest link in the sampling chain is the human element at the reduction of impartially-cut automatic samples to laboratory quantities. Hand-work should be reduced to a minimum, made easy, and checked periodically. Otherwise the person responsible for it may take short cuts which invalidate accuracy. The sizing analysis should not be considered as ending with the figure of, say, *minus* 200-mesh, 87 per cent, or something of that sort. The sub-sieve composition of this *minus* 200-mesh material is capable of providing an important controlling factor, both in relation to grinding cost and concentrator efficiency.

The significance of surface has already been shown in Table III. These figures show that in the coarser grade the 10.8 per cent by weight that is smaller than 10 microns in size represents more than 50 per cent of the total surface and even the 1 per cent by weight below two microns represents nearly 20 per cent of the total surface. The finer grade shows still greater disparity in the relationship between weight and surface, for an extrapolation of the grading curve shows that particles smaller than 0.1 micron, although totalling less than 1 per cent by weight represent nearly a quarter of the total surface.

Hence it is clear that for a study of the surface distribution of even moderately finely-ground minerals, the size measurement must be continued to a value not greater than two microns, which is about the limit for convenient experimental technique. In the case of very fine powders even to approach an exact knowledge of the surface distribution demands an extension of particle size measurements into regions that are at present unattainable, but which offer a fruitful field for research.

The ordinary method of recording and presenting sizing analyses thus fails to bring out the vitally important statistical position of surface area in relation to grinding effort, although this area is a dominant factor in both the economics and the technology of comminution, classification, flotation, hydro-metallurgy, and handling of pulps.

Another useful possibility in sizing analysis arises when work is being done upon ores carrying components of different densities. In the closed-circuiting classifier the denser mineral is retained in the grinding system after the lighter one of similar ' mesh ' has overflowed. If the dense fraction is the valuable ore-component it determines the correct overflow conditions at which classification should be maintained. If the valuable mineral is a heavy-metal sulphide more friable than the gangue-stuff efficient milling calls for discrimination with regard to release from the closed circuit. Sizing analyses alone do not present a complete picture under these circumstances. If the density of the two ore constituents permits they can be separated in a suitable heavy liquid, thus supplementing chemical analysis by giving a physical sample for micro-examination. Valuable data can then be obtained concerning the associations, liberation, and middlings in the screen fractions. The technique of such sink-and-float tests is simple and can be expedited by centrifuging slow-settling material. Separation of 'tops' from 'bottoms' can be neatly achieved by freezing[12] the contents of the tube. This control helps the operator to increase grinding capacity and at the same time reduce over-grind, by allowing him to increase the coarseness of the classifier overflow while maintaining a visual check on the 'values' in the ore.

The authors have prepared this paper because they believe that fundamental research upon the sub-sieve sizes of ores, combined with an increasingly high standard of laboratory control and interpretation in the sieving range will between them throw much-needed light on the physics of comminution, and bring the science of mineral dressing a stage nearer the point where every ore-body ceases to be treated as a law to itself, and falls into a category in accordance with its grinding-response. While the constituency of a mill-pulp remains largely unknown as regards its fractionation an important variable will continue to escape that factual control which alone can replace the art of ore-dressing by the predictable science underlying any technical process.

References

(1) Work, L. T. 'Methods of Particle Size Determination'. *Proc.* Amer. Soc. Testing Materials, 1928, Vol. 28, Part 2, p. 771. *Also* A.S.T.M. 1941 Symposium, ref. (7) below, p. 1.

(2) Dasher, J. 'Directions for Laboratory Mineral Sizing'. U.S. Bur. Mines I.C. 7224: 1942.

(3) 'Report of Discussions on Determination of Particle Size in Sub-Sieve Range'. The Brit. Colliery Owners Res. Assn. and the Brit. Coal Utilisation Res. Assn., 1942.

(4) Bell, J. W. 'A Method for the Measurement of Surface of Finely-Divided Material'. *Trans.* Can. I.M.M., 1944, Vol. 47, p. 424.

(5) Martin, G., Bowes, E. A., and Christelow, J. W. 'A Method of Accurately Determining Experimentally the Surface of Crushed Sand Particles'. *Trans.* Brit. Ceram. Soc., 1925–6, Vol. 25, p. 51.

(6) Gross, J., and Zimmerley, S. R. 'Surface Measurement of Quartz Particles'. *Trans.* A.I.M.E., 1930, Vol. 87, p. 7.

(7) Emmett, P. H. 'Symposium on New Methods for Particle Size Determination in the Sub-Sieve Range'. Amer. Soc. Testing Materials, 1941, p. 95.

(8) Gaudin, A. M., and Hukki, R. T. 'Principles of Comminution, Size, and Surface Distribution'. A.I.M.E. T.P. 1779, 1944.

(9) Ewing, W. W. A.S.T.M. 1941 Symposium, ref. (7) above, p. 107.

(10) Carman, P. C. 'The Determination of the Specific Surface of Powders'. *Journ.* Soc. Chem. Ind., 1938, Vol. 57, p. 225. *Also* A.S.T.M. 1941 Symposium, ref. (7) above, p. 24.

(11) Rigden, P. J. 'The Specific Surface of Powders: A Modification of the Air-permeability Method for Rapid Routine Testing'. *Journ.* Soc. Chem. Ind., 1943, Vol. 62, p. 1.

(12) MacKay, R. A. *Journ.* Chem. Met. Min. Soc. S.Af., May–June, 1944.

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Plate I.

Table II
Explanation of Columns in Table I

Column No.	*Description and Explanation*
(1)	Tyler $\sqrt{2}$ Sieve Series, apertures in inches or mesh.
(2)	Nearest corresponding B.S.I. (British Standards Institution) Perforated Plate or Fine Mesh Test Sieve, and aperture in mm.
(3)	Nearest corresponding I.M.M. (Institution of Mining and Metallurgy) Test Sieve, and aperture in mm.
(4)	Distance between wires or aperture dimension of Tyler $\sqrt{2}$ Sieve Series. (Note : Figures in all subsequent columns are based on the Tyler Series.)
(5)	Area of sieve aperture in square millimetres. Defines the smallest cross sectional area of the particle.
(6)	Diagonal of sieve aperture in mm. Defines the upper limit of the breadth or intermediate particle dimension for a tabular particle.
(7)	Arithmetic mean of the aperture of the sieve in the same line and of the next sieve immediately above. This average is expressed by the symbol A and is in mm.
(8)	Surface area in square cms. per gram of cubical particles of which the side dimension is A mm., specific gravity assumed to be unity. These figures equal 60/A. (Note : **Divide** the figures given by the actual specific gravity of the mineral concerned ; this applies to columns (8) to (11) inclusive.)
(9)	Surface as above for particles having dimensions A by A by 2A mm. long. Figures equal 50/A.
(10)	Surface as above for flat particles A by A by A/10 mm. thick. Figures equal 240/A.
(11)	Surface as above for particles of average shape. (See text for explanation.) Figures equal 82/A.
(12)	Weight of material in grams to expose surface of one square metre, if particles are cubes of side dimension A mm. and specific gravity is unity. This column equals 10,000 divided by column (8). (Note : **Multiply** the figures given by the actual specific gravity of the mineral concerned ; this applies to columns (12) to (15) inclusive.)
(13)	Weight as above for particles having dimensions A by A by 2A mm. long. Figures equal 10,000/column (9).
(14)	Weight as above for flat particles A by A by A/10 mm. thick. Figures equal 10,000/column (10).
(15)	Weight as above for particles of average shape. Figures equal 10,000/column (11).
(16)	Surface areas expressed relative to the surface area of material of the same shaped particles retained between 150- and 200-mesh Tyler. These figures are equal to 0·089 divided by the respective values of A from column (7).
(17)	The approximate weight of material that will cover an 8-in. diameter sieve to a uniform depth of one particle ; the voidage between particles has been assumed to be 35 per cent and the specific gravity unity. These figures are equal to 21·1 multiplied by A in mm., and must be multiplied by the specific gravity of the mineral concerned.

Subject to revision.] [*A Paper issued on March* 14*th*, 1946, *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, on Thursday, March* 21*st*, 1946, **at 5.30 o'clock p.m.**

Crushing and Grinding Efficiencies.*

By T. K. Prentice, *Member.*

Summary

In this paper evidence will be submitted which supports the increasingly growing belief that the energy absorbed in crushing and grinding is directly proportional to the new surface area produced in the material being comminuted.

It will also be indicated that the efficiency of crushing and grinding machines ranges from about 40 per cent to 20 per cent, depending upon the type of machine and the nature of the work it is called upon to perform.

It will be shown that, of the total power input to a machine, only from 55 per cent to 90 per cent is available to achieve comminution, the balance being dissipated in the prime mover and in transmission and friction losses, etc.

It will also be shown that over 80 per cent of the energy available to achieve comminution appears in the form of heat, which is dissipated in the surrounding bodies.

The fact that deformation of rock under compression or tension up to the point of fracture is approximately proportional to the volume of the rock does not easily conform with the finding that the energy absorbed in achieving comminution is directly proportional to new surface produced and, for this reason, all the data in connection with the experimental tests are submitted in some detail. The author may have erred in some of his deductions, but it is hoped that the presentation of the detailed data might assist in the elucidation of the complex theoretical and practical problems associated with crushing and grinding.

Introduction

The primary function of crushing and grinding, or comminution, is to reduce material from its existing to lesser sizes.

There is no clear line of demarcation between crushing and grinding, but, generally, crushers are used to reduce the material to approximately quarter-inch size and grinding is applied for the reduction of the quarter-inch material to the required finished size.

* Paper presented for simultaneous publication by the Chemical, Metallurgical and Mining Society of South Africa and the Institution.

Comminution is essentially a mechanical process and crushing and grinding efficiency may therefore be expressed as the ratio of useful work performed by a machine to the total work applied. This efficiency is neatly described by Wilson (1)* as the ratio of actual to perfect accomplishment. From the scientific viewpoint, the mechanical equivalent of the actual comminution effected should be expressed as a percentage of the energy available for comminution, but from the practical point of view, it should be expressed as a percentage of the total power input to operate the machine unit.

When comparing machines of similar type, it is not uncommon to measure their performance in terms of the power consumed per unit of the desired product.

In crushing and grinding, it is comparatively simple to measure the power input to the machines but the determination of the theoretical energy required to effect size reduction and the measurement of the degree of comminution achieved present major difficulties. In fact, the variability of the hypotheses and assumptions which have been applied in the calculation of power required to effect comminution and for the measurement of the resulting products has been such that different investigators have reported astonishingly divergent crushing and grinding efficiencies.

The late Dr. H. A. White (2) deduced that the efficiency of a tube-mill was only 0·015 per cent and implied that an ideal tube-mill of an internal diameter of 2·4 in. and a length of 9·6 in. would do the same work as a tube-mill 5 ft. 6 in. in diameter and 22 ft. long. He further deduced that, whereas the 5 ft. 6 in. by 22 ft. tube-mill, driven by a 175 h.p. motor, produces about 120 tons of *minus* 200 material per 24 hours, an ideal mill of the same size should only require a 20 h.p. motor to produce 10,000 tons of *minus* 200 material per 24 hours.

Martin (3) calculated the efficiency in ball-milling as 0·06 per cent and Fahrenwald (4) has reported cylindrical mill efficiencies ranging from 6 per cent to 25 per cent. Gross and Zimmerley (5) found the efficiency of cylindrical mills in an industrial plant to average 45 per cent.

In view of the foregoing and other divergent findings, it seems desirable that an analysis should be made of the theoretical power required for comminution and of the methods available for measuring the degree of fineness achieved. In this way, relevant laws might be established whereby improved crushing and grinding

*Figures in parentheses relate to the references given at the end of the paper.

efficiency might be brought within the sphere of practical achievement. Moreover, the suggestion that the efficiency of cylindrical mills is of the order of less than one-tenth of 1 per cent is surely a challenge to the ingenuity and skill of all who are concerned with grinding problems.

Determination of Particle Size and Surface Area

To facilitate any investigation into crushing and grinding efficiencies, it is necessary to be able to measure or calculate the sizes of the original and resulting particles. For particle size down to about 50 microns (0·050 millimetres) it is standard practice to report the sizes of the comminution products in terms of a series of square-mesh wire screens, in which the diameter of the opening of each successive screen is exactly half the diameter of the opening in the preceding screen. For more exact work, intermediate screens are, of course, available. For sizes below 50 microns, screens are not entirely satisfactory and, in such cases, screen sizing may be supplemented by water or air separation or by microscopic measurement.

In metallurgical grinding circuits, the sizing sieves in general use are the 28, 48, 100 and 200 Tyler linear-mesh screens. Data relating to the graded products resulting from the use of these screens on Witwatersrand quartzite are given in Table I, on the assumption that all particles are cubes.

Table I

Witwatersrand Quartzite, Sp. Gr. 2·72

	Per Particle				*Per lb. Weight*	
Graded Product, Tyler Series	*Diameter (side of Cube), mm.*	*Volume, cu. mm.*	*Weight, milli-gram*	*Surface Area, sq. mm.*	*Number of Particles*	*Surface Area, sq. ft.*
— 28+ 48	0·416	0·071993	0·19582	1·0384	2,316,000	25·9
— 48+100	0·208	0·008998	0·02448	0·2596	18,529,000	51·8
—100+200	0·104	0·001125	0·00306	0·0649	148,232,000	103·6

Not only can surface areas be calculated from screen analysis, as indicated in Table I, but direct determinations can be made by the formation of chemical or physical coatings on the particles, by the amount of gas absorbed, by optical density measurement in dilute suspensions ([6]), by permeability methods ([7]), and by chemical dissolution methods ([8]).

In connection with the research work to be described in this paper, three methods were used at the Rand Mines Laboratory for determining the surface area of ore particles :—

(I) By comparing with a predetermined standard the amount of water adhering to individual particles after immersion in water and draining for one minute.

(II) By comparing with a predetermined standard, the amount of residual liquid retained by a given weight of particles after immersion in water or paraffin and subsequent draining for 30 minutes in a Gooch crucible, covered to avoid evaporation.

(III) By Carman's permeability method ([7]).

The first method was applied to all particles greater than ¼ in. in diameter; the second to the range of particles from 20-linear mesh to ¼-in. size and the third method to *minus* 20-linear mesh particles.

Great accuracy cannot be claimed for either the first or the second method because of the doubt about the correctness of the standards applied. For example, in Method I, the standard was obtained by determining the amount of water adhering to each of several very smooth quartzite tube-mill pebbles, free of cracks and sufficiently regular in shape to permit geometrical surface area determination. It was found that 1 cu. cm. of water covered 382 sq. cm. of pebble surface, equivalent to a water coating of 0·026 millimetres or 26 microns.

To obtain the standard for Method II, the surface area of each of ten particles, approximately 0·3 in. in diameter, was determined by Method I on the basis of the first standard and therefrom the combined surface area of the ten particles was obtained. A Gooch crucible was then filled with water up to the point required to cover all the particles used in any of the second method tests and allowed to drain. When drainage was complete, the wet crucible was weighed. The aforementioned 10 particles were then placed in the crucible, covered with water and, after being drained for 30 minutes, the crucible containing the particles and the residual moisture was weighed. The difference in these two weights was equivalent to the known dry weight of the 10 particles *plus* the weight of the residual moisture. From the weight of the residual moisture and the accepted surface area of the 10 particles, it was calculated that, under the conditions of Method II, the covering capacity of 1 cu. cm. of water was 1,180 sq. cm. In a similar manner, the covering capacity of 1 cu. cm. of paraffin was found to be 2,800 sq. cm.

The tests carried out by Method II on *minus* 8 *plus* 20-linear mesh quartzite gave abnormally high results with water due to capillary action, but satisfactory results were obtained with paraffin.

Although the accuracy of Methods I and II cannot be guaranteed, it is probable that, owing to the relatively small surface area of the larger particles per unit of weight, these methods are sufficiently reliable for all practical purposes. For example, in a mixture of particles made up with equal weights of the three sizes given in Table I, the surface area of the coarsest size would constitute only 14 per cent of the total surface area of the mixture.

Method III was carried out much on the lines described by Carman (7). It was found that if the *minus* 20-linear mesh material contained only a relatively small percentage of *minus* 200-mesh product, the apparent surface area of the mixture was higher than when calculated from the weights and surface areas of the individual fractions determined separately. In all such cases the surface area of *minus* 20-mesh material was calculated on the basis of its separate fractions.

The results obtained by these three methods on graded quartzite ranging from 3½-in. lumps down to the *minus* 100 *plus* 200-linear mesh size are given in Table II, together with the theoretically calculated areas and those obtained by John Gross (9) when using the hydrofluoric acid dissolution method (8).

Table II discloses that the measured surface area is always greater than the theoretical surface area. For sizes less than 20-linear mesh, the permeability method, as used at the Rand Mines Laboratory (Method III), gave results about 70 per cent higher than the theoretical area, whereas the hydrofluoric acid method, as used by John Gross (9), gave results 2·6 times higher than theoretical. For sizes greater than 20 mesh and less than 3 mesh, the Rand Mines Method II and the hydrofluoric acid method gave comparable results, each being about six times the calculated theoretical surface area. John Gross states that this high ratio is due mainly to cracks in the particles, the balance being due to the irregularity of the fracture surface.

If the surface area per unit weight of a product is known, it is possible to calculate the average diameter of the particles comprising the product. For example, with Witwatersrand quartzite (Sp. Gr. 2·72), the average diameter per particle, in millimetres, is a constant 10·8, divided by the theoretical surface area, in square feet, of 1 lb. by weight of the product. The theoretical surface area can be determined from the relevant ratio of the actual to theoretical surface, as disclosed in Table II. Alternatively :

$$\text{Average diameter in millimetres} = \frac{10{\cdot}8 \times \text{ratio actual to theoretical surface}}{\text{Actual surface (sq. ft. per lb.)}}$$

TABLE II

WITWATERSRAND QUARTZITE, SP. GR. 2·72

Grading of Product	*Average Diameter of Particle, mm.*	*Theoretical Surface Area, sq. ft. per lb.*	*Measured Surface Area* — *Rand Mines Laboratory* — *Method No.*	*Rand Mines Laboratory* — *Sq. ft. per lb.*	*Rand Mines Laboratory* — *Ratio, Actual to Theoretical*	*HF Method John Gross* — *Sq. ft. per lb.*	*HF Method John Gross* — *Ratio, Actual to Theoretical*
3·45 in.	87·63	0·12	I	0·45	3·75	—	—
2·79 „	70·87	0·15	„	0·64	4·27	—	—
2·17 „	55·12	0·20	„	0·80	4·00	—	—
—2 in. + 1½ in.	44·45	0·24	„	0·83	3·46	—	—
—1½ „ + 1 „	31·75	0·34	„	1·30	3·82	—	—
—1 „ + ¾ „	22·22	0·48	„	2·28	4·75	—	—
— ¾ „ + ½ „	15·87	0·68	„	3·28	4·82	—	—
— ½ „ + 0·37 in.	11·05	0·98	„	5·27	5·38	—	—
—0·37in. + 3 mesh	7·92	1·36	„	10·20	7·50	—	—
— 3 + 6 „	4·70	2·29	II	14·60	6·38	18·58	8·11
— 6 + 8 „	2·79	3·86	„	23·73	6·15	24·83	6·43
— 8 + 20 „	1·40	7·69	„	28·12	3·66	35·09	4·56
— 20 + 48 „	0·495	21·76	III	35·20	1·62	67·20	3·09
— 48 + 100 „	0·208	51·80	„	83·98	1·62	131·17	2·53
—100 + 200 „	0·104	103·58	„	188·46	1·82	231·39	2·23

Theory of Crushing and Grinding

Rock may be broken by the application of force in any of the following ways:

(*a*) Impact.
(*b*) Shear.
(*c*) Bending.
(*d*) Compression.
(*e*) Tension.

In practical crushing and grinding, all of these factors are present in varying degrees, depending mainly upon the type of machine used to effect comminution.

If compression were applied over the whole area of the top of a cube constrained in a cavity in a steel block so that no space existed between the container and the rock, virtually no fracture would result. If, however, the cube were supported at the bottom but not at the sides and pressure were applied to the top, the cube could distend laterally and, if sufficient force were applied, the cube would fracture. In this case it may be claimed that fracture is directly caused by shear resulting from compression.

When a piece of rock is impacted or compressed with insufficient force to overcome the elasticity of the rock, no useful work is accomplished and the force thus expended is probably dissipated as heat. The force required to overcome elasticity is, without a doubt, directly proportional to the volume of the rock.

When sufficient force is applied to overcome elasticity and to effect fracture of the rock, the combined force thus applied may be considered to have done useful work, because fracture cannot be achieved without first overcoming the elasticity of the rock. Although the energy required to overcome elasticity is proportional to the volume of the particle, it will be demonstrated later that, when this energy is released, it produces new area of fracture which is directly proportional to the energy absorbed. It will also be shown that a large proportion of the energy required to overcome elasticity is transformed into heat and dissipated in the surrounding bodies.

The energy required to overcome elasticity constitutes virtually the whole of the power required to achieve fracture and, if the latter could be considered apart from the energy required to overcome elasticity, it would constitute only a small percentage of the total energy absorbed. When Edser ([10]) in 1921 calculated that the energy required to produce 1·0 sq. cm. of new surface in quartz was 920 ergs, equivalent to 0·06 ft.-lb. per sq. ft. of new surface, he ignored the energy required to overcome the elasticity of the rock. Likewise Dr. White [11] calculated that the energy required might be 2,333 ergs per sq. cm. (0·16 ft.-lb. per sq. ft.), but he stated that his calculations did not allow for the energy put into increased heat motion of the quartz particles and consequently he favoured Edser's finding.

Martin [12] also ignored the energy required to overcome the elasticity of the rock when he calculated that the theoretical energy required to produce 1 sq. ft. of new surface was 0·036 ft.-lb. which is only 60 per cent of Edser's finding.

Whereas to achieve fracture, *per se*, might require as little as 0·06 ft.-lb. per sq. ft. of fracture surface, it will be demonstrated that the combined energy required to produce 1 sq. ft. of new surface is approximately 3·5 ft.-lb., so that the work of preparation, prior to fracture, would appear to constitute 3·44 ft.-lb. or 98 per cent of the total.

To facilitate an appreciation of what happens when rock is broken, let us consider a theoretical case of an inch cube of rock (length of side = 25·40 millimetres) of Sp. Gr. 2·72 and weight 0·0983 lb., being broken into perfect cubes in 13 successive stages, in each of which the length of side (herein referred to as diameter) of the cube or cubes is exactly halved. Thus, in the first stage, the inch cube is broken into eight cubes each of $\frac{1}{2}$-in. diameter and in the second stage each of the eight $\frac{1}{2}$-in. cubes is broken into eight cubes of $\frac{1}{4}$-in. diameter and so on through the 13 stages, when there would be 549,756 million cubes each of a diameter of 0·003 millimetres. Although there is no change at any time in the combined weight or volume of the cubes, the number of cubes increases in the ratio of eight to one and the volume of each resulting cube is one-eighth that of each of the preceding cubes. Relevant data are given in Table III.

Table III demonstrates that the combined surface area of the cubes doubles with each halving of the diameter.

In 1865, Rittinger propounded that the energy required for crushing is proportional to the new surface formed in the operation. Thus, since the surface formed in each stage in Table III is doubled, the energy required to produce each successive stage increases in geometric progression, the ratio being 2.

In 1885, Kick postulated that the energy required for producing analogous changes of configuration of geometrically similar bodies of equal technological state varied as the volumes or weights of those bodies. From this it is deduced that, whatever power is required to break a cube into eight smaller and equal sized cubes, it would take one-eighth of that power to break one of these smaller cubes into eight equal sized cubes, because each of the smaller cubes is one-eighth the volume of the original cube. But, since there would be eight smaller cubes to be broken, it would take the same energy to break all of them into cubes one-eighth of their size as it would take to break the original cube into eight smaller cubes. Thus, according to Kick, it would require exactly the same amount of energy to effect each of the stages of reduction shown in Table III.

TABLE III

1-IN. CUBE QUARTZITE, SP. GR. 2·72

VOLUME = 1 CU. IN. WEIGHT = 0·0983 LB.

Stage of Reduction	*Diameter per Cube, mm.*	*Combined Cubes*	
		Surface Area, sq. ft.	*New Area Produced per Stage, sq. ft.*
At Start	25·400	0·0417	—
After 1st Stage	12·700	0·0833	0·0416
,, 2nd ,, ...	6·350	0·1667	0·0834
,, 3rd ,, ...	3·175	0·3333	0·1666
,, 4th ,, ...	1·588	0·6667	0·3334
,, 5th ,, ...	0·794	1·3333	0·6666
,, 6th ,, ...	0·397	2·6667	1·3334
,, 7th ,, ...	0·198	5·3333	2·6666
,, 8th ,, ...	0·099	10·6667	5·3334
,, 9th ,, ...	0·050	21·3333	10·6666
,, 10th ,, ...	0·025	42·6667	21·3334
,, 11th ,, ...	0·012	85·3333	42·6666
,, 12th ,, ...	0·006	170·6667	85·3334
,, 13th ,, ...	0·003	341·3333	170·6666
Total new surface produced		—	341·2916

Let us assume that it would take A ft.-lb. to effect the first stage of reduction shown in Table III. According to Kick, it would take 13 A ft.-lb. to achieve the 13 stages of reduction. According to Rittinger, however, if it took A ft.-lb. to produce 0·0416 sq. ft. of new surface, it would require 8,204 A ft.-lb. to produce 341·2916 sq. ft. of new surface. Therefore it would require 8,204 A ft.-lb. to effect the 13 stages of reduction given in Table III.

Thus, it is disclosed that, if it took A ft.-lb. to break an inch cube into eight smaller cubes, it would require, by Rittinger's and Kick's hypotheses, 8,204 A and 13 A ft.-lb. respectively to reduce the inch cube to cubes of 3 microns in diameter. Therefore, assuming A to be known, Rittinger's hypothesis demands 630 times more energy than Kick's theory to reduce a 1-in. cube down to 3-micron cubes.

If it can be proved that the production of new surface is directly proportional to useful work absorbed, then Rittinger's hypothesis must be accepted and Kick's discarded.

As already stated, it will be demonstrated that to cause fracture in rock of about 1 sq. in. cross section requires approximately 3·5 ft.-lb. per sq. ft. of new area produced. Thus, to break the 1-in. cube into eight equal-sized ½-in. cubes, whereby the new surface produced is equivalent to 6 sq. in. or 0·0416 sq. ft., the energy required would be 0·146 ft.-lb. Thus, to effect the first stage of Table III would require 0·146 ft.-lb. According to Kick, the total energy required to crush the inch cube to 3-micron particles would, on this basis, be 1·90 ft.-lb. According to Rittinger, however, it would be 1,197·8 ft.-lb.

It is known that, on the Witwatersrand, the power consumed in comminution is approximately 24 h.p.-hours per ton of ore crushed to *minus* 200-linear mesh in which the average size of the grains is 3 microns (13). This is equivalent to 23,760 ft.-lb. per lb. of ore or 2,336 ft.-lb. per 1 cu. in. particle.

On the foregoing assumptions the efficiencies of the crushing and grinding units on the Witwatersrand would average 0·08 per cent according to Kick and 51·3 per cent according to Rittinger.

In practice, in achieving any desired degree of grinding, it is not possible to prevent grinding some of the particles considerably finer than required, and it is frequently argued that the energy expended on these overground particles is wasted energy, which, it has been claimed, may amount to as much as 90 per cent of the total energy expended on grinding (14). It is true that, in certain base-metal metallurgy, overgrinding does adversely affect the extraction of the valuable constituent, but, in other cases, overgrinding does not adversely affect the product. For example, in Witwatersrand gold metallurgy, overgrinding actually increases the amount of gold extracted but not to a sufficient extent to compensate for the cost of the additional energy expended. Thus, in the latter case, it can be argued that only a portion of the cost of the additional grinding is wasted.

An important factor in the prevention of overgrinding is provision for quick passage of the ore through the cylindrical mills combined with adequate classification for the removal of the particles as soon as possible after reaching the desired fineness. It has often been claimed that the capacity of a mill can be increased materially by providing additional classification but, in actual fact, additional classification does not to any extent alter the amount of grinding performed by a mill. It does, however, minimize the work done on particles already fine enough and diverts such work to particles still requiring comminution. Thus, the amount of useful work performed by a mill can, by means of additional classification, be materially increased.

Since, therefore, overgrinding can be controlled, any overgrinding which may result should not be debited against the grinding unit.

Experimental Determinations of Energy Required to Produce Comminution

Gross [15], using a drop-ball on graded sizes of quartz varying from 4-mesh to 100-mesh, found that it required 3·82 ft.-lb. to produce 1 sq. ft. of new surface.

Bond and Maxson [16] used a pendulum impact testing device to crush samples of ore from different mining districts. The samples were crushed and the *minus* 14 *plus* 20-mesh fraction was tested in each case. The work absorbed per unit of surface area produced was as shown in Table VII, the surface area being

Table VII

Sample Tested	*Power Used*	
	Joules, per sq. metre	*Ft.-lb., per sq. ft.*
New Cornelia Copper Ore	**296**	20·3
Little Long Lac Ore.......................	**307**	21·0
Witwatersrand Springs Mine Ore	**462**	31·7
Petroleum Coke, Tenn....................	**760**	52·1

determined in each case on the basis of screen analysis. Table VII indicates that, for Witwatersrand quartzites, the power required to produce 1 sq. ft. of surface is 31·7 ft.-lb.

Gross and Bond both found that the energy absorbed in comminution was directly proportional to the new surface produced, but their results differ very materially. Thus, whereas Bond found it required 31·7 ft.-lb. to produce 1 sq. ft. of new surface, the finding of Gross was only 3·82 ft.-lb. per sq. ft., the former being approximately eight times higher than the latter. It should be noted, however, that Bond based the production of new surface on screen analysis, whereas Gross used the hydrofluoric acid method, which gave areas two to four times as great as those derived from screen calculations.

Experimental work has recently been carried out by the metallurgical staff of The Central Mining and Investment Corporation, Ltd., to determine the minimum power required to effect fracture by impact, shear, bending, compression, and by tension. In all cases, the area of fracture was measured by one or other of the three methods used at the Rand Mines Laboratory.

Impact Tests.—Tests were carried out on pieces of carefully-selected Witwatersrand quartzitic rock, but, instead of dropping a weight on the stationary rock sample, the rock was dropped on to a steel plate through increasing heights until the rock finally fractured. Similar tests were carried out by Stanley ([17]) in 1914.

The rocks were tested from heights of 10 ft., 20 ft., 30 ft., 40 ft., and 50 ft., each rock, after failing to break, being reweighed and tested at the next height. Fracture was assumed to have taken place when the largest remnant did not exceed 80 per cent of the total weight of the sample.

Twenty-four samples were tested, but it is not necessary to record more than 12 results, these being entirely representative of the total. These 12 tests are divided into four groups, wherein the minimum dimensions of the test pieces are not less than 2¾ in., 2⅜ in., 2 in., and 1¼ in. Of the 12 test pieces, three remained unbroken after the 50-ft. drop. Relevant data are given in Table VIII.

Shear Tests.—These tests were carried out at the Rand Mines Laboratory on Witwatersrand quartzitic diamond-drill cores of approximately 1-in. and 13/16-in. diameters.

In an attempt to emulate direct shear conditions, the test piece was firmly clamped in a pipe-vice with 2½-in. jaws in a horizontal position with 1¾ in. of core protruding beyond the vice. Thus, the distance from the centre of the vice to the end of the core was

TABLE VIII

Group	*Minimum diameter of Test Piece, in.*	*Weight, lb.*	*Height of Drop, ft.*	*Energy Input, ft.-lb.*	*Ft.-lb. per lb. of Test Piece*	*Fracture Area, sq. in.*	*Ft.-lb. per sq. ft. of Fracture Area*
	3	1·852	Unbroken	—	—	—	—
A	3	2·253	30	67·6	30·0	163·4	59·6
	2¾	1·731	Unbroken	—	—	—	—
	2½	1·583	20	31·7	20·0	195·3	23·4
B	2½	1·325	50	66·3	50·0	173·1	55·2
	2⅜	1·510	40	60·4	40·0	169·3	51·4
Average A & B	2⅝	1·668	35	56·5	33·9	175·2	46·4
	2⅛	1·182	20	23·6	20·0	256·0	13·3
C	2⅛	0·924	40	37·0	40·0	110·4	48·2
	2	1·065	20	21·3	20·0	48·1	63·8
	1⅜	0·578	40	23·1	40·0	141·4	23·5
D	1⅜	0·408	Unbroken	—	—	—	—
	1¼	0·507	50	25·4	50·0	84·8	43·1
Average C & D	1¼	0·851	34	26·1	30·7	128·2	29·3

3 in. Direct contact between the jaws and the test piece was avoided by interposing a 2½-in. length of split piping of suitable diameter to fit the core under test. A steel ball was dropped from gradually increasing heights on to the protruding piece at a point a ¼ in. from the end, equivalent to 2¾ in. from the centre of the vice, until fracture occurred. A 2-lb. steel ball was used on the larger diameter cores and a ½-lb. ball on the smaller. The dropping height was increased by ½ in. stages. The height at which

fracture occurred and the length and weight of the broken piece were recorded.

TABLE IX

Test No.	*Diam. of Core, in.*	*Cross Section of Core, sq. in.*	*Length of Drop, in.*	*Weight of Ball, lb.*	*Energy Input to achieve Fracture, ft.-lb.*	*Weight of Broken Piece, lb.*	*Ft.-lb. per lb.*	*Area of Fracture, sq. in.*	*Ft.-lb. per sq. ft. of Fracture Area*
1	1·053	0·871	8·0	2·0	1·33	0·179	7·43	4·15	46·1
2	1·044	0·856	4·0	,,	0·67	0·200	3·35	3·83	25·2
3	1·049	0·864	5·0	,,	0·83	0·205	4·05	4·10	29·2
4	1·049	0·864	4·0	,,	0·67	0·160	4·19	3·93	24·5
5	1·030	0·833	2·5	,,	0·42	0·182	2·31	4·46	13·6
6	1·040	0·849	3·0	,,	0·50	0·161	3·11	3·86	18·7
7	1·047	0·861	2·5	,,	0·42	0·174	2·41	4·61	13·1
8	1·039	0·848	3·5	,,	0·58	0·170	3·41	4·31	19·4
9	1·034	0·840	3·5	,,	0·58	0·163	3·56	4·35	19·2
Av'ge.	1·043	0·854	4·00		0·67	0·177	3·79	4·18	23·1
10	0·827	0·537	4·0	0·5	0·17	0·114	1·49	2·09	11·7
11	0·828	0·538	4·5	,,	0·19	0·110	1·73	2·20	12·4
12	0·828	0·538	4·5	,,	0·19	0·111	1·71	2·22	12·3
13	0·813	0·519	3·5	,,	0·15	0·103	1·46	2·02	10·7
14	0·810	0·515	4·0	,,	0·17	0·107	1·59	2·76	8·9
15	0·817	0·524	5·5	,,	0·23	0·104	2·21	1·68	19·7
Av'ge.	0·821	0·528	4.33		0·18	0·108	1·67	2·16	12·0

The usual point of fracture was about 1 in. from the centre of the vice, so that the broken piece averaged approximately 2 in. in length. In a few cases, when a long specimen was under test, the fracture actually took place on the remote side of the vice, presumably due to vibration transmitted through the core. The latter tests were not recorded.

Parallel tests were made which proved that the repeated impacts on each test piece prior to fracture did not weaken the test pieces, whereby low final results might have been obtained.

The actual areas of fracture were determined by water immersion in a manner somewhat similar to Rand Mines Laboratory Method No. I already described. The actual fracture areas determined in this way were from 1·6 to 2·7 times as great as the theoretical areas based on plane surfaces, the average being 2·3 to 1. Relevant data are given in Table IX.

Bending Tests.—Adams and Coker (18) carried out some interesting research upon the deflection of beams cut from rocks. In one test on pure white marble from Vermont, a lath-shaped piece, 12 in. long by 1·26 in. by 0·29 in., was rested on two wedge-shaped supports placed 11 in. apart and then loaded in the middle. The broader surface rested on the terminal supports. The weights were placed in a light brass pan hanging from a wire which passed over the middle of the lath and lay flat upon it. The deflection at the centre of the beam was measured after each weight addition up to the point of fracture and the total test took about half an hour. The final weight applied was 86 oz. and the deflection was 0·068 in. when fracture occurred.

Eleven similar tests were carried out at the Rand Mines Laboratory, on Witwatersrand quartzitic diamond-drill cores of 1-in. and $\frac{13}{16}$-in. diameters. With the larger cores, two tests were carried out with a 15-in. span between supports. With the smaller cores, spans of 14·0 in., 10·5 in., and 7·0 in. respectively were used. With the addition of each weight, the deflection at the centre of the beam was measured by micrometer and loading was continued until fracture occurred. The duration of each test averaged 31 minutes. The area of fracture was measured by water immersion as in the case of the shear tests, the ratio of actual to theoretical surface being about 2·4 to 1. In calculating the energy required to produce fracture, the weight of the core between supports was ignored owing to its relatively small effect upon the final result.

Details of the first test on the 1-in. core are given in Table X.

TABLE X

QUARTZITIC CORE DIAMETER = 1·039 IN.		SPAN = 15·0 IN. WEIGHT = 1·271 LB.	
Weight in Pan per Stage, lb.	*Average Weight in Pan per Stage, lb.* (a)	*Deflection per Stage, in.* (b)	*Work done per Stage in in.-lb.* (b × a)
0·0 to 7·0	3·5	0·001	0·0035
7·0 „ 13·9	10·5	0·001	0·0105
13·9 „ 20·8	17·3	0·001	0·0173
20·8 „ 27·6	24·2	0·001	0·0242
27·6 „ 34·5	31·1	0·001	0·0311
34·5 „ 41·4	37·9	0·001	0·0379
41·4 „ 48·3	44·9	0·001	0·0449
48·3 „ 55·2	51·7	0·001	0·0517
55·2 „ 62·1	58·7	0·001	0·0587
62·1 „ 68·9	65·5	0·001	0·0655
68·9 „ 75·8	72·4	0·001	0·0724
75·8 „ 82·7	79·2	0·001	0·0792
82·7 „ 89·6	86·2	0·001	0·0862
89·6 „ 97·8	93·7	0·001	0·0937
97·8 „ 106·1	102·0	0·001	0·1020
106·1 „ 114·4	110·2	0·001	0·1102
114·4 „ 122·6	118·5	0·001	0·1185
122·6 „ 128·1	125·4	0·001	0·1254
128·1 „ 135·6	131·8	0·001	0·1318
135·6 „ 145·0	140·3	0·001	0·1403
145·0 „ 154·4	149·7	0·001	0·1497
154·4 „ 159·1	156·8	0·001	0·1568
159·1 „ 163·9	161·5	0·001	0·1615
163·9 „ 167·6	165·7	0·001	0·1657
0 „ 167·6	84·95	0·024	2·0387

2·0387 in.-lb. = 0·1699 ft.-lb.

For each of the 11 tests, the work done was calculated in the manner shown in Table X, and the complete list of results are given in Table XI.

TABLE XI

Test No.	Portion of Core under Test				Breaking Load, lb.	Deflection, in.	Energy Input, ft.-lb.	Ft.-lb. per lb. under Test	Area of Fracture, sq. in.	Ft.-lb. per sq. ft. of Fracture
	Diameter in.	Cross Section, sq. in.	Length, in.	Weight, lb.						
1	1·039	0·848	15·0	1·271	167·6	0·024	0·170	0·134	4·205	5·82
2	1·034	0·840	15·0	1·259	116·3	0·016	0·083	0·066	4·349	2·75
Average	1·036	0·844	15·0	1·265	142·0	0·020	0·126	0·100	4·277	4·24
3	0·813	0·519	14·0	0·726	40·2	0·011	0·020	0·028	2·131	1·35
4	0·810	0·515	14·0	0·720	48·2	0·017	0·042	0·058	2·362	2·56
5	0·817	0·524	14·0	0·733	82·9	0·021	0·076	0·104	1·670	6·55
Average	0·813	0·519	14·0	0·726	57·1	0·016	0·046	0·063	2·054	3·22
6	0·816	0·523	10·5	0·549	54·3	0·014	0·031	0·056	2·419	1·85
7	0·820	0·528	10·5	0·554	85·0	0·017	0·063	0·114	2·318	3·91
8	0·815	0·522	10·5	0·548	68·4	0·015	0·044	0·080	2·261	2·80
Average	0·817	0·524	10·5	0·550	69·2	0·015	0·046	0·084	2·333	2·84
9	0·813	0·519	7·0	0·363	115·8	0·006	0·027	0·074	1·915	2·03
10	0·810	0·515	7·0	0·360	119·6	0·007	0·041	0·114	2·362	2·50
11	0·817	0·524	7·0	0·366	120·5	0·006	0·030	0·082	1·699	2·54
Average	0·813	0·519	7·0	0·363	118·6	0·006	0·033	0·091	1·992	2·39

Table XI discloses that to fracture the 1-in. core and the $\frac{13}{16}$-in. core required 4·24 ft.-lb. and 2·82 ft.-lb. respectively per sq. ft. of new surface produced. The average of the 11 tests is 3·15 ft.-lb. per sq. ft. of fracture area.

Another series of bending tests was carried out at the Rand Mines Laboratory on 1-in. and $\frac{13}{16}$-in. Witwatersrand quartzitic diamond-drill cores in which one end was fixed in a clamp $2\frac{1}{4}$ in. long and a load was gradually applied at a point $\frac{1}{4}$ in. from the free end. The deflection at the loading point was measured by micrometer. Other details were similar to those of the bending tests already described. The average duration of each test was 50 minutes.

Relative data are given in Table XII (Plate I).

From the data in Table XII, the average energy to fracture a 1-in. core as disclosed by the 12 tests thereon is 4·09 ft.-lb. per sq. ft. of new surface produced and 3·70 ft.-lb. per sq. ft. for the $\frac{13}{16}$-in. core. The average of the 15 tests is 4·02 ft.-lb. per sq. ft.

The average energy required to effect fracture in the 26 tests carried out in both series of bending tests is 3·73 ft.-lb. per sq. ft. of new surface produced.

Compression Tests.—In 1906, Adams and Coker [18] tested the cubic compressibility of marble, granite and other rocks without, in any case, carrying the test to the point of rupture. In Table XIII is given a summary of all of the tests carried out, in which the load applied was 8,000 lb. The compression is expressed in millionths of an inch and as a percentage of the length of the test pieces. In the final column is given the calculated compression per cent which would obtain on the basis of the applied pressure being 10,000 lb. per sq. in. It is indicated that for marble and granite the compressibility at 10,000 lb. per sq. in. is approximately 0·113 per cent and 0·139 per cent respectively. It is interesting to note that the compressibility of plate glass is 0·095 per cent.

Similar compression tests were carried out at the Government Mechanical Laboratory, Cottesloe, with the kind assistance of the Director and his staff.

Eight cylindrical test pieces were prepared from quartzitic diamond-drill cores of 1-in. and $1\frac{3}{8}$-in. diameter, the length of each test piece being double its diameter. The ends were carefully cut and ground so that they were exactly parallel to each other and at right angles to the long axis. The actual surface of the test pieces was 1·3 times theoretical.

Each test piece was compressed until fracture occurred, the broken pieces being retained by means of a guard of rubber

Table XIII

(After Adams and Coker)

Length of Specimen under Test = 1·25 in.

Load applied = 8,000 lb.

Rock	*Cross Sectional Area, sq. in.*	*Load, lb. per sq. in.*	*Compression in Millionths of an inch*	*Compression, per cent*	*Calculated Compression, per cent at 10,000 lb. per sq. in.*
Black Belgian Marble	0·922	8,680	960	0·077	0·084
Carrara Marble	1·033	7,740	1,220	0·098	0·127
Vermont Marble	1·029	4,860	800	0·064	0·132
Tennessee Marble	0·973	8,220	1,140	0·091	0·111
Montreal Limestone	0·755	6,620	900	0·072	0·109
Average for Marble	—	—	—	—	0·113
Italian Granite	0·750	10,670	1,720	0·138	0·129
Scotch Granite	1·029	7,770	1,220	0·098	0·126
New Brunswick Granite ...	0·984	8,130	1,210	0·097	0·119
Rhode Island Granite	0·855	9,360	1,600	0·128	0·137
Massachusetts Granite	1·012	7,910	1,470	0·118	0·149
Quebec Granite	0·976	8,200	1,815	0·145	0·177
Average for Granite	—	—	—	—	0·139
Montreal Syenite	1·003	7,980	1,090	0·087	0·109
Quebec Essexite	0·952	8,400	1,080	0·086	0·102
Ontario Olivine Diabase ...	0·800	10,000	920	0·074	0·074
Plate Glass.....................	1·010	7,920	940	0·075	0·095

insertion tied loosely around the specimen. The average loading rate was 6 tons per minute and each test piece was under compression for about three minutes.

The compression of each test piece up to the point of fracture was determined by measuring the distance between two points on the upper and lower jaw plates of the testing machine at regular intervals by means of a micrometer gauge. It was found necessary to apply a correction to the figures thus obtained due to take-up in the testing unit. This correction was determined by preparing steel test pieces of the same dimensions as the rock test pieces and of known elastic modulus (30,000,000 lb. per sq. in.). These steel pieces were tested over the same loading range as the rock pieces, the compression being measured in the same way. The difference between the observed and the theoretical deformation of the steel pieces at any load was then subtracted from the observed deformation of the rock test piece at the same load.

The work done on each test piece up to the point of fracture was calculated from the force applied and the deflection produced at each intermediate reading in the same manner as shown in Table X.

The quartzitic test pieces usually broke into two conical pieces, the bases of which constituted the relatively undamaged end faces of the original test piece, a number of longitudinal slivers which frequently approximated to the full length of the test piece and originally composed the cylindrical surface of the piece and a quantity of smaller fragments and fines from the interior of the test piece. In Fig. 1 (Plate III) is shown the total component parts of No. 1 test piece. Fig. 2 (Plate III) shows the two resulting cones from Test No. 2, alongside the corresponding steel test piece. Tests 3, 4 and 5 were carried out on test pieces taken from the same diamond-drill core. The fractures in these test pieces are shown in Fig. 3 (Plate IV). Test pieces 6, 7 and 8 were taken from another diamond-drill core and the resulting fractures are shown in Fig. 4 (Plate IV). It will be observed that a diagonal fracture with very little production of fines occurred in Test No. 8.

The fragments were graded into 10 sizes, ranging from 1-in. to 200-mesh material.

The areas of the *plus* ½-in. fractions of the test pieces were determined by Rand Mines Laboratory Method No. I.

The areas of the *minus* ½-in. and *plus* 200-mesh fractions were determined by weighing the portions and applying the factors shown in Table II, which had been determined on similar quartzitic fractions by Rand Mines Method No. II and the permeability Method No. III.

Table XIV

Tests Nos.		1	2	3	4	5	6	7	8	1	2	3	4	5	6	7	8
Grading of Products	*Surface Area, sq. ft. per lb.*	*Per Cent by Weight*								*Surface Area, sq. ft. per lb. Original*							
+½ in.	—	62·3	50·8	75·1	58·1	46·4	91·5	84·6	93·5	1·4	0·9	1·6	2·7	1·5	1·5	1·8	1·2
—½ in. + 0·37 in.	5·4	12·8	13·5	10·6	1·3	23·1	3·6	9·3	3·1	0·7	0·7	0·6	0·1	1·2	0·2	0·5	0·2
—0·37 in. + 3 mesh	10·2	6·0	7·6	4·1	17·8	10·6	—	2·8	1·1	0·6	0·8	0·4	1·8	1·1	—	0·3	0·1
— 3 + 6 ,,	14·6	6·7	11·7	3·6	9·2	6·9	2·3	0·8	0·5	1·0	1·7	0·5	1·3	1·0	0·3	0·1	0·1
— 6 + 8 ,,	23·7	1·8	2·2	1·2	2·5	2·2	0·3	0·5	0·3	0·4	0·5	0·3	0·6	0·5	0·1	0·1	0·1
— 8 + 20 ,,	28·1	4·3	6·0	2·1	5·0	5·5	1·1	0·8	0·5	1·2	1·7	0·6	1·4	1·5	0·3	0·2	0·1
— 20 + 48 ,,	35·2	3·2	4·2	1·7	3·7	3·0	0·5	0·6	0·4	1·1	1·5	0·6	1·3	1·1	0·2	0·2	0·1
— 48 + 100 ,,	84·0	1·4	1·9	0·8	1·3	1·2	0·4	0·3	0·3	1·2	1·6	0·7	1·1	1·0	0·3	0·3	0·3
—100 + 200 ,,	188·5	0·9	1·1	0·5	0·7	0·7	0·2	0·2	0·2	1·7	2·1	0·9	1·3	1·3	0·4	0·4	0·4
—200 ,,	1,183·0	0·6	1·0	0·3	0·4	0·4	0·1	0·1	0·1	7·1	11·8	3·5	4·7	4·7	1·2	1·2	1·2
Surface Area per lb.										16·4	23·3	9·7	16·3	14·9	4·5	5·1	3·8

The area of the *minus* 200-mesh fraction was determined by the permeability method on the combined *minus* 200-mesh material from all of the tests. This fraction contained 58 per cent. of *minus* 325-mesh material. The gradings of the products are given in Table XIV. Relevant data for the Compression Tests are given in Table XV.

Table XV discloses that it requires a load of approximately 30,000 lb. per sq. in. to crush Witwatersrand quartzites, which agrees reasonably well with the finding of Weiss [19]. It indicates also that at a pressure of 30,000 lb. per sq. in. the compression is approximately 0·30 per cent, which is somewhat less than that for marble and granite as shown in Table XIII.

Tension Tests.—Twelve tension tests were carried out at the Government Mechanical Laboratory on 1-in. diameter Witwatersrand quartzitic diamond-drill cores.

Nothing was done to the cores to ensure fracture at any predetermined point, therefore the point of fracture was unpredictable.

The cores were gripped by means of steel chucks specially designed to give an even pressure over a 1½-in. length at each end of the core. The pressure within the grips could be varied by means of a nut operating on three wedge pieces. The grips were attached to the tension machine by means of ball and socket joints so that the test piece automatically aligned itself and the force exerted was definitely longitudinal.

The length of core tested—i.e., the distance between the grips—was varied from 10·7 in. down to 2·7 in. Each test piece was under tension for about one minute.

During each test, the distance between two fixed points on the grips was measured by means of a micrometer gauge at loading intervals of 200 lb. The extension thus measured included strains in the grips and in the portion of each test piece within the grips and therefore a correction was necessary.

To determine the correction, three further tests were carried out in which the distance between the grips was nil. Theoretically, in these circumstances, there should have been no extension, but it actually amounted to one-thousandth of an inch at a load of 2,600 lb., equivalent to 0·00008 in. for each 200 lb. of load applied.

The energy inputs in the 12 tests were calculated from the load applied and the corrected extensions on the same lines as shown in Table X.

In six cases fracture occurred between the grips, in two cases at the grips and on four occasions inside the grips.

TABLE XV

Test No.	Test Piece					Crushing Load, lb. per sq. in.	Corrected Compression		Energy Input, ft.-lb.	Ft.-lb. per lb.	Surface Area after Crushing, sq. ft.	Surface Area Produced, sq. ft.	Ft.-lb. per sq. ft. of Surface Produced
	Diam., in.	Cross Section, sq. in.	Height, in.	Weight, lb.	Measured Surface Area, sq. ft.		In.	Per cent of Original Height					
1	1·380	1·496	2·762	0·403	0·14	35,600	0·0092	0·333	22·2	55·1	6·61	6·47	3·43
2	1·380	1·496	2·763	0·403	0·14	35,000	0·0082	0·297	24·2	60·0	9·39	9·25	2·62
Average	1·380	1·496	2·762	0·403	0·14	35,300	0·0087	0·315	23·2	57·6	8·00	7·86	2·95
3	1·045	0·858	2·024	0·168	0·08	30,000	0·0064	0·316	7·7	45·8	1·63	1·55	4·97
4	1·045	0·858	2·020	0·167	0·08	33,800	0·0059	0·292	8·1	48·5	2·72	2·64	3·07
5	1·045	0·858	2·020	0·167	0·08	32,600	0·0066	0·327	7·2	43·1	2·49	2·41	2·99
6	1·051	0·868	2·025	0·181	0·08	21,400	0·0056	0·277	3·8	21·0	0·81	0·73	5·21
7	1·051	0·868	2·030	0·181	0·08	22,400	0·0049	0·241	4·2	23·2	0·92	0·84	5·00
8	1·051	0·868	2·025	0·181	0·08	16,200	0·0039	0·193	1·9	10·5	0·69	0·61	3·11
Average	1·048	0·863	2·024	0·174	0·08	26,067	0·0055	0·272	5·5	31·6	1·54	1·46	3·77

The area of each fracture face was determined by water immersion as in the case of the shear tests. The ratio between theoretical and actual fracture area was found to be smaller (1·75 to 1) than in the case of the bending tests, which suggests that tension produces a smoother fracture surface than bending.

The relevant data are tabulated in Table XVI.

It is interesting to note that the load per sq. in. required to crush Witwatersrand quartzites by compression (Table XV) is approximately 18 times as great as that required by tension. This confirms results found by Wyburgh ([20]) on South African building stones.

Discussion of Experimental Results

A summary of the results obtained by breaking Witwatersrand quartzites by impact, shear, bending, compression and by tension is given in Table XVII.

Due to the fact that, in the impact tests on lump rock, no correction was made for the manner in which the test pieces may have dropped or for the extent to which they or the resulting fragments may have rebounded, no definite conclusions should be drawn from these tests.

Likewise, in the shear tests, due to the fact that no correction was made for the steel-ball rebound, etc., no definite conclusions should be made.

Table XVII discloses that, notwithstanding the heterogeneous nature of Witwatersrand quartzites, there is a reasonably close relationship between energy input and new fracture surface produced. For example, in the 11 bending test summaries given in Table XVII, the highest result is 5·05 ft.-lb. per sq. ft. of new surface and the lowest 2·39 ft.-lb., the average being 3·73 ft.-lb. per sq. ft.

In the compression tests, the production of 1 sq. ft. of new surface area required 3·36 ft.-lb., and, after crushing, approximately 30 per cent by weight of the test pieces was reduced to *minus* $\frac{1}{4}$-in. material.

It is significant that it takes virtually the same energy to produce 1 sq. ft. of new surface area irrespective of whether the rock is merely broken into two pieces or shattered into many fragments.

In the tension tests, summarized in Table XVII, it is not surprising that the energy required per sq. ft. of fracture area is roughly proportional to the length of the test piece and, indeed, it may be deduced from these results that, had the specimen under test been 2·0 in. long, it would have required 2·3 ft.-lb. per sq. ft. of fracture produced.

TABLE XVI

Test No.	Test Piece: Length between Grips, in.	Test Piece: Diam., in.	Test Piece: Cross Section, sq. in.	Test Piece: Weight between Grips, lb.	Breaking Load lb. per sq. in.	Corrected Extension: In.	Corrected Extension: Per Cent	Energy Input, ft.-lb.	Ft.-lb. per lb. of Test Piece	Area of Fracture, sq. in.	Ft.-lb. per sq. ft. of Fracture Area
1	10·7	1·054	0·873	0·926	1,380	0·0021	0·020	0·113	0·12	3·38	4·8
2	10·6	1·045	0·858	0·901	2,220	0·0047	0·044	0·379	0·42	2·71	20·1
3	9·2	1·040	0·850	0·775	1,880	0·0037	0·040	0·281	0·36	2·88	14·1
4	7·8	1·028	0·830	0·641	1,960	0·0036	0·046	0·289	0·45	2·51	16·6
Average	9·58	1·042	0·853	0·811	1,860	0·0035	0·037	0·265	0·327	2·870	13·30
5	6·2	1·045	0·858	0·527	2,800	0·0020	0·032	0·177	0·34	2·64	9·7
6	6·0	1·044	0·856	0·509	1,820	0·0010	0·017	0·073	0·14	3·23	3·3
7	5·2	1·040	0·850	0·438	2,220	0·0011	0·021	0·066	0·15	2·85	3·3
8	5·0	1·040	0·850	0·421	2,480	0·0024	0·048	0·240	0·57	3·30	10·5
Average	5·60	1·042	0·853	0·474	2,330	0·0016	0·029	0·139	0·293	3·005	6·66
9	3·4	1·047	0·861	0·291	2,340	0·0011	0·032	0·072	0·25	2·66	3·9
10	3·4	1·043	0·854	0·287	1,600	0·0009	0·026	0·062	0·22	3·46	2·6
11	3·4	1·040	0·850	0·286	2,560	0·0008	0·024	0·077	0·27	2·68	4·1
12	2·7	1·044	0·856	0·229	1,860	0·0005	0·019	0·029	0·13	3·21	1·3
Average	3·23	1·043	0·855	0·273	2,090	0·0008	0·025	0·060	0·220	3·002	2·88
a	Nil	1·049	0·864	—	2,894	0·0016	—	—	—	—	—
b	,,	1·049	0·864	—	2,430	0·0007	—	—	—	—	—
c	,,	1·045	0·858	—	3,030	0·0006	—	—	—	—	—
	,,	1·048	0·862	—	2,784·7	0·0010	—	—	—	—	—

TABLE XVII

Nature of Test	Test piece			Breaking Load		Corrected Deflection, in.	Energy Input, ft.-lb.	Ft.-lb. per lb. of Test Piece	Surface Area Produced, sq. in.	Ft.-lb. per sq. ft. of Surface Area Produced
	Length, in.	Diam., in.	Weight, lb.	Lb.	Lb. per sq. in.					
Impact	—	2 3/8	1.668	—	—	—	56.5	33.9	175.2	46.4
Lump Rock	—	1 1/4	0.851	—	—	—	26.1	30.7	128.2	29.3
Shear	2.0	1.043	0.177	—	—	—	0.67	3.79	4.18	23.1
Round Core	2.0	0.821	0.108	—	—	—	0.18	1.67	2.16	12.0
Centre Bending	15.0	1.036	1.265	142.0	—	0.020	0.126	0.100	4.277	4.24
Round Core	14.0	0.813	0.726	57.1	—	0.016	0.046	0.063	2.054	3.22
	10.5	0.817	0.550	69.2	—	0.015	0.046	0.084	2.333	2.84
	7.0	0.813	0.363	118.6	—	0.006	0.033	0.091	1.992	2.39
End Bending	15.0	1.043	1.269	23.7	—	0.095	0.104	0.082	4.219	3.55
Round Core	11.75	1.043	0.951	37.6	—	0.083	0.148	0.156	4.219	5.05
	7.5	1.043	0.635	59.0	—	0.049	0.139	0.219	4.219	4.74
	6.0	1.034	0.499	67.7	—	0.029	0.100	0.200	5.753	2.50
	5.0	1.039	0.420	92.9	—	0.022	0.105	0.250	4.205	3.60
	3.75	1.042	0.316	136.1	—	0.020	0.127	0.402	4.474	4.09
	15.0	0.827	0.798	13.0	—	0.095	0.056	0.070	2.179	3.70
Av'ge all Bending Tests	10.05	0.959	0.708	74.3	—	0.041	0.094	0.133	3.6295	3.73
Compression	2.762	1.380	0.403	52,800	35,300	0.0087	23.2	57.6	1131.8	2.95
Round Core	2.024	1.048	0.174	22,500	26,067	0.0055	5.5	31.6	210.2	3.77
Tension	9.58	1.042	0.811	1,590	1,860	0.0035	0.265	0.327	2.870	13.30
Round Core	5.60	1.042	0.474	1,090	2,330	0.0016	0.139	0.293	3.005	6.66
	3.23	1.043	0.273	1,790	2,090	0.0008	0.060	0.220	3.002	2.88

John Gross [15] found that it took 3·82 ft.-lb. to produce 1 sq. ft. of new surface. His areas were determined by the hydrofluoric acid method, which is comparable with the methods described herein.

Based on the foregoing evidence, it is concluded that the surface area produced in crushing and grinding is directly proportional to the energy absorbed, which is in accordance with Rittinger's hypothesis and contrary to Kick's theory. It is further concluded that, in crushing and grinding Witwatersrand quartzites, it requires approximately 3·5 ft.-lb. to produce 1 sq. ft. of surface area.

The Determination of the Efficiencies of Industrial Machines

Based on the finding that surface area produced in crushing and grinding is directly proportional to the energy consumed and that it requires 3·5 ft.-lb. to produce 1 sq. ft. of surface area, the efficiencies of crushing and grinding machines on the Witwatersrand can be calculated. For this purpose, the mechanical equivalent of the surface area produced (3·5 ft.-lb. per sq. ft.) is determined and the latter is expressed as a percentage of the power available for comminution or alternatively as a percentage of the total power input.

In industrial plants, from 5 per cent to 10 per cent of the total power input may be dissipated in the driving motors. A further 5 per cent to 35 per cent might be used to maintain the machines in motion without any load, so that the maximum power available to achieve crushing and grinding may only amount to 55 per cent to 90 per cent of the total power input. The efficiency of the machines is best expressed as a percentage of the power available to achieve crushing and grinding.

Efficiency tests have been carried out by members of the Metallurgical Staff of The Central Mining and Investment Corporation, Ltd., on rock drills, gyratory crushers, stamps and on cylindrical mills in operation on the Witwatersrand. In all cases, the rock surface areas were measured by one or other of the three methods in use at the Rand Mines Laboratory. All electric power readings were taken with test watt-meters.

Rock Drill Test.—At Modderfontein B. Gold Mines, Ltd., a $1\frac{3}{16}$-in. hole was drilled underground in the solid rock to a depth of 42 in. in three minutes. The grit and slime collected from the hole weighed 4·57 lb. and it was all *minus* 3-linear mesh containing

35 per cent *minus* 100-mesh. The surface area of the product was 500·0 sq. ft. per lb., equivalent to a total surface of 2,285·0 sq. ft. The surface area of the hole was 1·1 sq. ft., so that the total surface produced was 2,286·1 sq. ft.

The drill operated at 1,850 strokes per minute, each blow delivering 42·5 ft.-lb. The total energy applied in three minutes was thus 235,875 ft.-lb., equivalent to 103·2 ft.-lb. per sq. ft. of new surface produced.

On the assumption that the theoretical requirement is 3·5 ft.-lb. per sq. ft. and that the actual power consumption was 103·5 ft.-lb. per sq. ft. of new surface produced, the efficiency of the drilling machine was 3·4 per cent. Considering the fact that, due to the solid nature of the rock, it could not give under compression and that leverage could not be applied, this would appear to be a satisfactory performance.

Gyratory Crusher Tests.—Three tests were carried out at the Durban Roodepoort Deep mine with a 7-in. Newhouse crusher. The feed to the machine consisted of washed *plus* 1-in. rock ranging in size up to 7-in. pieces. Relevant data are given in Table XVIII.

Table XVIII discloses that 36·6 per cent of the total power input is dissipated in motor and friction losses, etc., leaving 63·4 per cent to achieve crushing. It discloses also that, notwithstanding the different qualities of feed and the different crusher settings, the power consumed per sq. ft. of new surface produced is reasonably constant, being 10·73, 11·10, and 10·20 ft.-lb. per sq. foot. This, of course, supports the Rittinger theory. Table XVIII indicates that the efficiency of the gyratory crusher, in terms of the power available to effect crushing, is 32·8 per cent.

Table XVIII further discloses that, although the feed to the crusher contained not more than 0·4 per cent of *minus* 20-mesh material, the surface area of the latter represented 87·1 per cent of the total surface area of the feed. Likewise, the surface area of the *minus* 20-mesh fraction in the resulting product averaged 92·6 per cent of its total surface area.

Stamp Tests.—Three tests were carried out at the Durban Roodepoort Deep mine with a five-stamp battery in the 100-stamp mill. A clean graded feed was used and all conditions for the tests were similar, with the exception that coarse, medium, and fine screening was used in the first, second, and third tests respectively. The average weight per stamp was 1,490 lb., the height of drop was 0·62 ft., and the number of drops per stamp averaged 103·5 per minute. Thus the potential energy in the five stamps was 14·5 h.p.

Relative data for these tests are given in Table XIX.

TABLE XVIII

Test No.	1	2	3
Type of feed	Coarse	Fine	Fine
Crusher setting	⅞ in.	⅞ in.	1¼ in.
Tons crushed per hour	48·8	56·2	58·7
H.p. on full load	49·8	48·4	37·4
H.p. without feed	17·3	17·8	14·3
H.p. available for crushing	32·5	30·6	23·1
Available h.p. per cent of total	65·3	63·2	61·8

Grading	*Size*	*Area, sq. ft. per lb.*	1: *Per Cent by Weight*	1: *Sq. ft. per lb. original*	2: *Per Cent by Weight*	2: *Sq. ft. per lb. original*	3: *Per Cent by Weight*	3: *Sq. ft. per lb. original*
Feed	—7 in.+4 in.	0·3	41·2	0·12	8·7	0·03	12·9	0·04
	—4 in.+2 in.	0·6	50·7	0·30	26·2	0·16	32·1	0·19
	—2 in.+1 in.	1·0	7·4	0·07	64·4	0·64	53·9	0·54
	—1 in.+½ in.	2·4	0·1	—	0·1	—	0·6	0·01
	—½ in.+6-mesh	10·4	0·1	0·01	0·1	0·01	0·1	0·01
	—6 +20	27·5	0·1	0·03	0·1	0·03	0·1	0·03
	—20	1366·8	0·4	5·47	0·4	5·47	0·3	4·10
			100·0	6·00	100·0	6·34	100·0	4·92
Product	—2 in.+1 in.	1·3	40·1	0·52	39·8	0·52	56·6	0·74
	—1 in.+½ in.	2·8	34·6	0·97	36·8	1·03	26·2	0·73
	—½ in.+6-mesh	10·4	15·6	1·62	15·4	1·60	11·4	1·19
	—6 +20	26·7	5·1	1·36	4·3	1·15	2·9	0·77
	—20	1368·3	4·6	62·94	3·7	50·63	2·9	39·68
			100·0	67·41	100·0	54·93	100·0	43·11

	1	2	3
Lb. crushed per minute	1,627	1,873	1,957
New surface produced; sq. ft. per lb.	61·41	48·59	38·19
New surface produced; sq. ft. per min.	99,914	91·009	74,738
Ft.-lb. available for crushing per minute	1,072,500	1,009,800	762,300
Ft.-lb. available per sq. ft. of new surface	10·73	11·10	10·20
Ft.-lb. required per sq. ft. of new surface (assumed)	3·5	3·5	3·5
Efficiency based on available power, per cent	32·6	31·5	34·3
Efficiency based on total input, per cent	21·3	19·9	21·2

Table XIX

Test No.	1	2	3
Screen opening, in.	0·62	0·31	0·13
Weight of ore fed, lb.............	4,800	3,600	3,000
Ratio water to solids	4·53/1	4·84/1	5·69/1
Tons crushed per hour............	2·796	2·454	2·196
H.p. on full load	26·1	26·0	25·7
H.p. to drive shafting, etc.......	11·6	11·5	11·2
Potential h.p. in stamps.........	14·5	14·5	14·5
Available h.p. per cent of total	55·6	55·8	56·4

Grading	*Size*	*Area, sq. ft. per lb.*	1: *Per Cent by Weight*	1: *Sq. ft. per lb. original*	2: *Per Cent by Weight*	2: *Sq. ft. per lb. original*	3: *Per Cent by Weight*	3: *Sq. ft. per lb. original*
Feed ...	—2 in. +1½ in.	0·8	35·9	0·29	32·8	0·26	22·5	0·18
	—1½ in.+1 in.	1·3	60·6	0·79	62·8	0·82	62·0	0·81
	—1 in. +½ in.	2·4	2·6	0·06	3·2	0·08	14·8	0·36
	—½ in. +6-mesh	9·1	0·1	0·01	0·3	0·03	0·1	0·01
	—6 +20 ...	27·5	0·1	0·03	0·1	0·03	0·1	0·03
	—20	1356·5	0·7	9·50	0·8	10·85	0·5	6·78
			100·0	10·68	100·0	12·07	100·0	8·17
Product	—¾ in.+½ in. ...	3·3	4·4	0·15	—	—	—	—
	—½ in.+6-mesh	12·5	24·3	3·04	18·9	2·36	2·0	0·25
	—6 +20- ,,	27·2	22·8	6·20	25·5	6·94	29·4	8·00
	—20	1013·8	48·5	491·69	55·6	563·67	68·6	695·47
			100·0	501·08	100·0	572·97	100·0	703·72

	1	2	3
Lb. crushed per minute	93·2	81·8	73·2
New surface produced; sq. ft. per lb.	490·40	560·90	695·55
New surface produced; sq. ft. per min.	45,705	45,882	50,914
Ft.-lb. available for crushing per minute	478,500	478,500	478,500
Ft.-lb. available per sq. ft. of new surface	10·47	10·43	9·40
Ft.-lb. required per sq. ft. of new surface (assumed)	3·5	3·5	3·5
Efficiency based on available power, per cent	33·4	33·6	37·2
Efficiency based on total input, per cent.........................	18·6	18·7	21·0

Table XIX discloses that, in stamp milling, 44·1 per cent of the total power input is dissipated in motor and friction losses, etc., leaving 55·9 per cent to achieve crushing. It discloses also that, notwithstanding the appreciable increase in the fineness of the product from the finer battery screening, the power consumed per sq. ft. of new surface produced is reasonably constant, being 10·47, 10·43 and 9·40 ft.-lb. per sq. ft. This agreement again gives support to the Rittinger theory. Table XIX indicates that the efficiency of stamps, in terms of the power available to effect crushing, is 34·7 per cent.

Immediately following the foregoing stamp tests, in which the feed consisted of graded *minus* 2-in. *plus* 1-in. rock relatively free of fines, normal feed containing 7·0 per cent of *minus* 20-mesh material was restored and a test was carried out to determine the production of heat during stamping.

The rate of feed was 15·5 tons per stamp per 24 hours, equivalent to 107·6 lb. solids per minute to the five-stamp battery. The ratio of water to solids passing through the 0·62-in. screen was 4·21 to 1 and in the feed sample 0·03 to 1. The specific heat of the ore was taken at one-fifth that of water—namely, 0·2. Relevant data are given in Table XX.

TABLE XX

Details	*Weight, lb. per minute*	*Water Equivalent, lb. per min.*	*Temperature, °F.*	*B.T.U. per minute*
Entering Mortar Box—				
Solids	107·6	21·5	55·78	1,199
Moisture in feed......	3·2	3·2	55·78	178
Water added	449·8	449·8	62·87	28,279
Totals	560·6	474·5	62·50	29,656
Leaving Mortar Box...	560·6	474·5	63·57	30,164
Heat produced...	—	—	1·07	508

Table XX discloses that 508 B.T.U. per minute were produced, which are equivalent to 12·0 h.p.

Thus of the 14·5 h.p. available to achieve crushing 12·0 h.p., or 82·8 per cent, were registered as heat generated during the crushing operation. In a similar observation, J. Cook [21] accounted for 80 per cent of the input energy in a stamp battery by heat conversion.

Cylindrical Mill Tests.—At the Durban Roodepoort Deep, Ltd., two 6 ft. 6 in. by 20 ft. cylindrical mills were chosen for test purposes, one in the primary and the other in the secondary grinding circuit. An additional test was carried out on a secondary 5 ft. 6 in. by 22 ft. tube-mill at the Crown Mines, Ltd.

TABLE XXII

Tube-Mill	*Details*	*Weight, lb. per minute*	*Water Equivalent, lb. per min.*	*Temperature, °F.*	*B.T.U. per minute*
Durban Deep Primary	*Entering Tube-Mill—*				
	Solids	1,910	382	78·11	29,838
	Moisture in feed	442	442	79·49	35,136
	Water added	431	431	74·80	32,239
	Total	2,783	1,255	77·46	97,212
	Leaving Tube-Mill	2,783	1,255	83·30	104,542
	Heat produced	—	—	5·84	7,330
Durban Deep Secondary	*Entering Tube-Mill—*				
	Solids	933	187	78·98	14,769
	Moisture in feed	286	286	78·80	22,537
	Water added	117	117	70·20	8,213
	Total	1,336	590	77·15	45,519
	Leaving Tube-Mill	1,336	590	90·30	53,277
	Heat produced	—	—	13·15	7,758
Crown Mines Secondary	*Entering Tube-Mill—*				
	Solids	600	120	79·00	9,480
	Moisture in feed	178	178	79·00	14,062
	Water added	33	33	74·52	2,459
	Total	811	331	78·55	26,001
	Leaving Tube-Mill	811	331	97·30	32,206
	Heat produced	—	—	18·75	6,205

The duration of each test was four hours, during which period, at regular intervals, inlet and outlet samples were taken and power readings and tonnage determinations were recorded.

After the tests, the power required to operate the mills when empty was determined and the difference between the power required to operate the empty mills and that required under full-load operating conditions was accepted as the energy available to effect useful work.

Relevant data are given in Table XXI (Plate II).

Table XXI discloses that, in tube milling, 12·6 per cent of the total power input is dissipated in motor and friction losses, etc., leaving 87·4 per cent to achieve grinding. It discloses that, under coarse grinding conditions, it required 14·18 ft.-lb. useful energy to produce 1 sq. ft. of new surface, whereas, in fine grinding, the requirement averaged 16·02 ft.-lb. per sq. ft. Table XXI indicates that the efficiency of the coarse and fine grinding mills, in terms of the useful available power, was 24·7 per cent and 22·0 per cent respectively.

During the foregoing tests, measurements were taken to determine the amount of heat generated during grinding.

The heat developed in each of the mills is shown in Tables XXII and XXIII.

TABLE XXIII

Tube-Mill	*Power Input*		*Heat generated B.T.U. per minute*	*Power Equivalent of Heat Produced*	
	Total h.p.	*Available h.p.*		*H.p.*	*Per cent of available h.p.*
Durban Deep Primary ...	248·1	222·0	7,330	172·8	77·8
Durban Deep Secondary...	261·7	228·2	7,758	182·9	80·1
Crown Mines Secondary...	185·8	158·7	6,205	146·3	92·2
Average....................	231·9	203·0	7,098	167·3	82·4

Tables XXII and XXIII disclose that, of the 203 h.p. available to perform useful work, 167·3 h.p., equivalent to 82·4 per cent, were registered as heat generated during grinding.

Discussion of Large-Scale Experiments

These large-scale experiments indicate the following efficiencies for crushing and grinding machines, based on the power available to achieve comminution :

Gyratory crusher	=	32·8 per cent
Stamps	=	34·7 ,,
Cylindrical mills	=	23·4 ,,

The tests also disclose that over 80 per cent of the power available to achieve comminution is converted into heat.

The tests add support to the generally accepted belief that the net energy input is directly proportional to new surface produced.

Conclusion

In conclusion, the author wishes to express his appreciation and thanks to the Director and the Staff of the Government Mechanical Laboratory at Cottesloe for willing assistance in conducting the compression and tension tests ; to Mr. J. J. P. Dolan, Mr. E. N. Johnson and the Staff of the Rand Mines Mechanical Laboratory ; and to Mr. L. Ackermann of The Central Mining and Investment Corporation Metallurgical Staff. Mr. Ackermann virtually carried out all of the small- and the large-scale tests detailed in this paper.

Bibliography

(1) Wilson, Raymond. 'A Method for Estimating the Efficiency of Pulverizers'. A.I.M.E. *Technical Publication* No. 810, February, 1937, p. 2.

(2) White, H. A. 'The Future of Gold Metallurgy'. *Journ.* Chem. Met. Min. Soc. South Africa, Vol. 43, April, 1943, p. 143.

(3) Martin, G. 'Researches on the Theory of Fine Grinding, Part III'. *Trans.* Ceramic Industry, Vol. 25, p. 66.

(4) Fahrenwald, A. W., *et al.* 'Thermal Determination of Ball Mill Efficiency'. A.I.M.E. *Technical Publication* No. 416 (1931), p. 11.

(5) Gross and Zimmerley. 'Efficiency of Grinding Mills'. U.S. Bureau of Mines Serial 2952, July, 1929, p. 17.

(6) Sharratt, E., *et al.* 'A Rapid Optical Method for estimating the Specific Surface of Powders'. *Journ.* Soc. Chemical Industry, Vol. 64, No. 3, March, 1945, p. 73.

(7) Carman, P. C. 'The Size and Surface of Fine Powders'. *Journ.* Chem. Met. Min. Soc. S.A., Vol. 39, February, 1939, p. 266.

(8) Martin, G. 'Researches on the Theory of Fine Grinding, Part II'. *Trans.* Ceramic Industry, Vol. 24, p. 51.

(9) Gross, John. 'Crushing and Grinding'. U.S. Bureau of Mines *Bulletin* 402, p. 34.

(10) Edser, E. 'The Concentration of Minerals by Flotation : Fourth Report on Colloid Chemistry'. British Association for the Advancement of Science, 1922, p. 281.

(11) Same as (2), p. 144.
(12) Same as (3), p. 66.
(13) Same as (2), p. 144.
(14) Same as (5), p. 19.
(15) Same as (9), p. 41.
(16) BOND, F. C., and MAXSON, W. L. 'Grindability and Grinding Characteristics of Ores'. A.I.M.E. *Technical Publication* No. 888, February, 1938, p. 12.
(17) STANLEY, G. H. Presidential Address to Chemical, Metallurgical and Mining Society of South Africa, Vol. 15, August, 1915, p. 29.
(18) ADAMS and COKER. 'An Investigation into the Elastic Constants of Rocks'. Carnegie Institute of Washington, June, 1906.
(19) WEISS, OSCAR. 'Experiments on the Fracturing of Witwatersrand Quartzites under Direct Pressure'. *Journ.* Chem. Met. Min. Soc. South Africa, Vol. 41, March, 1941, p. 356.
(20) WYBURGH, M. 'The Building Stones of the Union of South Africa'. Memoir No. 28 of the South African Department of Mines and Industries Geological Survey.
(21) COOK, J. 'Investigations in Ore Milling to Ascertain the Heat Developed in Crushing'. *Transactions* Institution of Mining and Metallurgy, Vol. 24 (1914–15), p. 235.

⁂ Extra copies of this paper may be obtained at a cost of 2s. 0d. each, at the office of the Institution, Salisbury House, Finsbury Circus, London, E.C. 2.

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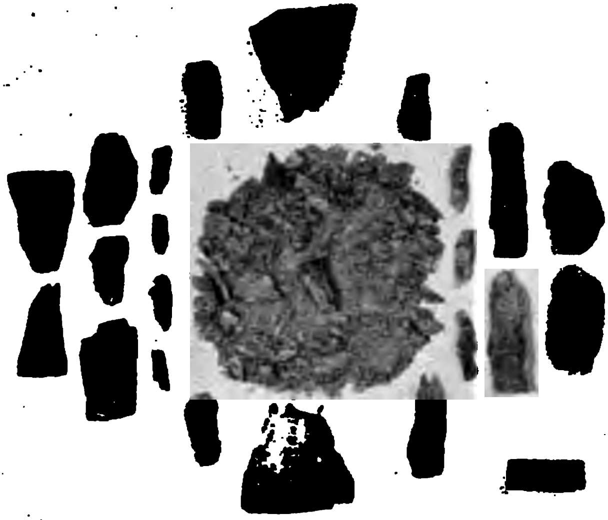

FIG. 1.

FIG. 2.

Fig. 3.

Fig. 4.

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Subject to revision.] [*A Paper issued on March 14th, 1946, 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, on Thursday, April 11th, 1946,* **at 5.30 o'clock p.m.**

A Survey of the Deeper Tin Zones in a Part of the Carn Brea Area, Cornwall.

By Brian Llewellyn, *Member*.

Introduction

Much has been written on the topic of Cornish mines and Cornish mining and many observations made, frequently contradictory. Evidence is often fragmentary and inconclusive and, in spite of the great advance in geological knowledge during the past half-century, it seems unlikely that the art of ore finding or underground prospecting will ever become an exact science, particularly in the case of tin.

Wherever possible only recorded facts and substantiated observations have been included in this paper; problems of metallogenesis and tectonics have purposely been avoided, for it is not the intention of the author to present a geological paper. On the plan a few inferences have been unavoidable in plotting the lodes at a specific horizon where very little development exists. Although the course lines of the lodes are often shown to extend for 1,000 ft. or more it is not intended to convey that mineral values will persist throughout. It is, in fact, exceptional for this to be the case with Cornish lodes, in which even the main leader may disappear for several fathoms. To the best of the author's recollection, no such ambitious attempt has hitherto been essayed and certainly no claim to finality is made by him.

The preparation of the statement of copper and tin production (Table II, Plate I) in the area presented numerous difficulties. Such records of output as were available were in many cases known to be incomplete, particularly where working dated back more than 100 years and sales records were not available. In such instances certain assumptions were necessary in relation to

FIG. 1. Sketch plan of the mines of the Camborne-Redruth area, after Spargo, 1865. (References opposite.)

Reference to mines shown in Fig. 1 above :

*1 Angarrack Consols
*2 Rosewarne & Herland
*3 South Herland
*4 West Rosewarne United
*5 North Unity
*6 Lanin
*7 South Rosewarne
8 Wheal Unity
*9 Clowance
*10 Wheal Treasury
*11 Wheal Annie
12 Rosewarne United
12A New Rosewarne
13 North Rosewarne
*14 West Dolcoath
15 East Rosewarne
*16 West Tolcarne
*17 West Hender
*18 Trevoole
*19 Gernick
20 Wheal Nelson
21 North Dolcoath
22 Wheal Hartley
23 Crane Mine
*24 West Roskear
*25 Portbello
26 South Seton
27A New Seton
27 West Seton
28 North Roskear
28A Roskearnoweth
*29 Gustavus
*30 Pendarves Consols
31 Dolcoath
32 Stray Park
33 Camborne Vean
34 Wheal Francis
35 West Stray Park
36 Wheal Tryphœna
36A West Condurrow
37 Tolcarne
38 Wheal Harriet
38A Condurrow
38B Carn Camborne
39 South Condurrow
40 Wheal Grenville
40A East Grenville
*41 Bollenowe
*42 Forest Mine
43 Buller & Basset United
44 South Wheal Frances
45 West Frances
46 West Basset
47 North Frances
48 South Dolcoath & Carnarthen Consols
49 Tincroft
49A Illogan Mines
50 Cook's Kitchen
50A South Crofty
50B Crofty
51 North Crofty
51A East Seton
52 Wheal Seton
53 Emily Henrietta
53A Ungranted
*54 East Seton & Maude
55 West Wheal Tolgus
55A North Pool
56 Wheal Agar
56A Tehidy Mine
57 East Pool
58 Carnbrea Mines
59 South Carnbrea
60 North Wheal Basset
61 Wheal Basset
62 South Wheal Basset
62A Ungranted
63 South Buller & West Penstruthal
*64 Carvannel
*65 Penstruthal
66 & 67 South Buller
68 Wheal Buller
69 Copper Hill
70 East Basset
71 Clyjah & Wentworth
72 North Buller
73 East Carn Brea
74 Wheal Uny
75 Wheal Union
76 Great South Tolgus
77 South Tolgus
*78 Old Tolgus
*79 Old Tolgus United
80 Great North Tolgus
*81 Wheal Mary
82 New Treleigh
*83 Wheal Harmony, Wheal Montague & Cardrew
84 East Tolgus
85 Pednandrea
86 Trefusis & East Trefusis
87 Pennance Consols
*88 Bell Vean
*89 Comford
*90 Tresavean
*91 Wheal Brewer
*92 West Trethellan
*93 Trethellan
*94 Wheal Edgcumbe
*96 Tingtang Consols
*96A Wheal Moyle
97 Wheal Damsel
*98 Cathedral
*99 North Damsel
*100 South Gorland
101A Grambler & St. Aubyn
101 North Grambler
102 Wheal Cupid
103 Old Treskerby &c
103A Treskerby
104 Great North Downs
104A Wheal Rose
104B Great Briggan
105 North Downs
*106 Unity Woods
*106A Killifreth
106B Great Wheal Busy
*107 Wheal Pink
108 St. Day United
108A Creegbrawse
*109 Wheal Jewell
110A Wheal Damsel
*110B East Damsel
*110 Carharrack
111, 112 Clifford
113, 114 Amalgamated Mines
*115 Wheal Squire

Mines marked thus * were idle in 1865.

TABLE I

	Roskear Dolcoath	*South Crofty Tincroft*	*North Pool East Pool and Agar Carn Brea*	*Tolgus Tehidy*	*Direction of Dip*
(1)	?	Fane's	Dauhuz	South Tolgus	North
(2)	?	Trevenson	Trembath	Trembath	North
(3)	North Roskear Main	South, No. 5 North	No. 4 North	—	South
(4)	Complex	Cherry Garden No. 6 North	?	—	North
(5)	—	No. 4 North	No. 4 North	—	South
(6)	?	Longclose Main	Rogers	'Tin Lode' Great South Tolgus	North
(7)	South Roskear Main	?	—	—	North
(8)	—	No. 3 North	—	—	North
(9)	?	No. 2 North	Middle	—	North
(10)	?	No. 1 North	No. 2 South	—	South
(11)	South	?	New North	—	South
(12)	New South	Main	Great Barncoose	Turnpike	North
(13)	?	Pryce's	Engine	?	South
(14)	Dolcoath Main	Highburrow	Highburrow	?	South

Lodes

The 'right-running' lodes vary in strike from N. 50° E. to N. 70° E., true bearing, as compared to the average strike of the elvans, which is N. 58° E. The strike of the caunter-lode series varies from N. 80° E. to N. 85° E. The dip of the majority of lodes is over 70° in the deeper levels, with occasional exceptions of 60° and even 50°—e.g., 'North Pool' lode. The average lode width as exposed by development in recent years has been about 4 ft., although widths of 10 ft. and over were often recorded in South Crofty mine in the past, notably on the upper levels of the Middle lode, whilst sections of the Main lode of Dolcoath were over 20 ft. in width, and stope widths greater than 40 ft. have been recorded in that mine.

Fourteen main 'East and West' lodes appear to traverse the area at the horizon specified, although probably six only persist throughout, and these, in order from north to south, are set out in Table I.

The four principal caunter lodes are, from north to south: Tehidy caunter, Fortune caunter, Reeve's lode (also called 'Old Tom's'), and Longclose caunter. Reeve's lode is by far the most important of the four and can be traced almost through the area, although information in depth to the north-west is lacking.

Mineralization of Lodes

The normal downward succession of economic minerals in the area is, roughly: Copper and arsenic ores in the killas from surface to the upper margin of the granite; wolfram from the granite-killas contact to 60 fathoms below (although occasionally deeper); tin ore from the contact to an average depth of 200 fathoms below the granite. Tin generally persists to a greater depth in the north-dipping than in the south-dipping lodes. The most important accessory minerals are: Quartz, tourmaline, fluorspar, chlorite, iron pyrites, and hæmatite, with, more rarely, blende, siderite, and calcite.

Many instances have been recorded of the occurrence of tin ore above copper, which has again given place to tin, and the author believes that in a large number of such cases the explanation is provided by the leaching of copper from the upper parts of lodes, since denuded, and its re-deposition in a secondary sulphide form. Many instances also occur in which a copper lode has not persisted downwards as a tin lode and frequently there has been inadequate search to provide conclusive evidence as to the cause. Nevertheless, the general law of succession in the district is as above. and the author contends that insufficient exploration has been done in recent years to confute its general applicability. Apparent exceptions may often be ascribed to one of the following causes :—

(*a*) The transition or copper-tin zone is frequently unpayable and sinking is stopped through lack of funds.

(*b*) The lode has been disturbed by faulting or by the influence of an elvan.

That competent observer, Brenton Symons, writing in 1884, at a time when a large number of mines were still active, recorded :

> Like the granite-loving tin, copper ores are occasionally deposited outside the strata it most affects, but though rich bunches have been, as at Tresavean and Penstruthal, worked in the granite, yet a copper lode entering that rock generally changes to tin, nor do cupreous ores ever reach the depth obtained by stannic oxide. The sulphuretted ores of copper are found in connection with altered rocks and prevail in the greenstone near the junction, notably in Carn Brea and Gwennap districts, and in the greenstone skirting the Western coast of the Land's End district, whilst the yellow ores occur mostly in slates more distant.

In 1918, an eminent mining geologist, Dr. Malcolm Maclaren, who made an exhaustive survey of this district, observed :

> Since in the Camborne-Redruth area copper lodes in the killas and greenstone are normally continued downwards as tin lodes in the granite, the question of depth of the underlying granite beneath the killas is naturally one of prime importance.

Productiveness of Lodes

The north-dipping lodes are commonly stronger in character than those dipping to the south, and the latter are never strong in their upper levels in the killas ; in consequence their outcrop is difficult to trace wherever the killas is deep and they must, therefore, be prospected in the granite. North-dipping lodes which are strong in the granite tend to form branches on emergence into the killas and tin ores rarely persist upwards to any distance in the killas, except in proximity to an elvan.

South-dipping lodes frequently have a ' linked ' character and consist of two or more ramified sections, often connected by caunter-running spurs. They are accordingly difficult to follow. Their walls are irregular ; so is their mineralization.

Caunter lodes are generally pinched and ill-defined in the granite and their mineral contents are irregularlydi stributed, some sections appearing to be quite barren of economic minerals.

Junctions of lodes are important, especially those at a small acute angle, because not only is the width of a lode usually augmented but also its ore content. Occasionally these lodes coalesce, as at South Roskear mine, where the Main, Caunter, and Roberts lodes formed a junction and continued together for 300 fathoms, producing the rich copper deposits for which Old Roskear was famous.

Elvans, which usually dip northwards at angles between 60° and 45°, have commonly a beneficial influence on tin lodes, although the intersection of a lode with an elvan has the usual effect of splitting the former into numerous stringers which are difficult to follow.

Greenstones, which are mostly altered dolerite or diabase containing secondary pyroxene, have in the past exercised an important influence on the productiveness of lodes and, in fact, on all mining activities in the northern part of this area. Few lodes wholly in the greenstone were profitable and these were exceptionally strong copper-bearing lodes in regions abounding in actinolite, which is said to have had a beneficial influence on copper. Certainly no rich tin lodes have persisted in the greenstone, from which it would seem that those lodes which have been worked profitably in the past down to the greenstone at a comparatively shallow depth are certainly worth prospecting at a horizon below such influence. In this area the mines affected in this manner are North and South Roskear, Wheal Crofty, North Crofty, South Crofty, North Pool, West Tolgus, South Tolgus, and Great South Tolgus.

Extension in Depth

Dr. Malcolm Maclaren has ably epitomized the common experience of mines in this area and the author cannot do better than quote from his comprehensive report of 1918 to Messrs. Bewick, Moreing and Company:

> The whole experience so far gained from the deeper workings of the Cornish Mines in this neighbourhood points in one direction only, viz. :— that there is a limit to the depth of tin ore bodies in the granite. The strongest ore body ever known in Cornwall, that of Dolcoath, has failed at the 510-fm. level, but the average depth below adit of all the other ore-bodies along the Dolcoath-Highburrow Channel was much less than that of the great Dolcoath ore shoot. Few of the others reached 360 fathoms below adit ; it is true that some were carried below that level, but with one exception above-mentioned (viz. : Dolcoath) profitable mining ceased above 360 fathoms. This represents an average depth below the granite-killas contact of some 200 fathoms and that is, in my opinion, the greatest depth we are justified in expecting tin ore. Should it be obtained at greater depth, it should then be regarded as an unexpected stroke of good fortune.

It may be contended that in the case of an exceptionally strong lode channel at the bottom of an operating mine it is good policy to continue downward development in the search for such 'good fortune'. There are, however, many excellent prospects of lateral extension in this area at favourable depths, which can be gauged by a study of the configuration of the granite-killas contour lines. Much unprofitable exploration could be saved thereby and there seems to be no geological reason why the lives of such mines as South Crofty, which has deepened its mine at the remarkably low average rate of *9 ft. a year* during the past 30 years, should not be prolonged almost indefinitely.

By reference to Table II, Plate I, a mental picture may be drawn of the tin-ore possibilities in depth of those mines which have formerly been worked in the shallower zones only. There seem to be good grounds for supposing that tin will take the place of copper in depth and in an average proportion approximately equal to that of the deeper zone mines, in terms of comparative ore values.

On this assumption, if the first ratio of Table II, Plate I, be applied to the second group of mines, the corresponding value of the tin awaiting exploration is then £3,139,176 × 4·55, which equals £14,283,250, *less* the value already extracted—viz.: £320,078; equal in round figures to £13,963,000. At present tin prices this value becomes nearly £51,000,000 sterling.

History

Within the limits of a short paper the author does not propose to give a detailed account of the history of mines within the area and confines himself to the following condensed observations. (See also Table II, Plate I).

To the south the mines nearest to the granite outcrop, whose lodes would therefore be normally exhausted at a comparatively shallow depth below surface—viz.: Dolcoath, Cook's Kitchen, Tincroft, and Carn Brea—may be assumed to be worked out. It is true that possibilities still remain in Tincroft in the less important lodes at deeper levels, also in the easterly extension of the Highburrow lode in Carn Brea sett, but it is doubtful if either of these mines would now be worth unwatering.

To the north-west the granite lies at a considerable depth below surface, far deeper than was expected in 1925, when the sinking of the New Roskear shaft was commenced with the intention of exploring in depth the highly profitable North and South Roskear lodes. Most of the lodes opened up from the cross-cuts at the foot of this shaft were above the contact and were irregular in the killas, as might be expected. The discoveries made may be regarded as most encouraging, nevertheless, whilst the two lodes explored to the south in the granite were distinctly promising. The sole reason for ceasing exploration was the premature exhaustion of capital, due to prior under-estimation of the depth of the granite and the thickness of the greenstone.

To the north-east also the granite is exceedingly deep, this depth surpassing all expectations (and especially those of Dr. Malcolm Maclaren, the consulting geologist) when the sinking of the Tolgus New shaft was undertaken in 1923 for the purpose of exploring the easterly extension of the East Pool Great lode and other lodes. A few tin and arsenic values were obtained by diamond drilling but, generally speaking, the main value of the latter lay in the geological information obtained *and the work was not done at a favourable horizon.* This was realized by the company's engineers, when they advised the abandonment of the project after the expenditure of a large sum of money.

The history of all mines in the north of the area is very similar. They were mostly profitable copper-producers in the shallow levels in killas, and they were nearly all abandoned when they reached the upper greenstone sill at a depth of less than 100 fathoms. Some had the courage to penetrate the upper sill, only to be again baulked by the lower sill. It may fairly be said that the northern area has never been prospected in depth.

The centre of the area is at present being worked by two old-established mines—viz., South Crofty and East Pool and Agar. In the earlier days of South Crofty the lode most productive in tin was the Main (or Middle) lode, with important additions from the south-dipping No. 1 North and Pryce's lodes. These lodes can now be said to be almost worked out within the boundaries of the South Crofty sett.

The East Pool and Agar mines enjoyed many years of prosperity, even after the disastrous collapse of the two East Pool hoisting shafts in May, 1921. The principal lodes worked were the South and Great lodes, Engine lode, New North lode, and Rogers lode, the latter forming the mainstay of the mine after the sinking of the Taylor shaft in 1922. The easterly exploration of the Rogers lode, confused as it was by elvans and cross-courses, was neglected when the Moreing lode showed early signs of promise and the even more promising Tolgus Tunnel discovery had to be abandoned owing to pumping difficulties. Although some profitable ore was disclosed in the No. 4 North and other lodes, generally the policy of lateral exploration was not vigorously pursued after the failure to disclose any important lodes in the 1,600-ft. North cross-cut from Taylor shaft, for reasons which are not quite clear but which probably included shortage of funds. Preparations were made in 1941 to abandon the mine, the remaining ore blocks and pillars of which were stoped out, whilst development was suspended until the East Pool and Agar company was enabled to prevent flooding in 1942 during a time of national emergency and mining operations were re-commenced. With no advanced development, it was inevitable that the milling grade should fall and that the cost of tin production should then be uneconomic.

Tin Ore Zones

If a line be drawn on the plan (Plate II) to the south of the 1,900-ft. granite-killas contour, at a distance of 2,600 ft., which is the horizontal equivalent of a height of 200 fathoms or 1,200 ft., at a slope angle of 25° (estimated), then it will be seen that the area enclosed between the two lines represents the horizontal projection of a triangular prism, within which lies the favourable tin ground underlying the killas above that particular horizon. On either side of this zone will lie narrower zones where tin ore occurrences are likely to be irregular.

The interval between the deeper levels of the South Crofty mine is now 25 fathoms, or 150 ft., and it will be seen that for every 150 ft. increase in depth, the favourable depth zone will be northwards approximately 330 ft. For example, a lode which is approaching the lower limit of its payability at, say, the 310-fm. level, would be quite unpayable at 335 fathoms and its place would be taken by another lode, at least 330 ft. to the north, which is in the corresponding depth zone.

The limits stated are not strictly correct, for the upper limit of the tin zone does not quite coincide with the plane of the granite surface and is controlled by the temperature gradient. For all practical purposes, however, they may be accepted.

An inspection of the plan will now show clearly that nearly one-third of the line of the central lodes of South Crofty and East Pool and Agar—e.g., Main and Great lodes—between the Dolcoath–Cook's Kitchen Great cross-course and the Portreath Railway cross-course, lie too deep in the granite to warrant much further expectation of profitable ore above the 1,900-ft. horizon. The remainder lies close to the limiting line. It will also go far to explain the absence of outstanding values in recent development of the deeper levels of East Pool and Agar, notably in the south-dipping lodes, which have less persistence in depth.

That large areas of excellent promise remain to be explored will be obvious, however, and the author has marked these A, B, C, D, and E on the plan.

Zone A.—This lies east of the Great cross-course and within the old setts of Wheal Crofty, Longclose, and North Crofty. This zone could best be explored by cross-cuts in the deeper levels of New Cook's Kitchen shaft. There are many lodes awaiting discovery, some of which appear to have flat intersections, and the past records of the old mines working the upper levels of these lodes are outstanding.

Zone B.—West of the Great cross-course and within the old Dolcoath and South Roskear setts. Two lodes have already been partly developed, with encouraging results, from the New Roskear shaft, and it is probable that at least two more exist within the granite margin.

Zone C.—On either side of the South Crofty–East Pool boundary and north of existing development points. In advancing west the East Pool 1,900-ft. West drive on No. 4 North lode is getting shallower in the granite and values may be expected to improve. The presence of the Trembath lode, 10 ft. to 12 ft. in width and carrying good tin values higher up, is also suspected in this zone. These lodes can be developed from either South Crofty or East Pool.

Zone D.—Beyond the Portreath Railway cross-course east of the Taylor shaft of East Pool and Agar ; this ground is believed to be much disturbed and is heavily water-bearing, but there appear to be good chances here, not only of picking up the extension of the Great lode, to which the spectacular values exposed in the Tolgus Tunnel are ascribed, but that of the Rogers and other important lodes. This zone is perhaps somewhat limited in extent, but seems to be well worth exploration. Direct entry could be made by a central tunnel from the bottom of Taylor shaft, driven to cut and follow either the Rogers or Great South Tolgus lodes, with north and south cross-cuts. This zone is somewhat inaccessible at present and cannot be properly prospected by bore-holes.

Zone E.—This is on the northern limit of the area and cannot be properly prospected from East Pool and Agar, except by bore-hole. The Daubuz lode, with its eastern extension (South Tolgus), was rich in copper in the upper levels, although little is known of its western extension (Fane's lode). It seems doubtful whether the long bore-hole from the end of the 1,600-ft. cross-cut north of Taylor shaft was put out far enough to reach this lode. This bore-hole might well be extended and two additional holes put out on either side. (See also Plate II).

Factors in Exploration

In order to explore in depth the mineral possibilities of this area with the highest efficiency and, therefore, at lowest cost, it would be desirable that the two operating mines—viz., South Crofty and East Pool and Agar—should first be amalgamated ; or, at least, that they should work under some central authority which would direct the development policy.

It is vital that the pumping capacity of these mines should be considerably increased in order to avoid any recurrence of ceasing development of a wet (and probably profitable) lode or stopping work after cutting an inflow of water in a cross-course, owing to the danger of flooding the mine.

Ventilation should be so laid out that there would be a continuous air flow along selected main drives and between distant shafts, and the ventilation plan should be kept up to date.

Suggested Plan of Exploration

The first step in an exploration programme would be the holing of the 315-fm. West drive on No. 2 North lode to the New Roskear shaft, after first unwatering by an independent pump.

The next step would be to drive a through connection on one of the northern lodes at a favourable horizon. The No. 4 North lode of South Crofty at Robinson's shaft 290-fm. level is suggested and holings could be made between the 315-fm. North cross-cut from New Cook's Kitchen shaft and the 290-fm. North cross-cut from Robinson's ; also between the latter and the 1,600-ft. North cross-cut from Taylor shaft, *via* rises.

From this main air base cross-cuts and bore-holes could then be put out northwards at intervals as indicated by exploration and the development net expanded, aided by trial rises and winzes.

If valuable ore-shoots were exposed, main winzes would be sunk on one of the northern lodes and inter-connected. Whenever sufficient information had been accumulated in regard to the slope of the granite surface north of the area boundary and sufficient encouragement existed to warrant a more direct entry into depth, the sinking of a new vertical circular shaft could be considered. This might be 2,500 ft. to 3,000 ft. in depth and be located to strike the boss of the granite at its apical position.

Conclusion

A study of the 25-in. plan, together with the foregoing notes, shows that an extensive mineralized area remains which warrants exploration. The potential value of tin ore alone is around £50,000,000 in this area.

Many lodes traverse the area and no survey of national mineral resources would be complete without further knowledge of the deep zones to the north of the area.

The area has been so little explored that diamond drilling alone would not provide sufficient information and exploration by means of drivages is therefore required. The district has been so thoroughly prospected at surface in the past that it would be unlikely that new discoveries at surface would give rise to new mines: hence, if the existing mines were allowed to flood, it is probable that the mineral resources in the area above referred to would be permanently lost.

Acknowledgments.—The author is alone responsible for the facts and opinions given in this paper, but it should be mentioned that a great deal of the work involved in its preparation was carried out for the Non-Ferrous Mineral Development Control of the Ministry of Supply. His thanks are due to the Controller, Sir William J. Larke, and to the Ministry of Supply for permission to publish the paper, and to Messrs. Bewick, Moreing & Co., C. V. Paull, Thomas Pryor, and W. C. C. Rose and his many friends in Cornwall and elsewhere for facilities and help afforded during the course of his work.

Selected Bibliography

Thomas, R. Arthur, and Bewick, Moreing & Co. 'Dolcoath's Future'. *Min. Mag.*, January, 1921.

Collins, J. H. *Trans.* Roy. Geol. Soc. Corn., Vol. XIV (1912).

Davey, J. C. 'The Geology of Tolgus Shaft'. *Trans.* Roy. Geol. Soc. Corn., Vol. XV (1925).

Davison, E. H. 'The Geological Basis for the Estimation of Cornish Mineral Areas'. *Trans.* Corn. Inst. Eng., Vol. X (1925).

Ibid. 'Geology of First Part of Roskear Shaft'. *Min. Mag.*, June, 1925.

Ibid. 'Geology of Roskear Section Dolcoath Mine'. *Min. Mag.*, March, 1929.

Dewey, H. 'Copper Ores of Cornwall and Devon'. Memoir Geol. Survey, 1923.

Gregory, Maurice. 'The New Dolcoath'. *Min. Mag.*, November, 1923.

Henwood, W. J. *Trans.* Roy. Geol. Soc. Corn., Vols. V (1843) and VIII (1871).

Jennings, J. 'The Heaves of the Great Lode of East Pool and Agar'. *Trans.* Roy. Geol. Soc. Corn., Vol. XIII (1911).

MacAlister, D. A. 'Geology of Falmouth and Camborne'. Memoir Geol. Survey, 1906.

Maclaren, Malcolm. 'The Geology of the East Pool Mine'. *Min. Mag.*, May, 1917.

Ibid. 'Report on the Setts held by the East Pool and Agar Co. Ltd.' (to Bewick, Moreing & Co.), July, 1918.

Salmon, H. C. 'North Crofty Mine, and The Seton Mining District'. *Mining and Smelting Magazine*, Vol. II (1862).

SPARGO, THOMAS. 'Statistics and Observations on the Mines of Devon and Cornwall', 1865.

STEPHENS, F. J. 'Redruth, North Gwennap, and Illogan'. *Min. Mag.*, August, 1939.

SYMONS, BRENTON. 'Sketch of the Geology of Cornwall', 1884.

APPENDIX

EXTRACT FROM *Mining and Smelting Magazine*, VOLUME II, 1862: H. C. SALMON (EDITOR)

NORTH CROFTY

The ground was originally worked as Trevenson Mine, after which it was amalgamated with Long Close and Dudnance Mines, to the south, and worked as East Crofty. A few years ago the ground was again divided in North Crofty and South Crofty—the former, lying to the north of the turn-pike road, comprising the old Trevenson ground; and the latter, lying to the south of the road (adjoining Tincroft and Cook's Kitchen), including the Long Close and Dudnance ground.

The most important lode in this sett is Reeve's Lode, bearing about 25° N. of W. magnetic (or 3° N. true W.), and underlying north from 18 inches to 2 feet per fathom. The other important lodes, to the south of Reeve's, are the Trevenson or Engine lode, bearing about W. 2° N., and underlying north 2¼ feet per fathom, which as hereafter stated, falls into Reeve's lode going east; and the Cherry Garden lode, bearing some degrees to the south of west, and underlying south. Above the 10-fm. level, the Trevenson and Cherry Garden lodes are together, and have the same back upwards—at that level they diverge in opposite underlays, the Trevenson dipping north, and Cherry Garden south. To the north of Reeve's lode is Fane's lode, bearing W. 8° S., and underlying north.

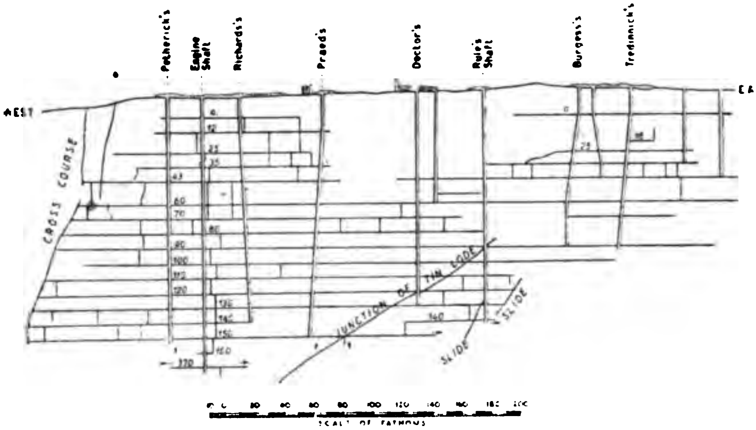

FIG. 2.

The accompanying section (Fig. 2) shows the workings on Reeve's lode. In the eastern part of the mine, which was originally worked as Pool Mine, that is, east of Rule's shaft, the lode made considerable deposits of copper above the 43. Between Rule's and Doctor's, the ore made to the 70; between Doctor's and Praed's, it made to the 80; and west of Praed's, the lode was productive to the 100 and even deeper. This dip of the ore is to be accounted for by the junction of the lode with an elvan course, bearing about 12° S. of W., and underlying north 6 feet in a fathom, or at angle of 45°. This elvan course, which is also met with in North Roskear Mine to the west, consequently intersects Reeve's lode at an angle of 37°, and overtakes it in the underlie. The point of intersection between the lode and elvan at surface is in the extreme eastern part of the workings, about Pool Village; but coming west this deepens rapidly, passing through Rule's shaft about the 43, but not being met with in the 60 until about 15 fathoms west of Doctor's, and not at the Engine shaft until about the 150. The copper ore deposits on Reeve's lode evidently made *above* this elvan, and consequently dipped with it. The Trevenson (or Engine) lode makes, it will be seen, an angle of about 23° with Reeve's lode, and (lying to the south) also overtakes it in the underlie. The Engine shaft and Praed's shaft shown in the section, are really sunk from surface on this lode, but the latter comes into Reeve's lode about the 70, although the Engine shaft does not come into it until a little above the 170, although below the 100, the Engine lode dwindles to nothing: Rule's and Doctor's shafts are entirely on Reeve's lode. The Engine shaft is sunk perpendicular to the 12, and below that on the north underlie of the Engine lode to its intersection with Reeve's at the 170. The distances east from this shaft to the point of junction of the two lodes, are as follows, at the levels given: At the 60, 88 fathoms; at the 70, 66 fathoms; at the 80, 52 fathoms; at the 90, 48 fathoms; at the 100, 38 fathoms; and at the 110, 18 fathoms.

Cherry Garden lode has made considerable returns, both above and below the elvan (which intersects it much shallower than it does the lodes to the north), but principally below it. The productiveness of this lode, however, was never anything to be compared with the others.

Compared with the great lodes of the district, Reeve's lode, bearing as it does so much to the north of west, is a strong caunter. Yet it is a lode of great strength, and through its run has been very productive. In the eastern part of North Crofty, it made a magnificent deposit of copper at the old Pool Mine, under the village of that name; and westward, it was also extremely productive at Wheal Crofty, now in North Roskear sett, where it is called the Great Caunter. This lode also goes into Wheal Seton, and altogether it is the main one of this run. East of Rule's shaft, the back of Reeve's lode is in South Crofty sett, but as it underlies north, the deeper parts are in North Crofty sett.

Under the elvan, Reeve's lode thus became poor, so much so, that, although the mine had been sunk by the old party to below the 170, the two bottom levels were abandoned and the water let into the 150, about 14 years ago. It is only within the last month that this has been forked, and the 170 ends commenced driving. The 170 west is now driving by 4 men at 7*l*. per fathom, in a lode from 4 to 6 feet wide, turning out 150 sacks of work per fathom, worth 3 cwt. of tin per 100 sacks, giving 4½ cwt. of tin per fathom. The 170 west is driving by 4 men and 4 boys at 8*l*. per fathom; the lode here is small and poor, but the end is being pushed on to get under the eastern tin ground, to reach which there is about 40 fathoms of ground to drive.

The eastern tin ground in North Crofty lies about Rule's and Doctor's shaft. The best end in the mine at present is the 150 east, which is now being driven by 6 men at 7*l*. 10*s*. per fathom. The lode is not large—not exceeding 1 foot—but it produces very rich work, varying from 10 to 12, and 15 cwt. per 100 sacks, up to as high as 22 per 100 for some journeys. The lode produces about 75 sacks per fathom, and for the last 4 fathoms has turned out 2¼ tons of tin, which gives a produce of 15 cwt. per 100 sacks, or 7½ per cent—very rich work. The character of the lode here is also good, although it is small for the district. At the back of 50, to the west of this end, a stope is working by 4 men at 3*l*. 5*s*. per fathom, on a lode from 3 to 4 ft. wide, producing work worth 6 cwt. per 100, giving a lode worth about 30*l*. per fathom.

Above this level, the 140 end east is being driven by 4 men at 8*l*. per fathom, on a lode which is now poor and disordered, but has been worth from 10*l*. to 15*l*. per fathom. Under the 120, several stopes are working on a lode producing coarse work for tin. Below the 150, east of Praed's shaft, a winze is sinking by 4 men at 8*l*., in a lode producing a little copper and tin: winze now down between 2 and 3 fathoms.

These are the main points of operation at present at North Crofty. The driving of the 150 east to the slide will be an interesting point: hitherto, in the bottom levels, that is below the 110 where the slide is first seen, the lode has not been traced beyond the slide, which so disorders the ground as to render its recognition afterwards impossible. The driving of the 170 east also, to come up under the eastern run of tin ground is also a point of much importance. It may fairly be doubted, however, whether this run of tin ground in the 150 east is really so important as that met with in the 170 west, where the lode is so much larger, and where the tin seems to continue in the dip of the copper in the upper levels.

Whether, under the rich deposits of copper long since worked away, North Crofty will, like her neighbours to the south, make a profitable mine in depth for tin is a matter upon which as yet it is scarcely possible to speak positively. Tin which would pay to come away was first met with 3 fathoms above the 110, and in that level it made for 50 fathoms long, low quality work. But considering altogether the character of the lode; and taking also into account the recent discovery of granite at East Pool, immediately to the east of North Crofty, which shows us that we may expect to meet with that rock at a much less depth than we could before have anticipated; the prospects seem decidedly in favour of the mine. Increased depth, however, is probably required, before permanently remunerative deposits of tin are met with, and this will take time; but with time, I can see no reason why North Crofty shall not make a deep tin mine. The first step towards prosecuting the mine in depth has just been taken in forking the water from the 170, and the success which has attended this—a good lode being found in 6 ft. driving west—should be an encouragement for further explorations in depth. In this district, tin has only been met with in valuable quantities under the rich copper deposits by sinking through a considerable depth of comparatively poor and unproductive ground; and the key to success seems to be the necessary courage to do this. It may be well to remark that there is probably another elvan—the South Roskear elvan—to be met with in North Crofty *in depth*.

North Crofty is under the management of Captain Joseph Vivian, so well known as the manager of the adjoining mine of North Roskear. Since he has been here Captain Vivian has had up-hill work, for he took the mine when abandoned by the late adventurers in East Crofty, principally for the sake of North Roskear. From a concern deemed hopeless, even by the lords and their agents, for they were the principal adventurers, North Crofty has now been worked into the position of a favourite progressive mine. The other agents are Captain William Thomas and Captain George Bennetts, the purser being Mr. Almond E. Paull of Camborne.

When Reeve's lode is not productive it generally partakes of a flucany character. It also often makes a flucan part parallel to the main part of the lode, or separated from it by a "horse" of ground. Thus in the eastern part of the mine the tin ground below the 110 has made in connection with, or rather parallel to, a flucan of this kind.

⁎ *Extra copies of this paper may be obtained at a cost of 1s. 9d. each, at the office of the Institution, Salisbury House, Finsbury Circus, London, E.C. 2.*

CROWTHER & GOODMAN (*Branch of The Argus Press Ltd.*), 2, Carmelite St., London, E.C.4.

BRIAN I

(I) *Mi*

Cook's E

Tincroft.

Tincroft

Tot

(II) *Mi*

Plate II.

THE INSTITUTION OF MINING AND METALLURGY

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FOURTH ORDINARY GENERAL MEETING

OF THE

FIFTY-FIFTH SESSION

Held in the Rooms of the Geological Society, Burlington House, Piccadilly, W.1.

ON

Thursday, January 17th, 1946.

THE LATE PRESIDENT

Dr. J. G. Lawn, Hon. Treasurer, who took the Chair at the opening of the proceedings, referred in terms of great regret to the death of the President, Colonel Edgar Pam. He was a man, he said, who had occupied many positions of eminence and had fulfilled them all with capacity and success. In the last war he was prominent in the Royal Engineers and in this war he worked very hard and long in various sections of the war effort. From the point of view of the Institution his death was a great loss. He was keenly interested in its progress and full of ideas for its future. They would miss him very greatly. In tribute to his memory and as an expression of sympathy with his family, he asked those present to stand for a few moments in silence, and this was done.

Election of New President

Dr. Lawn then announced that **Mr. G. F. Laycock,** President-Elect, who had for many years served on the Council of the Institution and had also been Acting President during Colonel Pam's long illness, had been appointed President for the remainder of the current session. He was sure that the Institution would flourish under his presidency and that he would be a great asset to them.

Mr. Laycock was then inducted into the Chair. He said how much he appreciated the high honour conferred upon him by the Council and that with their guidance and the assistance of the very capable secretariat and staff he would do his best to maintain the prestige of the Institution and the high standard set by previous occupants of the Chair. He fully realized that this would be no easy task. His only regret was that it should have become necessary for him to take office somewhat ahead of the usual date owing to the great loss they had all suffered in the death of Col. Pam.

He announced that in future it was hoped to hold meetings monthly instead of bi-monthly, but that owing to paper still being strictly rationed it would not yet be possible to publish the *Bulletin* every month.

DISCUSSION

ON

Notes on the Development of the Blyvooruitzicht Gold Mining Co., Ltd., South Africa.

By A. Savile Davis, *Member.*

Dr. F. E. Keep introduced the paper in the absence of the author. He said that Mr. Davis had made a comprehensive survey of the operations at this property since the inception of the Blyvooruitzicht Gold Mining Company in 1937. Mr. Savile Davis was appointed manager of the company in 1940 and accordingly had a very intimate knowledge of the difficulties which had been encountered and the victories which had been won during the major portion of the company's existence.

The author pointed out that owing to the outbreak of war it was impossible to carry out the normal procedure in the financing of a large South African mine and that capital funds only permitted the completion of No. 1 shaft to 5,042 ft. depth, the sinking of No. 2 shaft below 2,255 ft. being suspended until the profits won from the treatment of development ore were sufficient to pay for the completion of sinking. At the present time No. 2 shaft had reached a depth of 5,073 ft. and a connection on No. 6 level at 4,830 ft. below the surface had been made between the two shafts. Owing to the provisions of the South African Mines and Works Regulations stoping was not permitted in a single exit mine on account of ventilation difficulties, so that it had not been possible to treat any ore other than that won from development faces since milling was started early in 1942. It would be realized how war conditions had extended the time between the commencement of sinking and of mining, a period of eight years having elapsed, with several months' delay yet to follow, before treatment of stoped ore on any considerable scale could be expected. A normal time under Far West Rand conditions might be five or six years.

Mr. Savile Davis described the method used in shaft sinking through the 3,957 ft. of dolomite overlying the rocks of the Witwatersrand System in the No. 1 shaft area. The members of the Institution would be able to realize the amount of organization, professional skill, and anxiety which were involved in this undertaking. The author did, indeed, point out that at the start of development considerable anxiety was felt as to the possible results of intersecting a water fissure in which large quantites of water under a pressure of 1,800 lb. per square inch would be expected to occur and described the events which followed when a 1½-in. diameter diamond-drill pilot-hole cut a fissure and a flow of 50,000 gallons per hour resulted.

The experience gained during this work, freely made available as it was, should be extremely valuable to those who may have similar problems with which to contend.

It was noteworthy that wet kata-thermometer readings in the development ends, many of which were several thousand feet from No. 1 shaft, averaged 10 millicalories per square centimetre per second, such a result showing the success with which the problem of ventilating dead ends distant from the shaft had been met. Drilling and blasting practice in development, although varying little from standard methods, had evidently been carried to a high degree of efficiency, as shown by the average rate of advance of over 7 ft. per round, the use of a type of ' burn ' cut permitting an advance of 120 ft. in 12 rounds in one 8-ft. by 8-ft. end. Blasting was only permitted to take place once in 24 hours, so that the maximum length of round possible in a shift's work was the object at which one aimed. These successful results followed, as the author emphasized, from the use of the ' burn ' cut, a type of cut which was also gaining favour in other mining fields.

The high-grade of the ore developed, averaging 15 dwt. per ton over an assumed stoping width of 45 in., was surprising when it was realized that the gold-bearing reef—the Carbon Leader—was rarely above 9 in. in thickness. The reef was described by the author as

> an insignificant-looking collection of small pebbles set in a dark matrix containing carbon, pyrite, pyrrhotite, and a consistently large quantity of gold . . . A carbon seam is practically always present on the footwall parting, varying from a pencil line to ¾ in. thick, and only in this seam has visible gold been found, either spread like butter on the upper and lower carbon contacts, or as vertical threads in fractures of the soft satin-like carbon.

The speaker had written to Mr. Savile Davis and asked him could send him some specimens ; these he had received and they were passed round the meeting.

The author also discussed the stoping methods which it was tentatively intended to use and it was to be hoped that the present paper would be followed by one descriptive of the success which might be achieved by the various modifications which would be tried out. The control of the hanging-wall in deep mines was only equalled in importance by the efficiency of the ventilation arrangements and it was interesting to note that waste packing would be used extensively in this mine.

In the small 10,000-tons per month capacity reduction plant very satisfactory extraction results had been obtained. The actual figures for the year ending June 30th, 1945, were 97.88 per cent extraction with recovery of 11.115 dwt. per long ton and residues of 0.240 dwt. per ton. Only development ore was treated, with surface sorting of 18.03 per cent of waste.

In conclusion a tribute must be paid to the completeness of the picture portrayed by the author and one was, he was sure, only expressing the feelings of the members of the Institution when the hope was again expressed that Mr. Savile Davis would submit a later paper giving his experience after full-scale stoping had been in progress for some time at the mine.

Dr. J. G. Lawn congratulated Mr. Davis on a model paper—concise, orderly, and with diagrams not unduly complicated. This Blyvooruitzicht mine was really the ' wonder mine ' of the Rand ; there had never been anything like it before, with its 100 per cent payability and high values. He supposed the nearest approach from the point of view of high values was the Sub-Nigel mine, but there the values ran in shoots, with areas of low-grade ore between. The average value of the reef itself, at Blyvooruitzicht, ran to over 60 dwt. According to the last annual report the development of the previous year averaged 62 dwt. over 12 in. Of course, the ideal from a mining point of view, would be to take that 12 in. only, and get a running grade of some 60 dwt., but that was quite impossible. It was difficult to attain a very low stoping width, especially as the reef was somewhat flat, and, therefore, in calculating the ore reserve a stoping width of 45 in. had been assumed. The question of the stoping width was, however, one of great importance from the point of view of the value to be milled. The author had not dealt very fully with that point, and, he thought, wisely, as there had been hitherto no stoping in the ordinary sense of the word, and it was only by experience, perhaps after some years, that the best method could be evolved.

There was another point of view, too, which made it desirable to keep the stoping width as low as possible and that was to secure as little disturbance of the hanging-wall as possible. This was a mine which had immense quantities of water above it. The Dolomite was nearly 4,000 ft. thick; it extended for many miles and the hope was that the draining of the Dolomite could be avoided. There was no Dolomite above the mines of the Central Rand, but there were mines under the Dolomite in the Far East Rand. There in sinking shafts the great trouble was water; they did not encounter much black mud, which was the residue of the disintegration of the Dolomite by circulating water. In the Dolomite on the western end of the Rand they had this 'wad,' as they called it, which had given a great deal of trouble. The first mine to encounter difficulties was the Venterspost mine and there the story of the sinking of No. 1 shaft was a real mining epic. They got along better in sinking at Blyvooruitzicht, no doubt because they were in part prepared, but even there they did on one occasion lose the bottom of the shaft, and the water rose right up to the water level at the rate of 3 ft. a minute. In one respect, however, they had been fortunate. The Dolomite was of great thickness, and the bottom part was solid and not so liable to contain cavities and channels of water as the upper part. Further, the reef was comparatively little disturbed by faults. The author rather modestly said it was no worse than the average; the speaker would have said it was better than the average, and evidently the faults did not bring large quantites of water, unlike some of the open faults on the Rand which formed a ready means of circulation. With the help of cementation they had succeeded in keeping the workings comparatively free from water, which was a striking achievement.

It seemed to him a fortunately-placed mine, but it had been developed under great difficulties. It had suffered from shortage of native labour from the first. In the last annual report it was stated that they were not able even to keep the comparatively small mill running at full capacity or to carry out all the development which would have been possible with a plentiful supply of labour. They might hope, however, that now the mine would go on from strength to strength, and in time, as Dr. Keep had suggested, they might have a further paper on the experiences gained in stoping.

Professor S. J. Truscott said that Dr. Keep had had a very agreeable office in introducing this well-written paper. For himself he took an equal pleasure in the fact that the wide conception which the author had taken of what constituted development allowed him to mention the enterprising spirit shown both industrially and technically, as a result of which this important reef was disclosed under so great a cover of Dolomite. It was really one of the romances of mining. In the foot-wall of the Main Reef Series, he said, there was an alternation of shales and quartzites; in the hanging-wall mostly quartzites. Most of the shale beds in the foot-wall were magnetic, advantage of which was taken by magnetometric survey to get in touch with their sub-outcrops below this tremendous cover of 4,000 ft. The two technical men principally connected with this work were Dr. Krahmann and Dr. Reinecke—the former a physicist, the latter a geologist.

Having established the sequence of magnetometric anomalies in the foot-wall along the Central Rand where the beds actually outcropped, they proceeded to make magnetic traverses across the line of presumed extension to the south-west, where with delicate instruments they found themselves able to pick up the same sequence of anomalies, indicating the concealed presence of the same foot-wall shales. The whole idea was outlined to the speaker by Mr. Robert Annan in 1936, who with justified satisfaction pointed out that his company had taken options on farms for 30 or 40 miles of length, and having established the magnetometric position they were able to place bore-holes with reasonable hope to intersect the Main Reef Series, most of the holes actually striking the Series.

This boring began to the north and then proceeded south-west until this Blyvooruitzicht mine was reached, a mine which was undoubtedly, as Dr. Lawn had said, a 'wonder mine', others being there associated in the venture.

The author had touched on the origin of the gold in the Banket. The speaker's competence to say anything at all about the origin of the Banket derived from the fact that over 50 years ago he was a sampler on the Central Rand. He did not know exactly how they carried out their duties to-day, but he thought that he was one of the first technically-trained samplers doing nothing but underground sampling. His samples were brought up, reduced by his boys, sufficient sample pulp sent to the assayer, and the remainder he panned for value, entering immediately an estimate in the office sample ledger, on the basis of the panning, before the assay was made. In that ledger they had a column for mentioning the pyrite. Whether abundant or but little it was reckoned a factor to be taken into account, not only because it was thought that some gold was bound up with the pyrite, but because in panning *the gold tail* could not be got so clearly away from the pyrite as it would be from ordinary quartzite grains and the estimate accordingly was more difficult to make. In this panning they got an idea of how tenaciously this fine gold clung to the pan; and from that it was, to him, not surprising that it and pebbles occurred together in a bed.

He had also sampled other mines, including the Nigel and had a working knowledge of one of the Black Reef mines, so that altogether he had a certain competency to speak on the ground of personal observation. It was only by the assembly of observations that any complete idea of the origin could finally be obtained. The dykes on the Rand, which, criss-crossing the field, cut the ground up into irregular paddocks, disturbed the gold rather than had anything to do with its original deposition. For himself he felt quite positive that these dykes had nothing to say in the original deposition of the gold.

Mr. James Whitehouse, after congratulating the author on his excellent paper, extended his congratulations to all those who had been engaged in the development of the Blyvooruitzicht mine. To have jumped forward from Libanon 16 miles, and to have tackled the sinking of 4,000 ft. through the Dolomites, was an operation that deserved success and this had been achieved in no small measure. No other mine on the Rand had shown such consistent or such high values over the scale of development accomplished to date in this mine, nor had the cost of shaft-sinking and development ever been met from the crushing of development rock from current operations.

The figures which Mr. Davis gave for sinking were interesting. In the last 2,000 ft. of Dolomite the rate of sinking averaged 209 ft. per month, which was excellent going when they remembered the precautions that had to be taken during the whole time.

With regard to the lay-out, the description of it given in the paper was reminiscent of that at the Crown Mines, in that circular shafts were to be sunk from the bottom of the shafts from surface, and incline shafts at intervals of 3,500 ft. would handle labour and materials. Such a lay-out should prove economical and ensure adequate ventilation.

He noticed from the details given in Mr. Davis's paper that development technique had progressed in recent years and considered an average advance of 7 ft. per round most satisfactory, and that as 120 ft. advance had been achieved in 12 rounds still better speeds might be looked for in the future.

The figures for temperature rise in depth compared very favourably with those of the Central Rand.

Whilst the correlation of the Carbon Leader with the known reefs on the Rand had not yet been established, they were informed that it was generally considered to correspond to the Main Reef. This emphasis that the characteristics of the reefs and their values might change very materially and over distances such as they were considering was not surprising. Examples of these changes were already known and no doubt others would be found as more widespread development of the reefs in the Union was extended. After all, within the area of the Rand, as known before its extension westward, there were variations both in characteristics and values, particularly in the South Reef and it was not surpising, therefore, that considerable changes took place in more remote areas.

The geological information given in the paper created an urge to examine the reefs in detail, and the gold content of the North Leader as compared with the Carbon Leader was most interesting.

With regard to production, the outstanding question he thought would be the recovery of fines of high value resulting from the breaking up of the Carbon Leader. The method of stoping proposed would be best suited to the conditions and no doubt sweeping and washing down of the foot-wall would be carefully carried out. Having ensured the recovery from the mine of the gold in the fines, there should be no serious problem in the reduction plant, since they were told that the small plant now operating had a recovery of 98 per cent.

All things considered the future of the mine as a highly successful producer seemed assured and perhaps it was not too much to hope that the adjoining West Driefontein mine would be equally fortunate, so that these and other mines in the western area would, in their turn, ensure the maintenance of production on the Rand as did the East Rand years ago.

Dr. F. E. Keep wished to say that he was in entire agreement with Professor Truscott's remarks as regards both the origin of the gold and the effect of the fugitive elements accompanying the dyke intrusions in dissolving and reprecipitating some of the gold in their immediate neighbourhood.

Another remark he wished to make arose out of Mr. Whitehouse's comments on the reduction operations and percentage recovery. It must not be forgotten that with a high-grade mill head it was much easier to get a high percentage recovery, so long as the ore was not refractory. The difficulty was to get the last bit of gold from the residues. Although this 98 per cent. extraction was a very fine performance it was not perhaps quite as exceptional as one might consider at first glance. The acid test was how much gold remained unrecovered.

CONTRIBUTED REMARKS

Mr. Victor Hodgson: In this interesting paper Mr. Savile Davis has given a concise description of the opening and preliminary development of a mine which is extraordinary not only in the payability and richness of the reef, but also in the problems that it has set for mining geologists and engineers.

The use of detachable bits in drilling the 40-ft. pilot-holes in No. 1 shaft to relieve congestion in the shaft bottom is interesting. In many mines detachable bits have not yet been introduced because they cannot compete with forged bits in overall cost under the local conditions. When small shafts in hard rock are to be sunk where these circumstances apply the advantages of the detachable bit in avoiding the crowding of the shaft bottom with drill steels are worth consideration.

It is noted that the settled sludge is pumped through a separate column from the 6th.-level pump station to the 3,000-ft. station, where it is transferred to the 10-in. water column. Would Mr. Savile Davis describe how the transfer is effected ?

The rate of development maintained at depths between 4,000 ft. and 5,000 ft. from a single shaft with a limited labour force under wartime conditions, and amidst other difficulties, is a matter for the congratulation of everybody concerned. The improvement in the rate of advance in the drives that has resulted from the use of ' burn ' cuts is considerable ; a description of this cut, with details of the blasting, the amount of explosive loaded in the holes, and a sketch of the round generally drilled would be welcomed. Has Mr. Savile Davis used the ' burn ' cut in rises ? I have successfully used them on several occasions when it has been necessary to avoid damaging equipment in ' holing ' into working shafts, etc. In these cases the amount of ground remaining to be traversed was usually 3 ft. to 4 ft. Some time ago I made experiments with ' burn ' cuts in drives 7½ ft. by 5½ ft. in cross-section. The rock was tenacious, and the average advance in these drives was 4½-ft. per round when a 26-hole round with a ' pyramid ' cut was used ; the explosives consumption was 9 lb. of 50 per cent ammon. gelignite per foot advanced. The best advances of 8 ft. per round, with an explosives consumption of 11.5 lb. per foot, were attained with a ' burn ' cut having three parallel holes with their centres steel industry and was also one of the few minerals occurring in this country of which the production satisfied the whole of our domestic requirements. It was the only substantial source of the element fluorine and just recently there had been new industrial applications of compounds of that element—he might mention the gas 'freon,' an important item in refrigeration, particularly in America. There were also new uses for fluorine products as insecticides.

Just before the war this country produced approximately 40,000 tons of fluorspar a year, which was valued at about £40,000. By 1944 this had risen to about 70,000 tons, and the value was £220,000. Thus there had been a substantial rise in production and a spectacular rise in value. Comparing it with perhaps a better known non-ferrous industry in this country—the Cornish tin industry—the value of fluorspar production was about half that of tin and employed about one-third of the number of men engaged in tin mining. The proportions of production used for metallurgical, acid-making, etc., purposes were given in the paper.

The possible need for re-investigation of fluorspar resources was foreseen by the Geological Survey before the war and they instituted a re-survey of the deposits in the North, in Durham chiefly. Mr. Pearson would desire to give credit to the Geological Survey for bringing to the attention of the Ministry of Supply very early in the war that there was a danger that if more fluorspar were needed it might not be obtainable with existing plants except at the expense of a fall in grade. A joint investigation was accordingly carried out by the Control and the Geological Survey, and as a result it was shown that: First, certain methods of washing and treatment in common use were definitely wasteful of mineral and if continued could not fail to react to the detriment of the industry as a whole, and, secondly, that if production were increased there would be almost certainly a fall in grade which would, of course, react against steel production, and, what was almost as important, would involve using transport for material of which a proportion had relatively little value.

When the supply position became more acute the first step to be taken was to control fluorspar, to license its sale and purchase, and regulate its distribution. The co-operation of all producers was secured, and later it was found practicable to set up a scheme whereby fluorspar was sold at prices which had some relation to the grade of the product—a thing which hardly obtained at the beginning of the war. Mr. Pearson would wish to give credit to Fluorspar Control for the work they had done in this connexion, since he felt it would prove to be of lasting benefit to the industry.

The metallurgical side of the investigation really started in 1942. About that time there developed a possible extra demand for acid-grade fluorspar and in view of this, coupled with the continued shortage of metallurgical-grade spar, it was definitely agreed that some provision might have to be made for a new plant or plants. A series of metallurgical tests was undertaken to decide whether there was a suitable occurrence of raw material on which a new plant could be based and which could be brought into production, if necessary, sufficiently quickly. Mr. Pearson made quite clear in the paper the reasons which led to a site being chosen at Stanhope, in Weardale, Durham. Three mines were working in that vicinity, and together it seemed certain that they could supply the quantity of raw material required to keep the plant going at full capacity, for several years at any rate. The possible demand for acid-grade spar made it essential that they should be able to draw on a mine or source of supply of suitable feed for the plant, and that finally led to the decision on the project to be located in Stanhope.

The results of the tests were given in fairly full detail. One point which Mr. Pearson wished to be mentioned was that complete drawings were made, but, unfortunately, owing to the paper shortage, it was not possible to publish them all. They had, therefore, been lodged with the Institution and were available if necessary for any interested parties to see. With regard to treatment costs, Mr. Pearson did go into these in the paper. Estimates had, of course, been made, and he would be glad to give anyone his views should they be interested.

Concerning capital cost, £56,000 might seem a lot of money for a plant to treat fluorspar, but during the war it was necessary to be prepared for any contingencies. Of course, when the turnover of the industry as a whole was considered—£200,000 a year or so—it was hardly attractive to consider plants costing £56,000. The main points during the war were to ascertain the ore-reserve position and carry out the metallurgical tests, and this was done to the best of their ability under the conditions then obtaining.

He wished, on behalf of the author, especially to thank Mr. L. A. Wood, Mr. Michael Callow (British Geco Engineering Co.) Mr. James Jackson and Dr. K. C. Dunham, for the help they had rendered while the investigation was proceeding.

Mr. E. J. Pryor said that with today's economic position demanding the fullest utilisation of this island's mineral resources a paper of this character had especial value. Since the proposed concentrating plant was still at the blue-print stage he had read the paper with possible operating difficulties in mind. The author observed that fluorspar was 'an extremely friable mineral'. Elsewhere colloidal slime, clay, barytes, etc., were mentioned; also the existence of dumps which might be re-worked. Presumably, then, a good deal of primary slime would be fed into the treatment plant. Most non-metallic flotation plants tried to avoid treating primary slimes, but in the flow-sheet presented nothing seemed to be done to remove them before secondary—or value-bearing—slimes were created in the ball-mill. It was not clear to the speaker whether these slimes might be expected to interfere, but he thought there was something of a case for de-sliming between the drum-washer and the ball-mill. If present in quantity, such slimes would consume reagents, hinder selectivity by the collector reagent, and crowd the air-bubbles to the detriment of desired floating mineral.

Next, he would ask why a high-discharge ball-mill was proposed rather than a low-discharge mill—ball or rod. It seemed that over-grinding would be a danger in preparing this friable fluorite for flotation and subsequent treatment. With a low-discharge mill the dwelling time would be shorter than with high-discharge, allowing for better removal of finished material in the close-circuiting classifier and if a rod mill were used better selectivity of grind, directed upon the coarsest material in the mill, could be obtained. He was assuming that overgrinding would lead to trouble somewhere.

Turning to the batch tests, it would help in interpreting the work if the connection between size and assay value was known for representative samples of concentrate, middling, and tail of the flotation products. Such information was, he considered, essential for good grinding control and in its turn good grinding control was a major factor in preparation of a suitable feed to the flotation section. Beside being a chemical process flotation could be thought of as an artifice for temporarily changing the density of part of the solids in an ore-pulp by attaching them to an air-bubble. The reactions leading to this temporary association were easily influenced in either a bonding or a dispersing direction and grinding mesh was an important factor in these influences. Loss, or recovery, at each size in a screen-fractionated sample of product, therefore, gave valuable information in studying a flow-sheet.

Next, he wondered whether there was a special reason for choosing Denver cells. Though galena was being removed, the main object was to lift some 82 per cent of the raw feed as a high-grade fluorite. This suggested to the speaker the possibility that the best machine might be one with considerable overflow capacity and transporting power upward in the cell, even if this entailed comparatively 'soft' aeration.

As one unfamiliar with the commercial phraseology of fluorite, he did not learn anything from reading the specifications in Tables I and II, and perhaps others might feel that the time had come for better clarification. Would it be possible to market calcium fluorite in terms of units instead of the rather wide percentages quoted? The trend of modern technology was always toward closer specification, so that all parties to a transaction knew with good accuracy the values and impurities residing in a shipping product. If modern processing methods were applied to fluorspar, they would be aided by good specification of materials dealt in.

In conclusion, he desired to thank the author for a valuable and timely paper, which put on record information that could have much practical value.

Dr. K. C. Dunham said that he was very glad that Mr. Pearson had put on record the results of the Non-ferrous Minerals Development Control's work on fluorspar, for even though the projected mill was not built, the preliminary investigations had given valuable information about the characteristics of the ore from many British deposits. The survey of the Durham and Derbyshire areas in 1939-42 had clearly shown the inadequacy of fluorspar dressing equipment in both areas. Particularly in Derbyshire, present dressing practice led to a very high loss of fines and substantial wastage of reserves. This is only one of several metallurgical problems awaiting solution in the fluorspar fields.

As far as the limited evidence went it appeared that considerable reserves of fluorspar remained and practical efforts to solve these problems, therefore, appeared to be justified, particularly under the improved economic conditions for the industry which had been created in the past four years. The plan for the Stanhope mill, had it been carried out, would have been of considerable benefit to the Durham mines, and it was to be hoped that this plan, or something like it, would be brought to fruition. In this area a central mill, drawing ore from several mines, appeared to have a better chance of success than one catering only for a single deposit.

Mr. L. A. Wood said that due to the late hour he would prefer to make a written contribution on the paper itself, but Mr. Pryor had raised one or two points of interest and it might be just as well to deal with those. He thought that Mr. Pryor had looked at the matter in a very theoretical way and had introduced a few theoretical factors that really did not exist in the case.

In the question of colloidal slime, for instance; in some of the deposits one did find a little clay, but in the particular deposit which they were intending to treat there was a remarkable absence of clays. There was a great deal to be said for Mr. Pryor's point of view with regard to colloidal slimes, especially where one had a mine where there had been a certain amount of weathering, but normal colloidal fines did not interfere except where this weathering had taken place. In the majority of cases the fines were secondary fines and these helped, because without these fines the bubbles did not remain stable. Fines helped to form a kind of armour plating.

There was some misconception on another matter. The normal practice at the present time was to grind with only 30 per cent moisture. This made a thick slurry which held up the coarser material and did not allow it to fall to the bottom as it would in a dilute pulp. Therefore, the coarse material came from a high level, just as well as from a low level. Some years ago a number of tests were carried out with Hardinge ball-mills and rod-mills, and there was very little to choose between the two.

Concerning mesh analysis; it was very pleasant to have all these data, but fluorine assays took somewhere about a fortnight or three weeks and cost something like ten shillings each, so that they had to be careful. Hence most of the work was done by observation. It was all very well to talk about fixing up grinding to give the best results, but how was the grinding during the mechanical stage to be controlled? If one had to grind to 80-mesh one had to take what was coming with the 80-mesh. Fortunately in this particular case they did not find any difficulties at all with the flotation. As a matter of fact, flotation under the conditions employed was just as good as would be got with a lead ore or a zinc ore.

That again brought up the question of the cells. There had been dozens of cells invented, all with various little differences which were supposed to give benefit. The Denver cell had been chosen as a practical machine of known performance.

The question of percentage of froth was not so important as it sounded. It was just a matter of taking the froth off a little faster. The pulp certainly thinned out, but the froth carried some 50 per cent of moisture, and that, of course, reduced the amount of water left to the tailings. The trouble evened out in practice.

On the proposal of **the President**, the hearty thanks of the Institution were accorded to the authors of the two papers.

CONTRIBUTED REMARKS

Mr. L. A. Wood: I should like to express to the author an appreciation of his paper in which the subject is dealt with in a simple, logical and comprehensive manner. Although the proposed mill was never put into operation Mr. Pearson has recognized that the information and experience gained are of definite value to those concerned in the fluorspar industry and are not without interest to others of our profession who are not directly concerned. He has, moreover, translated his recognition of this value into practical aid by collecting, correlating, and imparting this information. An extra measure of praise is due to him for the spirit of duty and desire to serve which prompted him to submit this paper.

I venture to offer a few brief remarks to amplify somewhat some points concerning the concentration of the fluorspars mentioned in the paper.

Experience showed that the main impurities associated with fluorspar are; silica; the sulphides of lead, zinc and occasionally iron; some oxidized iron (mainly limonite); calcite; barytes, and, sometimes, clay.

The Weardale ores are free from calcite and barytes but were marred by association with quartz, which permeated the calcite as a network of fine veinlets, and by galena and blende in small proportion. The sulphides, however, were segregated and fairly coarsely crystallized.

The Derbyshire ores were more coarsely crystallized and free from an excess of fine quartz veinlets, but were associated with varying proportions of calcite and barytes, in addition to sulphide minerals and quartz. One deposit, however—locally known as Dunstone—carried some 75.5 per cent fluorite; 14.5 per cent silica; 8.5 per cent barytes, and 1.5 per cent of carbonates. These impurities were so intimately associated that even at 200-mesh the minerals were not completely freed. On the whole, therefore, it was found that whereas jigs were in fairly successful operation in Derbyshire for fluorspar concentration they were not of continued success in Weardale. Generalization, however, is dangerous, and those interested in the industry will be well advised to study each deposit on its merits before deciding on methods of concentration.

In studying the various samples from the point of view of producing good-grade metallurgical spar, all of which must, theoretically remain on a ⅛-in. screen, some rather ominous data were collected. Lump spar, from 7 in. to 1 in., when broken to ½ in. by most careful stage-crushing, always produced 30 to 35 per cent of fines through ⅛-in. Moreover the fines were always richer in fluorspar than the coarse sizes.

Run-of-mine fluorspar, after crushing to size suitable for metallurgical purposes, usually exceeds this proportion. A not unusual example is that of a Stotfield Burn shipment in 1941 which showed 65 per cent of *minus* ⅛-in. fines.

An extreme case was that of the Masson mine, Matlock, Derbyshire, where the ore as mined carried 8.22 per cent—3 in.+½ in. material: 23.81 per cent—½ in.+⅛ in.; and 67.97 per cent—⅛ in. When the oversize was crushed to ½ in. for jigging, there was left only 25.8 per cent of + ⅛ in. To produce a metallurgical-grade spar honestly according to schedule, 74.2 per cent of the total weight containing 77 per cent of the total fluorspar content would need to be discarded before jigging.

That this waste could be eliminated by froth flotation with the production of a superior product, was amply demonstrated. That the flotation product could be briquetted for metallurgical use was also tested and proved, but prior claims on manpower and binders prevented war-time application of these processes.

Froth floatation of fluorspar for the production of acid-grade spar was already conducted on a small scale at two plants in Derbyshire. Both plants were of the secret process type. Other plants of larger capacity (up to 10 tons per hour) were in operation in the United States. There was also vague information of fluorspar flotation plants on the continent.

From information available, mostly from American sources, it appeared that the reagents employed to date were all within the range of experience at the laboratories of Minerals Separation, Ltd., in 1916 to 1918, when the late Professor Edser was in control of research on the flotation of non-sulphide ores. As activator, a fatty acid or soap was employed and as modifiers, sodium silicate, sodium carbonate, tannin, glue, starch, or sulphonated products. It was found that sodium silicate in circuit neutralized to pH about 7.2 was the best depressant for silica, as it produced thick stable froths and allowed rapid separation. This simple depressant, however, did not eliminate calcite or barytes.

The addition of bichromate depressed calcite and barytes to some extent, but for final elimination of barytes it was necessary to employ tannin in circuit made alkaline to pH 10 or over with soda ash. Tannin, however, depresses fluorite as well as barytes, although not so completely, with the result that the suitable differential froth will carry fine fluorspar only and in such small quantity that a much larger flotation plant is necessary. Bichromate produces its effect by making the froth more brittle and showery. It also demands some extension of plant and somewhat finer grinding.

Choice for production of acid-grade spar fell most decidedly on the better-grade Weardale ores, which, although too heavily impregnated with silica to make first-class metallurgical spar, are sufficiently free at about 60-mesh to yield high recoveries of good-grade acid spar with simple silicate reagents, and with the minimum of flotation machines. Many Derbyshire ores yielded excellent-grade metallurgical spar in one simple treatment with bichromate and silicate, but all of them needed a second treatment with tannin in alkaline circuit to get the requisite acid grade.

From the purely technical point of view, therefore, it was concluded that Weardale was the best district for the production of the cheapest acid-grade spar, and Derbyshire for metallurgical spar. Purely technical conclusions, however, often have to be modified by other considerations and we may take it that the fluorspar industry will develop according to circumstances outside our sphere.

Mr. F. Bice Michell: Mr. Pearson's paper is both most interesting and timely, and should prove of assistance to those interested in the treatment of fluorspar in this country. The whole of the non-metallic mining industry seems to have been somewhat neglected for a considerable time and little has been done in recent years to improve the operation of fluorspar dressing; in fact, often no attempt is made to treat the fines which cannot be jigged. Usually, when tables have been used, the concentration is unsatisfactory from the viewpoint of recovery, as although sufficiently high grade a concentrate can be made, the tailing is rich in spar.

The suggested flow-sheet as well as the details of flotation tests are most interesting and I should like to know if the author can give any further information on the following:

(1) Was there any particular reason—such as the selective grinding of the soft fluorspar—for using the Lightning crusher?

(2) The jig tailing is shown as being dumped, but such tailing is often high in fluorspar. Were there any tests made with a view to ascertaining the grade of the spar produced in this way and the recovery?

(3) Were the jigs designed to concentrate 'over' or 'through' the sieve? It has been my experience that much better work can

be done with a ' pen and pipe ' discharge when jigging on the sieve, in preference to the more usual practice of hutch working.

Mr. Pearson also mentions agglomerate tabling, which to me seems to be a promising alternative to flotation and I have found that ' table flotation ' will produce a good grade of fluorspar in one operation, while it allows of gravity concentration for any galena that may be present—a fact which greatly simplifies the layout of a small plant. It is true that a certain percentage of the fines cannot be treated in this manner and if this proportion is sufficient to justify the outlay it should probably be treated by flotation in the usual manner. The disadvantages of having a combined process, however, seem to be offset by the ability to treat coarser material with reduced grinding costs, by elimination of much of the loss if any desliming is necessary prior to flotation, and by the saving of costly filtering plant. On one ore it was found possible to make a concentrate in one operation which was lower in silica than that obtained by straight flotation with one cleaning stage.

In the production of fluorspar by flotation briquetting is obviously required and the statement made regarding the favourable report received from a steel maker will be interesting to many producers who no doubt would also be interested to know what type of plant was in the engineer's mind. A similar binder has been used in some tests made recently in Canada and 3.75 per cent of pitch was considered adequate. It is stated that if the concentrate contains more than 3½ to 4 per cent moisture it should be pre-dried, as the binder will not adhere to the fluorite. The petroleum asphalt binder was added at a temperature above 200°F. and before briquetting was lowered to between 170 and 180°F. It would be interesting to know how these results compared with the experiment mentioned in the paper.

In the discussion of the flotation investigations the use of neutralized sodium silicate is most interesting, as the addition of large quantities of ordinary sodium silicate to reduce contamination in the froth will increase the pH beyond the optimum for good fluorite flotation and tends to produce a very brittle froth. The use of quebracho for depressing barite also calls for some comment, as although it is highly successful on calcite its efficacy on barite seems to vary on different ores. Furthermore, it tends to depress the fluorite as well and a careful balance is needed between the oleic acid and the quebracho if the latter is added in the rougher operation.

Finally, a 30-ft. by 10-ft. thickener is shown in the flow-sheet *between the grinding* and the flotation unit. Was partial desliming intended at this point and if so, what percentage of the flotation feed was expected to be lost in this way ?

Mr. Donald Gill: Mr. Pearson is to be congratulated upon having produced a useful and workmanlike paper. I should like to comment upon some points raised by Mr. Colin Rose in his introductory speech :

First, Mr. Rose indicated that sympathetic encouragement would be given to the publication of suitable technical papers dealing with the activities of the Non-ferrous Minerals Development Control. This enlightened attitude is much to be commended; it is to be hoped that more than one valuable paper may result from it. During the phase of gradual decontrol through which the country is now passing there is great danger of technical information of potential value to the community being entombed in official files and ultimately burned or lost. Publication of whatever may be of value, within the limits imposed by 'security,' is, therefore, to be encouraged.

Secondly, the fact that the detailed plans of the proposed mill are on file with the Institution should be noted with satisfaction.

Thirdly, Mr. Rose said that fluorspar was the only available substantial source of fluorine in this country and he mentioned the increasing use of fluorine compounds—such as freon gas. I should like to draw attention to the considerable tonnage of combined fluorine which enters this country in our imports of phosphate rock and to suggest that the utilization of part of this fluorine might be practicable. All phosphate rock contains fluorine, the percentage varying from around 1 per cent to about 3.4 per cent. The fluorine in phosphate rock exists as an integral part of the rather complex molecules of the oolitic grains composing the rock, the composition of the grains varying between the limits shown in Table XIII.

TABLE XIII

Minerals	*Bulk formulae*				*Fluorine, per cent*
	CaO	P_2O_5	CO_2	F	
Colophanite	100	30	10	5	0·9
Francolite.....................	100	29	7	18	3·3
For comparison : Fluor-apatite	100	30	Nil	20	3·7

The tonnage of phosphate rock imported into Great Britain in a normal year is of the order of 500,000 tons. Imports during the war have been abnormal as to both quantities and sources, but if a normal year, say 1938, is taken the quantity of imported combined fluorine may be gauged.* Table XIV shows tonnages of phosphate rock and of contained fluorine imported into Great Britain and Eire in 1938. To a first approximation, nearly 9,000 tons of combined fluorine arrived in Great Britain and Eire in 1938, in phosphate rock carrying on the average about 2 per cent of fluorine.

TABLE XIV

Source	*Phosphate Rock Imports, Metric Tons, 000's*	*Contained Fluorine*	
		Per cent	*Metric Tons (approx.)*
Morocco	96	3·3	3,150
Algeria (Constantine) ...	83	3·0	2,500
Tunisia (Gafsa, Mdilla)...	208	1·0	2,080
,, (Kalaa Djerda)...	19	2·2	420
Florida pebble	4	3·0	120
Nauru/Ocean...............	22	2·6	570
Egypt (Kosseir)............	7	1·3	90
Totals and average ...	439	(2·03)	8,930

Now, a large proportion of the phosphate rock entering the country is converted to 'superphosphate' by treatment with sulphuric acid. In the course of treatment, about two-thirds of the contained fluorine is driven off in the form of fume, together with steam and carbon dioxide, while about one-third of the fluorine remains in the superphosphate. The reaction is somewhat complex and is illustrated by Gray (*loc. cit.* p. 130), who gives a balance-sheet. The exact composition of the fumes is probably uncertain, but they may be regarded as a mixture of gaseous hydrofluoric acid, silicon fluoride, hydrofluosilicic acid, steam, and carbon dioxide.

*GRAY, A. N. 'Phosphates and Superphosphate', 1944. The tonnages of phosphate rock imports are taken from the statistical tables in this useful book.

At all events, they are extremely noxious and they represent a loss into the atmosphere of fluorine which, with minor necessary plant changes, could probably be recuperated.

The production of superphosphate in Great Britain and Eire in 1938 was 559,930 metric tons, probably from about 330,000 tons of phosphate rock. If the rock used contained on the average about 2 per cent of combined fluorine, as is indicated in Table XIV, and if about two-thirds of this fluorine could have been recuperated from plant-fumes, in the form of hydrofluosilicic acid or fluosilicates, then it would appear that in 1938 about 4,400 tons of fluorine were wastefully emptied into the atmosphere of the British Isles. At the present time the wastage may be much greater. I am not aware whether any recuperation of fluorine from superphosphate plant fumes has been attempted in Great Britain, but it is stated* that in the U.S.A. 'many superphosphate plants are utilising these gases for the production of hydrofluosilicic acid and fluosilicates.'

It appears from the paper (p. 2) that there is a market in this country for about 5,000 tons per annum of 'chemical-grade' fluorspar (equivalent to about 2,500 tons of fluorine) and that the chemical uses are increasing. I should like to ask Mr. Pearson, in the interest of conservation of resources, whether and to what extent fluorine recuperated from superphosphate plants might in the future afford substitutes for fluorspar for use in the non-ferrous, glass, enamel, and chemical industries.

Mr. J. Norman Wynne: Mr. Pearson's contribution towards the study of Britain's potentially-valuable fluorspar resources is of particular personal interest to the writer, who was responsible for the technical development of the Stanhopeburn fluorspar deposits in Weardale and for the establishment during the war-years of ore reserves exceeding 250,000 tons. It must be admitted that Britain has unfortunately lagged far behind the United States and Germany in the beneficiation of fluorspar, which (as our war demands immediately demonstrated) must be regarded as one of the primary strategic minerals. In the years ahead, with keen foreign competition in the steel industry, production of this essential ingredient in steel-making will need greatly to be improved. As Mr. Pearson indicates, the steel companies themselves were largely responsible for the almost complete lack of attention to the preparation of the crude spar as produced either from mines or accumulated lead-mine tailing-dumps. Although, as stated, the average fluorspar

*Gray, *loc. cit.*, p. 45, quoting Technical Bulletin No. 364 of U.S. Dept. of Agriculture, published 1933.

cost in smelting is only about 4½d. (and was considerably less in the pre-war years) per ton of raw steel produced, the criterion with many steel firms was ' cheap ' buying price—with purity apparently a very secondary consideration. As recently as 1936 the normal selling-price of crude fluorspar was under 20s. per ton and prior to 1914 it was as low as 6s. f.o.r. So long as the material bore some resemblance to fluorspar, many steel companies were apparently content to purchase it provided the price was low, and spar containing as low as 50 per cent CaF_2 was readily purchased during the critical war years. There was, therefore, little encouragement for producers to lay out considerable sums on plants designed to beneficiate low- or medium-grade spar to a standard more appropriate to modern steel-making technique.

It may be noted that before 1928 upwards of 30,000 tons of fluorspar was shipped annually from Britain to the United States ; but this valuable export from the Tyne gradually decreased and finally was entirely lost, simply because the material shipped fell seriously short of American steel producers' increasingly strict specification.

This export trade can and should, in the writer's opinion, be not only revived, but considerably increased. Britain has vast fluorspar resources, but producers must be prepared to supply refined fluorite approximating to, or bettering, that consistent grade which America is herself producing (if somewhat expensively) from comparatively low-grade domestic sources, principally in Illinois and Kentucky. Two years ago the United States Bureau of Mines sought to purchase from Britain some 40,000 tons of fluorspar per annum, but the relatively high metallurgical grade required was not forthcoming. In Britain, the steel companies are today paying much closer attention to the CaF_2, SiO_2, and $BaSO_4$ content in the fluorspar they purchase. Plainly British fluorspar producers must, without further delay, set their own house in order and become ' assay conscious.' In this direction the British Fluorspar Producers Association has already done much valuable work, particularly in establishing a close technical liaison between the steel companies and the fluorspar producers.

The correct preparation of refined fluorite, however, presents some difficulties, fortunately readily overcome. Mr. Pearson mentions the extreme friability of fluorspar and the consequent production of ' fines ' during mining and transport of the ' run-of-mine ' to surface ; at the Stanhopeburn mines, this is perhaps the most serious difficulty. It is important to appreciate that fluorspar vein-matrix is comprised of aggregations of microscopic crystals of fluorite, intermingled with SiO_2 (and in Derbyshire, barytes) crystals. In the Stanhopeburn lodes (which attain a width of 30 ft. in parts) the fluorspar is so easily broken that the average explosives consumption is under 0.5-lb. per ton mined and great masses are reduced to ' smalls ' by a single blow from a hammer or pick; in some parts of the lodes the spar may be reduced to powder by squeezing in the hand. With a view to reduction of ' fines', specially low-powered explosives are used in development and stopes, yet, notwithstanding careful attention to this point, a recent large-scale screen analysis test showed that nearly 50 per cent of the normal ' run-of-mine ' material was *minus* 60-mesh.

Another difficulty with the Weardale spars is that the SiO_2 occurs in a finely-divided state along the cleavage planes of the minute fluorite crystals, as well as in the form of veinlets ramifying the vein-matrix. Some crushing is, therefore, necessary in order to ' disentangle ' the two principal constituents prior to separation and naturally the finer the crushing the greater the disentanglement.

The projected treatment mill at Stanhope, it is stated, included jigs ' for the production of a jig concentrate of metallurgical grade'. In Britain the steel specification is nominally (*faute de mieux*) 10 per cent SiO_2; but in America and Germany the standard is 'not exceeding 6 per cent SiO_2'. In view of the closeness in specific gravity of CaF_2 and SiO_2 (3.01—3.25 and 2.65—2.66, respectively), the writer considers that jigging is incapable of reducing the *silica* content in the Weardale spars. Doubtless jigging effectively reduces the barium minerals (barite, Sp. G. 4.3—4.6), but for the beneficiation of Stanhopeburn fluorspar (which contains no barite) the writer came to the conclusion that jigging would be a waste of time and money and on a par with the so-called ' washing ' of the ' run-of-mine ' material even today commonly practised. It would be interesting to hear whether jig tests were made on the Stanhopeburn and Hope Level spars by the British Geco Engineering Co. or others, and, if so, what the product from such jigging assayed. Unless jigging reduces the *silica* content from about 20 per cent to under 6 per cent, this method can hardly be regarded as justified in the case of the Weardale (barium-free) spars. The galena may readily be effectively reduced and recovered as a valuable by-product by ordinary gravity tables.

The writer consulted a number of steel companies and from them obtained much valuable information regarding their requirements. All emphasized that fluorspar crushed to less than about 20-mesh would be unacceptable, because finely-powdered material would be swept into the flues of the blast furnaces without coming into intimate contact with the molten steel-charge, and, therefore, the liquifying of the slag and the removal of the sulphur and phosphorous (the principal object of the application of the fluorite) would not then take place. They pointed out that the 'fluorspar' (however high its CaF_2 content) *must* be of 'gravel' size that could conveniently be cast by shovel (in a spreading shower) to fall on to and penetrate the blanket of molten slag covering the steel. Therefore, in order to produce metallurgical-grade fluorite it would seem necessary to avoid *fine* crushing, and to keep the spar particles as large as possible—consistent with ability to disentangle the CaF_2 and SiO_2 components sufficiently to obtain a reduction of the latter to *minus* 6 per cent.

Froth flotation methods readily produce the acid-grade fluorite (containing about 0.5 per cent SiO_2) and for chemical use the product *must* be finely ground. For the production of acid-grade spar, therefore, froth-flotation methods are ideal. Unfortunately, as Mr. Pearson points out, the war-time annual demand for acid-grade spar amounted to only 5,000 tons. On the other hand the steel industry consumes about 86 per cent of the total production of fluorspar in Britain (and practically as much in the United States) : the producers' market is plainly in the steel industry, therefore, even though, as seems likely, the demand for acid-grade (for refrigerants, petroleum refining, synthetic rubber manufacture, etc.), may increase very considerably in the future. But, because the resulting concentrate is in too finely divided a state for application to blast furnaces, froth-flotation methods of beneficiation seemed to the writer to be anything but ideal for the production of metallurgical-grade material. His attention was, therefore, focused upon the recently-developed 'table-agglomeration' method, whereby it is cheaply possible to produce a *minus* 4 per cent SiO_2 grade of ⅛ in. size particle and approximating to the fine 'gravel' specified by the steel companies. Large-scale tests on the Stanhopeburn fluorspar were carried out and the resulting 'granular' concentrate pronounced ideal in every respect for steel-making.

Even in rough crushing preparatory to table-agglomeration (which process is based upon the differential affinity for oil of CaF_2 and SiO_2) it is impossible to avoid the production of a certain percentage of *minus* 60-mesh material, which is, of course, additional to that resulting from blasting, trucking, hoisting, etc., in the mine. Obviously it would be uneconomical to screen and reject the total percentage of 'fines', although some of this total percentage could doubtless be sold as ceramic grade. Therefore, in the writer's view, the *real* problem in the beneficiation of crude fluorspar is what to do with the substantial proportion of *minus* 60-mesh material (100 per cent in the case of froth-flotation concentrate) resulting from mining operations and/or crushing preparatory to either table-agglomeration or froth-flotation separation treatment. This problem, incidentally, has been and still is the 'headache' of American fluorspar producers, whose research work is far ahead of Britain's.

On p. 7 Mr. Pearson refers to the briquetting of the metallurgical-grade concentrates produced by froth-flotation treatment and mentions obtaining a good-class briquette by admixture of 5 per cent pitch as the binder. In the Lake Ainslie District of Nova Scotia, it was found* that pre-heating of the concentrate to approximately 250° F. was necessary, because if the concentrate contains more than 3½ to 4 per cent moisture the petroleum asphalt binder used will not adhere to the fluorite. Between 3 and 4 per cent by weight of the asphalt binder, at a temperature exceeding 200°F was added, and the whole was thoroughly mixed. The temperature of the mix was then lowered to between 170° and 180°F. before briquetting. In the discussion following the presentation of the paper: 'The Concentration of Barite-Fluorite ores from the Lake Ainslie District, Nova Scotia,' by Mr. E. Lee Cameron, before the Mining Society of Nova Scotia, a speaker rightly, in the writer's opinion, stated :

> I would also look for an improvement in the briquetting process. The method the author used calls for completely drying the concentrates and heating with a fairly high percentage of petroleum asphalt binder, which is usually high in sulphur. This is an expensive process and, moreover, it may make difficulties in conforming to the sulphur specifications for metallurgical fluorspar. . . . It is reported that satisfactory briquettes of fluorite concentrates have been made in the United States with addition of one-half of 1 per cent of bentonite as a binder. Such a process will be found to be superior to that of using the asphalt binder.

The writer's information is that the addition of bituminous material (apart from the gratuitous introduction of objectionable sulphur and carbon to the steel charge) results in an undesirable explosive action when fluorite thus bonded comes into contact with the molten charge. But, apart from all these objections, it is difficult to see how virtually 'sparmac' can possibly be cast into open-hearth furnaces as a shower of 'gravel,' particularly in summer heat or close to furnaces when the tendency would un-

*Trans. Can. Inst.M.M., Vol. XLVIII, 1945, pp. 567–587.

doubtedly be for the pitch or similar binder to soften and produce a treacly mass which it would be impossible for the charge-man to handle ; it is, of course, out of the question to hurl chunks of fluorspar into the molten mass—the *particles* of calcium-fluoride must be spread, in intimate contact, over the *whole* surface of the charge.

Experiments were made with the use of sulphite lye (from pulp mills) and/or hydrated limes as the binding media, but this was abandoned because the steel producers strongly objected to the introduction of undesirable adulterants to their steel charges. As already mentioned, the latest trend in America is towards the use of bentonite (a hydrous aluminium silicate derived from the alteration of volcanic ash) and the production of non-sticky ' pellets ' of a size more suitable for application by shovel.

Whatever the binder used may be, however, it certainly is no cheap or easy matter effectively to admix intimately (and, in the case of asphalt or similar material, to heat the whole tonnage) so small a quantity as even 6 per cent by weight throughout the mass treated.

The simple (and economic) key to the problem would appear to have been found in Britain. After considerable experimenting the writer discovered that if fluorite concentrates (of whatever mesh) be subjected to a temperature of about 1,100°C.—i.e., well below fusing point of the fluorite—for about 10 minutes a natural re-agglomeration of the particles takes place, and a hard homogeneous mass results. No binding medium whatever is required ; which cuts out an expensive and messy process. It is interesting to note that this natural bonding takes place while the concentrates are still at maximum heat and not upon cooling. The precise reason for this agglomeration (in ceramic work known as ' fritting ') is not fully understood, for, as microphotos of sections (Figs. 2 and 3, Plate II) of the bonded material show, no sintering (as usually understood) takes place, and the fragments of fluorite remain sharply angular. Analyses show that no chemical change takes place in the CaF_2 composition and its action upon steel fluxes is, therefore, unaffected. Tests showed that this process is cheap in fuel cost, handling is reduced to the minimum, and heat-bonded concentrates may be produced in any desired size or weight. Samples of the heat-treated fluorite (certified to contain only 4 per cent SiO_2, and no barium or metallic compounds) were submitted for critical examination to a number of steel firms in Britain ; its suitability is perhaps best proved by their readiness to pay £5 per ton (alike for the ' granular ' screened ⅛-in. mesh and the bonded product) instead of the current £3 per ton for the crude (or imperfectly treated) so-called fluorspar normally supplied. As a result it should now be possible for Britain to develop a very valuable and substantial export trade.

Mr. E. W. O. Dawson: Although this project was only considered under the stress of war something on the lines suggested would not only provide an efficient means of dealing with the fluorspar in the district, but also greatly help the neighbourhood in times of peace, by preventing it from again becoming (as it was prior to 1939) a distressed area.

A further point of interest is the mention of the possibility of utilizing the Derbyshire dumps for the recovery of barytes by flotation. Whereas fluorspar was treated, pre-war, by flotation in Derbyshire, so far as I know no one has ever attempted the flotation of barytes in this country on a commercial scale. One difficulty which the barytes producers have had to surmount is that very few, if any, of the lithophone manufacturers will take the fine material in a wet state, but I see no reason why the paint manufacturers' requirements should not be met with this material.

Barytes is a fairly easy flotation proposition, giving, thereby, much better recoveries than can be obtained by gravity treatment. Another point is that the material would be already ground fairly fine and after bleaching and drying would require very little more grinding to meet with the paint manufacturers' specifications, a considerable saving in grinding costs being so achieved.

Mr. J. H. Hohnen: The idea of a central mill for the Durham fluorspar industry to be situated near the Stanhopeburn mine in Weardale enthused me when it was projected back in 1941. It seemed to me that we had a blueprint for post-war development of one small part of the British non-ferrous industry as well as a plan for local employment. Had the mill gone ahead in 1942 it would have been at work to-day, operating under a Government-controlled corporation and providing stimulus to the development of the small mines whose owners, as Mr. Pearson says, 'lacked the necessary capital to indulge in what to them was the luxury of expensive washing plants'. There is no doubt that, relieved of the milling problem and guaranteed a local market for 'run of the mine' ore, a greater amount of mining development would result and, furthermore, small parcels would come in from new or re-started adits in the neighbourhood.

The idea of State-controlled milling plants is not new and should have some appeal to the present British Government.

The gold batteries operated by the State in Western Australia have always given encouragement to the small operator. It is a pity, therefore, that Stanhopeburn mill is not more than a paper project and to-day possibly a forerunner of similar enterprises.

× 30
FIG. 2.—Thin section No. 1

FIG. 3.—Thin section No. 2. × 30

FURTHER CONTRIBUTED REMARKS

ON

Some Aspects of Rock-Drilling Practice. (The Witwatersrand Goldfield).*

By C. J. Irving, *Member.*

Mr. W. Elsdon-Dew :† Mr. C. J. Irving in an excellent paper has presented an up-to-date review of conditions as they prevail today on the Witwatersrand goldfields.

The importance of rock drilling, etc., calls for no comment, especially as the record (16) of references given in the paper of previous publications indicates how much attention has been paid to this subject in the past.

I may be forgiven if I refer to some historical facts which may not be known to all the members. The work of the Investigation Committee of the Central Mining-Rand Mines Group referred to by the author was inaugurated by me in 1919/1920 and brought about closer co-operation and co-ordination between the mining engineering and mechanical engineering staffs in tackling common problems. Progress since that time has been substantial and I strongly support the author's plea for more extensive co-operation and co-ordination amongst all the technical staffs.

It is on record before the general adoption of the jack-hammer that for each hole drilled four jumpers were used and the number of holes drilled by each hammer boy never exceeded two to three holes per shift. The prediction then by the late Mr. Leslie Pryce of one drill to one hole seemed rather optimistic, but the position today is one drill one hole, or even one drill two holes sometimes, with the native jack-hammer boy drilling 40 to 60 holes per shift.

At that time the introduction and adaptation of the hand-held jack-hammer was a most important advance in mining practice and marked the beginning of important improvements and economical drilling practice. I consider the introduction of the detachable bit (P. & M. Patent) now marks a new advance and makes it possible for just as important developments to be made in rock drill mining practice and efficiency. Even to be again called the 'incorrigible optimist' I now venture to predict that with the equivalent of one bit four holes will be drilled and, again, each

*Bull. 474, September, 1945.

†Formerly Consulting Electrical and Mechanical Engineer of the Central Mining and Investment Corporation, Ltd.

bit will be re-conditioned at least four times before it need be discarded. The economy to be obtained is not so much the saving of drill steel, but the more efficient drilling to be obtained and also what is just as important is the saving of 'machine reaction on the operator'.

It is unnecessary to refer in detail to the other advantages and improved efficiencies—such as, increased life of stem, water feed direct to the stem, as well as the enormous saving in transport and handling costs and the saving daily of shaft-winder hours—all of which constitute important economical factors brought about by the introduction of the detachable bit.

The desirability of having a detachable bit has been fully recognized for probably over 30 years, and many designs have been tried out without full success as far as the Rand practice is concerned. When trying out the Riley bit, which was not economically successful, on one of the Rand mines, the P. and M. bit was brought to our notice. The simplicity of design was attractive and it was decided to make a thorough investigation of its possibilities. Initial drilling tests gave sludging difficulties which were soon overcome by suitable grooving, whilst tests of different skirt thicknesses and depths soon showed the short skirt to be practical.

The following severe test was carried out at the Mechanical Laboratory using a standard $2\frac{5}{8}$-in. jack-hammer, 1,600 blows per minute, 36 ft.-lb. per blow at 60 lb. air pressure with a $\frac{7}{8}$-in. diameter hexagon steel stem 54 in. long and a $1\frac{3}{8}$-in. diameter detachable bit, to drill through two $\frac{1}{2}$-in. mild-steel plates bolted together a hole of approximately $1\frac{1}{2}$ in. diameter in 20 minutes. The bit, although blunted, had to be detached in the usual manner.

All doubts having been dispelled about the skirt we were able to concentrate on the design of a suitable 200-ton press to make bits. It was decided to use the cruciform shape, as it had proved itself in practice. The making of bits on the mines with our locally-made presses is now fully established and is being extended as fast as we can get presses built. To date more than 6,000,000 new bits have been made. The record of drilling up to 10 in. and 12 in. per minute, using the ordinary cruciform design of bit, can be taken as a reasonable standard rate of drilling and therefore, whatever design of bit may be used in the future, this standard of performance will have to be considered.

It is desirable to comment on the author's statement in Section III (C) of the paper—viz: 'In the absence of other variables Intensity of Blow per Unit Length of Cutting Edge (IBULCE) must be a measure of drilling speed'. This requires some elaboration.

Let us consider the standard jack-hammer giving 1,600 blows per minute, each blow having, say, 36/40 ft.-lb. value, and with that power the object in view is to drill a hole in the rock with a depth of, say, 42 in. and a finished diameter of, say, $1\frac{1}{8}$ in. As is well known, the chisel bit is the fastest drilling bit as long as it retains its cutting edge at the periphery. Comparisons of different designs of bits are most interesting, and these designs, mainly to overcome the shortcomings of the chisel bit, which is likely to stick in the hole, are receiving attention, and will, I am sure, meet with success.

The author refers to my suggestion of an even-duty bit and he illustrates certain suggested shapes for different sizes of bits (Table XIV and Fig. 3) and also gives the relative volume of rock to be removed for each $\frac{1}{8}$ in. increase in diameter of hole. These figures are instructive and should be considered when designing a bit to give an *even-duty service* over its entire cutting face.

It is not possible to design such a bit on paper on account of manufacturing requirements, but always aiming at the ideal (IBULCE) of a minimum length of cutting edge across the face and sufficient cutting edge where the greatest work is done, shaped to maintain gauge, I see no reason to doubt the possibility of suitable bits that will be superior to the standard cruciform bit of today.

The author suggests further research and experimental work. I am in full agreement with this and I go so far as to say that it is particularly necessary that such research work must be carried out under scientific control in a properly-equipped laboratory where the controls are standardized and where comparisons of performance, brought about by changes in technique, are more correctly compared. While recognizing the extremely valuable work done on mines in effecting improvements in the past, there comes a time when close scientific control is difficult. It is essential that research work should be under highly-trained technical men, for the border line between success and failure is very narrow.

It is not necessary to discuss the designs and shapes of bits providing we can have the principle of research and experimentation accepted on the lines suggested. During the war there was no possibility of carrying out much research work, but information is now released and various alloy steels can now be obtained. Therefore, it is now necessary for comprehensive tests to be carried out. The points to be settled are—shape of an ideal bit, method of manufacture, and the composition of the steel.

I may now define what in my opinion, is the 'ideal bit'—an even-duty bit, a bit that will drill at a maximum rate of penetration, will drill without losing gauge, and drill three to four holes before it is necessary to re-condition for further use.

I would also suggest the following—

(1) Research work is necessary to determine what alloy steel the stem should be made of to obtain at least 1,000/1,500 drilling feet.

(2) Research work to provide a reliable external water feed, independently attached to the head of the jack-hammer into which the stem can be inserted.

(3) Research to determine whether the present standard $2\frac{5}{8}$-in. jack-hammer with its 1,600 blows per minute is the best machine, or whether a lighter machine with a lighter blow and a greater number of blows per minute would not reduce vibration and reaction on the operator without loss in performance.

(4) Having a lighter jack-hammer, but using a higher air pressure may also be of importance and deserves attention.

AUTHOR'S REPLY TO DISCUSSION*

ON

Tunnelling in Gibraltar during the 1939-1945 War.

By W. H. Wilson, *Member*.

Major W. H. Wilson: I had regarded the compilation of this paper rather as a duty to the Corps and to the Tunnelling Companies ; so much work and effort had been put into the tunnelling system in Gibraltar by so many hundreds of men—forever nameless—that it seemed imperative that a record should be made. From the technical standpoint, too, the Corps would gain by the dissemination of knowledge of the methods used, which—for the most part—were good methods. There was yet another point—as Professor W. R. Jones's erstwhile colleague, a fact in which I take some pride, I was fully aware of the dangers attendant upon the inhalation of mine dusts. I was aware that, while dry-drilling the pure limestone of Gibraltar presented no particular hazard, in some other rocks it would be attended by the greatest possible danger. This warning note had to be sounded, as it was evident that the average non-mining R.E. officer would not know of this distinction. In thanking those who so kindly attended the meeting, and in particular those who took part in the discussion, I do so on behalf of the Tunnellers as a whole.

To answer in turn each of the many points arising from the discussion would provide an uninteresting recital of data and the reply is, therefore, couched in general terms, covering the ground of the discussion in a wide sense, but perhaps missing out one or two of the more minor points, some of which the questioners will find answered on closer examination of the original paper.

In thinking of these tunnels mining men must use a little imagination ; they were not mines, but places where men could—and did—live, sleep, and work. The most elegant excavations made by the Tunnellers were not on The Rock, but in England—for Combined Operations Headquarters—which were complex and elaborate blocks of underground accommodation. On The Rock the work was on a bigger scale, not nearly so elaborate, but, nevertheless, each excavation had a purpose—hospital, workshop and the like—and had its appropriate building within it, which served exactly the same purpose as a similar structure would do on the surface.

Bull. 476, January, 1946.

Accuracy and good workmanship were, therefore, most desirable, and the safety of the back, especially in the larger chambers, of paramount importance. Indeed, before I left The Rock at the end of the war, we were already experiencing trouble with the back of some of the flat spans excavated under the press of urgency in the early years of the war. To ensure permanent safety I am of the opinion that the perfect arching of diamond-drill blasting should be supplemented by arched steel trusses keyed into the walls, carrying a monolithic concrete roofing, rather than that surface-type buildings should be erected underground.

Brig. R. S. G. Stokes has pointed out the difficulty of military costing. The costing of mining in Gibraltar would present extraordinary difficulties. The War Office may know how much the average sapper costs the Government, but I certainly do not, although I have heard mooted the figure of £1 a day. Nor do I know the price to the Government of the various machines and commodities used in the work. While detailed costing is necessary in commerical undertakings, the comparative efficiencies of mining operations in different countries, or at different periods of time, cannot be entirely comprehended in terms of money. The value of money is not steady, while another variable is that the standard of living differs from country to country. Hence the difficulty in correlating costs in one country as compared with another, or from one period to another in the same country.

It is well known, however, that there are certain physical quantities in mining by which efficiency may be estimated and compared. Reduced to the simplest form, two main factors emerge—namely, the tons of rock broken per machine-shift, and the number of underground-shifts needed to keep a machine at work. This later figure, less the number of men required to operate the machine, gives the number of men engaged in loading and tramming, and the more remote tasks, such as packing waste and timbering, are attendant—in one way or another—upon the machine.

Table VIII gives these, and other, data for the more recent operations undertaken on The Rock. For the average machine-shift (two drillers), 15 tons of rock was broken, and 6.6 underground-shifts were worked to provide each machine-shift. Thus, the number of tons broken per underground-shift was 2.3 tons. Each ton of rock required 3.3 ft. of drilling. These figures exclude diamond-drill blasting, which, from its late introduction on a large scale, did not constitute a great proportion of the total tonnage mined.

In lode or banket mines, using the hand-machine, the number of tons broken per machine-shift should not differ radically from the figure given above, as the operations are comparable. The number of feet drilled per machine-shift will vary somewhat according to circumstances, as also will the number of feet drilled per ton of rock. The really important figure is that for the number of underground-shifts per machine-shift. In the normal mine this may vary from 20 to as much as 70. Lack of mechanization is most clearly indicated where this figure is *disproportionately* high, and in such circumstances any gain in drilling technique, or in the drilling task, will be so diluted by the inertia of spoil removal, as to be scarcely perceptible as a gain in overall efficiency.

It is reaffirmed that the drilling task on The Rock was adjusted to fit in with the various cycles of operations. In the 8-ft. by 8-ft. tunnels the footage was too much for one machine, but not quite enough for two, so the later alternative was chosen, to provide a positive 8-hour cycle, without prejudice to the drillers. Since the drillers were less than one-third of the total underground strength, this decision did not materially affect the overall efficiency. Major Moorhead has rightly remarked that this cycle could have been a six-hour one, but I did not follow this practice, as the other cycles could not be similarly dealt with. In the larger tunnels—12 ft. by 12 ft., and 15 ft. by 8 ft.—the task was adequate for three machines, and the cycles would certainly have broken down had an attempt been made to withdraw one of them. Similarly in slashing, see Table VIII, the tasks were unalterable without grossly overloading the cycles.

The use of the drifter, however, would have enabled a substantial increase to be made in the depths of the rounds in the larger tunnels without increasing the driller-shifts and the mechanical loading would have proved elastic enough to cope with the increased quantities of spoil without a proportionate increase in man-shifts. The rounds adopted for the hand-held machines gave about the optimum advances and it would have been most difficult to have increased them with the hand-machine. The faster-drilling drifter would have been needed for deeper rounds, the bigger drills, furthermore, permitting the use of larger diameter cartridges, and thence of the larger quantities of explosive needed for breaking the extra rock.

It will be seen from Table VIII that the diamond-drill accounted for from 50 to 77 tons of rock per shift for 31 ft. of drilling. This work was not on a task basis, as the operators were nearly all learners—our ordinary miners—the diamond-drilling platoon being too busy to take over all of this work. I am of the opinion that, as these men gained experience, and with a 'peace time' supply of machine spares, the footage would have risen. An article just to hand, entitled 'Diamond Drilling as Applied to Ore Production', *Engineering and Mining Journal*, Oct. 1945, gives data of many North American mines using diamond-drill stoping. This is, without doubt, the most exhaustive aggregation of data so far published on this topic. Several of these mines used the same type of machine as was used on The Rock, the average footage drilled per machine-shift being 48 ft., some being well above and others below this mark. When it is borne in mind that we could drill but one hole from each set-up, that the holes were unusually long, and that some of the set-ups were in most awkward places, it will be appreciated that the figure attained was not altogether unreasonable.

The tonnage of from 50 to 77 per machine-shift is no mean tonnage, especially as the method of mining eliminated the loading and tramming operations that consumed some two-thirds of our underground strength, as the lorries, instead of loading at a box, drove right into these very big excavations, and were there loaded by ordinary surface excavators.

The alternative to diamond-drill blasting in these chambers would have been benching from the top downwards in high chambers, or undercutting and blasting down the back with the hand-machine, as described for certain of the tunnels, in low chambers. The former method is a good one, but it is neither particularly easy, nor very economical.

The critical need for diamond-drilling arose in the following manner. A chamber, originally planned at 50 ft. wide and 19 ft. high, and under-cut at some 12 ft. high in preparation for blasting down the back using the hand-machine, was re-planned at 34 ft. high. The under-cutting ruled out the possibility of benching, and to increase the height from 12 ft. to 34 ft. with the hand-machine seemed to present overwhelming difficulties. The problem of diamond-drilling was, therefore, studied, and Brig. Nutt, after consideration of the proposed method and estimates, had sufficient confidence in us to order the necessary machines and special explosives. The project entailed a great deal of anxiety, and we were unfortunate in suffering two casualties, but the method was certainly established. That my R.E. predecessors did not use the excellent weapon to hand in the diamond-drill may be attributable to the fact that they were not faced with this particular problem, nor with that of the much bigger chambers illustrated in Fig. 17, planned as a result of the successful completion of the smaller ones.

The latest news from The Rock regarding tungsten carbide bits is that the tests being made there are held up, as the solid type bits have not yet been received. Research, however, has been in progress in this country and on the Continent, and it is understood that results may soon be published.

In my opening remarks on the discussion, I ventured to doubt whether the design of the buckets of the R.B. excavators was correct for digging the large, angular spoil from the diamond-drill blasts. Since no commentary has been forthcoming on this point, I am impelled to amplify my remarks, although I had much preferred the more expert commentary of a specialist in excavating machinery. The fault with the R.B. bucket is that all the spoil must pass through its hinged bottom and that quite small rocks can jam between the teeth of the bucket and across the bucket opening. In contrast, the Eimco bucket tips out its contents and is, therefore, able to deal with rocks, when the necessity arises, a good deal bigger than the bucket size. From North American literature it would appear that excavator buckets are available which have a hinged, tipping motion. Such a bucket, based upon a sturdy body, with a small radius of swing, is what is needed for this special underground work. The lorry, too, might be replaced by a more squat vehicle, so that the boom height of the excavator could be less.

A small proportion only of the diamond-drill spoil needed secondary blasting. While this blasting was carried out in one chamber, the excavator loaded in others, thereby minimising the inconvenience caused. Most of the big slabs which Brig. Stokes has discerned in Fig. 23 were so fractured in the blast, that a prod from the excavator bucket broke them up.

Electric detonation was used for these big blasts since it was our standard practice, from which there was no reason to depart. It is suggested that, if it is required to detonate more than two or three banks of holes, with an instantaneous effect for each bank, detonating fuse could be used for the banks, with electric detonators in successive delays to initiate the fuse. The danger in such a method would be that the air blast from the first bank of holes might break the fuse in the others.

Professor Ritson emphasizes that the 'business' end of the detonator should always point towards the charge. One of the advantages of the electric detonator is that it can be placed at the bottom of the drill-hole (less one cartridge as a 'buffer'), a practice regarded as unsafe in fuse blasting, as the fuse would have to burn down the side of the explosive. In safety fuse blasting the detonator is, therefore, placed at the top of the hole, less one cartridge as a buffer to the stemming.

Evidence accruing from numerous mines indicates, contrary to Brig. Stokes' fears, that the effect of diamond-drill blasting is not prejudicial to the normal running of a mine. On The Rock the air-blast was caused principally by the ' bellows ' effect of the back falling and expelling the air from the chamber. When the back was blasted over a length of 261 ft. by a width of 50 ft., falling some 20 ft. (see Fig. 11) the quantity of air moved was some 261,000 cu. ft. The blast emerging from the 12 ft. by 12 ft. adit entry was by no means commensurate with this figure, so it would appear that some remarkable compression and eddy effects must have taken place within the chamber. Another point of interest is that the weight of rock blasted on this occasion was some 10,000 tons, so that some 200,000 ft. tons of energy was expended in a split fraction of a second, from the fall alone. When the electric switch was thrown over there was a fairly loud rumble, a rush of air out of the entry, and a very light seismic shock.

The amount of explosive used per ton of rock in diamond-drill blasting is the smallest attainable in underground mining. This may be one of the reasons why the back is left in a perfect condition. The amount of barring-down needed after blasting is relatively small, and spot levels taken along the arch vary by a few inches only.

Lt.-Col. Williams remarks that the methods described pertain to the post-1943 period. I had made this point in my Introduction and in my opening remarks, but in a more general way. The diamond-drilling technique, the standard rounds employed (formerly all tunnels of large cross-section were driven at 8 ft. by 8 ft. and slashed to size, except those driven by the Canadian Company), underground spoil boxes, the use of hole directors, detachable bits and of electric detonation, all belong to this last phase of the mining, and brought the efficiency almost to the calculated limit. Any further advance would have entailed the introduction of new methods.

To Professor Ritson's and Mr. S. H. De La Mare's queries respecting the fan tests, I would say that while nearly all the readings were made by myself, the constancy of the air pressure from test to test may have varied, as we had no master air gauges, and different fans were used on the various tests, the efficiencies of which would not be quite the same. The iron ducting was 12 in. in diameter, and each length telescoped some 12 in. into the adjacent length. The conclusions to be drawn from the tests were unmistakable and guided all subsequent ventilation of ' development ' ends.

When the systems of chambers were developed, ventilation no longer depended upon fans. By means of draught-doors across certain of the entries, natural air currents were deflected into the chambers, of strength sufficient to obviate the need for fans or for exhaust conditioners on the excavators and lorries operating within the chambers. Such conditioners were needed only on the locomotives servicing development ends. Conditions of temperature and humidity were excellent throughout the year.

Mr. A. F. Skerl will find details of electrically-driven diamond-drills in an article entitled 'Electric Core Drills Stope Soudan's Hard Ore'.* In answer to his second question, the drillers preferred the smallest possible sizes of Rip-Bits, as their use speeded up drilling and decreased the manual effort involved.

In replying to Mr. A. Alec Jones, servicing and all but major overhauls of underground machinery was carried out underground, a large electrically-lighted chamber being set aside for this purpose. On the survey side, and again in answer to Mr. Alec Jones, the symmetrical lay-out of the tunnels made it comparatively easy to find the survey stations. Professor Ritson's bore-hole survey instruments are much too large to enter these small bore-holes. In this technique of blasting, however, I should always recommend the second, or inspection slot, which gives a flat, well-trimmed end to the chamber; and since this procedure was followed, the question of bore-hole survey instruments did not arise.

A detailed study of the geology of The Rock was made in the early years of the war by Sgt. A. L. Greig, and later by Capt. C. B. Alexander, both well-qualified geologists. In the lay-out of the tunnels it was necessary to avoid major faults and broken ground as much as possible and it was desirable that the larger chambers should run parallel to the dip of the strata. Capt. C. B. Alexander pointed out that caverns and solution cavities would be most pronounced on those horizons corresponding to the several raised beaches on The Rock. I am grateful to Mr. Jones for bringing this important matter forward.

I feel that members will appreciate further information on the cave—at present, I believe, unnamed—mentioned by Professor W. R. Jones. This almost inaccessible sea cave, in the eastern, precipitous face of The Rock, had been entered in 1907 by a Capt. Goram, who had left an inscription painted on the wall. It was re-discovered in 1945 by two R.E.M.E. sappers, whose names I do not recall, and reported by them to the Rev. F. E. Brown, who is

* *Eng. Min. Journ.*, Oct., 1945.

probably the greatest scholar on The Rock, and is well versed in the archæology of southern Spain. The Rev. Brown contacted me, as I was then O.C. Tunnellers, and showed me some unbroken pottery and a piece of ore—subsequently identified by Professor Jones—from the cave. We both realized the outstanding importance of the discovery, of which the Governor, Sir Ralph Eastwood, was then informed. In due course the newly-arrived geologist, Capt. Alexander, took charge of operations, and displayed considerable skill in making the preliminary examination and report to the British Museum.

We all accompanied the Governor on the first visit, which was made easy by the use of his launch. The cave extended back some hundreds of feet, and the floor, covered with many inches of soft, powdery guano, was raised some 70 ft. above the sea, and consisted of raised beach material, the cross-section of which could be examined as we climbed up from the sea to the entry. Under the guano, and in the first few inches of the floor, was a great deal of broken pottery, estimated to run into tons in weight, and down the steep face, dropping to sea level, flints and bones were found. The floor was marked and mapped by Capt. Alexander, and the pottery was gathered up over a small area. These fragments, when washed and cleaned, were sorted according to colour and texture and soon the outlines of pottery vessels were built up.

The importance of southern Spain and of Gibraltar in the study of neolithic remains lies in the fact that the advancing glaciers of the Ice Age drove life down to the Mediterranean coast, where the sea formed a then impassable barrier, isolating Europe from Africa. That mammoth bones and bronze rings have now been found in this cave is most interesting, but some 30,000 tons of deposit still await examination in what may well prove to be one of the most remarkable finds in Europe. Other finds of fossil bones, principally of a species of carnivorous deer, were made during the tunnelling operations, and another notable discovery, made in the early part of the war, was of a series of gigantic underground caverns—now known as Lower St. Michael's Cave.

On this note of interest, coupled with a word of thanks to Lt.-Col. D. M. Thomson, for his human remarks, I close this brief account of tunnelling on The Rock. It was undoubtedly D. M. Thomson's genial and forceful disposition, together with the foresight of Lord Gort and Lt.-Gen. Sir Noel Mason Macfarlane, that promoted the initiation of this great work.

W. H. Wilson. *Tunne*
1939-

Operation	Tons per Machine-Shift
Machine :—Hand-held, 2 dril	
Tunnel, 8 ft. by 8 ft., see Fig. 5 of paper...............	9·2
Tunnel, 12 ft. by 12 ft., see Fig. 6	17·2
Tunnel, 15 ft. by 8 ft., see Fig. 7...	14·2
Dropping the back to raise the height of a 15-ft. by 8-ft. tunnel to 13 ft., see Fig. 7	56·7
Dropping the back of a semi-circular chamber of 14-ft. radius, see Fig. 8	46·3

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The Institution of Mining and Metallurgy

(*Founded* 1892—*Incorporated by Royal Charter* 1915.)

Bulletin No. 478.

MAY 9TH, 1946.

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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, MAY 16th, 1946, at 4 o'clock p.m.**

AGENDA

Part I (*open to Members and Associates only*) :

(1) Minutes of the previous Annual General Meeting.
(2) Appointment of Scutineers to examine Balloting Lists.
(3) Announcement regarding Benevolent Fund.
(4) Report of Council and Accounts for 1945.
(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, Associates, Students and Visitors*) :

(7) Presentation of Awards :
  (*a*) The Gold Medal of the Institution.
  (*b*) Certificate of Honorary Membership.
  (*c*) 'The Consolidated Gold Fields of South Africa, Limited' Gold Medal and Premium of Forty Guineas.
(8) PRESIDENTIAL ADDRESS by MR. G. F. LAYCOCK.
(9) Report of Scrutineers on election of Members of Council.

NOTICE OF GENERAL MEETING.

The FIRST ORDINARY GENERAL MEETING of the Fifty-Sixth Session of the Institution of Mining and Metallurgy will be held, also in the Apartments of the Geological Society, on THURSDAY, JUNE 20th, 1946, at 5 o'clock p.m., when the following Paper, of which a copy is attached, will be submitted for discussion :

Sandfilling at Mufulira.

By A. C. TURTON.

Light refreshments will be provided at 4.15 p.m. for members and friends attending the Meeting.

FIFTY-SIXTH SESSION : 1946–1947.
DATES OF SUBSEQUENT MEETINGS.

The following dates have been provisionally fixed for subsequent General Meetings of the Institution during the Session 1946–47 :

1946.	1947.
Thursday, September 19th.	Thursday, January 16th.
,, October 17th.	,, February 20th.
,, November 21st.	,, March 20th.
,, December 19th.	,, April 17th.
	,, May 15th.

ROYAL ENGINEERS WAR MEMORIAL.

The following announcement is published at the request of the Chief Royal Engineer:

The Corps of Royal Engineers have decided to provide a War Memorial to commemorate the deeds of the Corps in the Second World War and to honour those who have fallen. An appeal for subscriptions has been circulated to all units in the Corps, but it is certain that it did not reach many men before they returned to civil life.

It is intended to provide a memorial of a beneficent type, such as cottage homes for the disabled, a convalescent home, or endowed beds in hospitals, from which ex-soldiers of the Corps can benefit.

It is hoped that members of the Institution who served in the Royal Engineers and have not already subscribed will avail themselves of this opportunity of contributing towards a Memorial which will stand for all time to commemorate our gallant comrades who made the supreme sacrifice.

Subscriptions should be sent to The Honorary Treasurer, Royal Engineers War Memorial Fund, Gibraltar Barracks, Aldershot.

SPECIAL LECTURES ON VENTILATION PROBLEMS IN MINING.

A series of four lectures on ventilation problems in mining has been arranged by the Royal School of Mines and will be delivered in their Mining Department at Prince Consort Road, London, S.W. 7. The first lecture, on 'The effect of humidity on working efficiency', will have been delivered by Professor David Brunt, F.R.S., before the publication of this *Bulletin*, but the others are as follows:

Tuesday, May 14th: 'Dehumidification of mine air', by Jack Spalding, A.R.S.M., Assoc. Inst. M.M.;

Tuesday, June 18th: 'The theory of air flow in mines', by Professor I. C. F. Statham, M.Sc.;

Thursday June 27th: 'The use of booster and auxilliary fans in mines', by Professor I. C. F. Statham, M.Sc.

The lectures will be given at 4.30 p.m., and no charge will be made for admission.

'FRENCH NORTH AFRICA': REPORT OF MINERALS DIVISION, BRITISH ECONOMIC MISSION, ALGIERS.

This report, by three members of the British Economic Mission, was prepared with the object of presenting to the Ministry of Supply information and views gained during a period of nearly three years of intensive work in North Africa. The authors, Messrs. A. H. Steedman (Chief, Minerals Division), Donald Gill, M.Inst.M.M., and André Choubersky, Assoc. Inst. M.M., have, with the Ministry's permission, presented a copy of the report to the Library of the Institution, where it is available to members.

The report consists of over 100 stencilled foolscap pages, bound with 39 maps and plans, and there are 25 Appendices bound separately and illustrated by a number of plans. A résumé of the work of the Minerals Division and descriptions of the political and economic background of French North Africa are followed by chapters on the phosphate and iron ore industries and other minerals, and there are sections devoted to strategic minerals unsuccessfully investigated, metallurgical residues, and a forecast of possible future developments. A detailed directory of mines and mineral occurrences is included, and the mines are marked on one of the maps, a separate coloured copy of which is also available. Additional notes on many of the mines visited are contained in the Appendices. Useful information on ports and roadsteads and a chapter of statistics closes the report.

BALLOTING LIST FOR ELECTION OF MEMBERS OF COUNCIL.

The Council wish to draw the attention of Members and Associates to Section IV, clause 6, of the By-Laws, which contains the following provision: 'The Council shall receive the name of any Member or Members, submitted in writing by any Member or Associate previous to November 1st in any year, and shall decide by ballot upon the inclusion thereof, or otherwise, in the balloting list'.

Any Member or Associate who wishes to suggest a name for inclusion in the balloting list for the election of Members of Council for the Session 1947–48 is requested to notify the Secretary of the Institution before November 1st, 1946.

LIBRARY SERVICE.

The Library has now been brought back to London, and applications for books should be addressed to the Librarian, I.M.M. and I.M.E. Joint Library, 424, Salisbury House, London, E.C.2. Books at present on loan should of course be returned to this address. Members who are unable to visit the Library and borrow books in person may still borrow them by post. It is regretted that periodicals cannot be lent.

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 March 14th, 1946:—

To Membership—

Chappel, John Traer (*Harrow, Middlesex*).
Dannatt, Cecil William (*London*).
Hawes, James Thomas (*La Carolina, Spain*).
Wardrop, John George Lessels (*Kirtlebridge, Dumfriesshire*).

To Associateship—

Bath, Silas James (*Camborne, Cornwall*).
Birkbeck, James Martin (*Royal Engineers*).
Clark, William Lionel Gladwell (*Johannesburg, Transvaal*).
Dunstan, Arthur Stanley (*Camborne, Cornwall*).
Edwards, Tom (*Johannesburg, Transvaal*).
Fitch, Frederick Harry (*London*).
McCormick, David (*Nelson, New Zealand*).
Monro, Donald Francis (*London*(.
Walker, Stanley Edmond (*Selukwe, Southern Rhodesia*).

The following have applied for admission into the Institution since March 14th, 1946:—

To Membership—

Giegerich, Joseph R. (*Kimberley, British Columbia*).
Kirkpatrick, William Stafford (*Trail, British Columbia*).
Osborn, George Howard (*Aylesbury, Buckinghamshire*).

CANDIDATES FOR ADMISSION—*continued.*

To Associateship—

Arthur, John Albury (*West Rand, Transvaal*).
Bhatt, Gunvantrai Vrajalal (*Calcutta, India*).
Gobert, Mancil Joseph (*Selkirk, Manitoba, Canada*).
Jacouris, Constantinos Anthony (*Mbarara, Uganda*).
Krishnaswamy, Erapally (*Champion Reefs, South India*).
Napier, Alexander Nicol (*Aboso, Gold Coast*).

To Studentship—

Daniel, Kenneth Edward (*Birmingham, Warwickshire*).
Earl, Stephen James (*London*).
Edwards, Richard Charles John (*Bushey, Hertfordshire*).
Follows, Edward Arthur (*Birmingham, Warwickshire*).
Goodwin, James (*Derby*).
Quarm, Thomas Alfred Arthur (*London*).

TRANSFERS AND ELECTIONS.

The following have been transferred (subject to confirmation in accordance with the conditions of the By-Laws) since March 14th, 1946 :—

To Membership—

French, Jack Hayward (*Klerksdorp, Transvaal*).
Grey, Donald William John (*Morro Velho, Brazil*).
Lamb, Cyril Leonard (*Springs, Transvaal*).
Lethbridge, Robert Frederick St. George (*Henley-on-Thames, Oxfordshire*).
Rickard, René Emile (*Manchester, Lancashire*).

To Associateship—

Calhoun, Clarence (*Llangollen, Denbighshire*).
Greenberg, Reginald (*Johannesburg, Transvaal*).
Peck, Arthur (*Longbenton, Northumberland*).
Terry, Leo (*Bulawayo, Southern Rhodesia*).
Thurston, Frank Edward (*Nsuta, Gold Coast*).
Webb, John Stuart (London).

The following have been elected (subject to confirmation in accordance with the conditions of the By-Laws) since March 14th, 1946 :—

To Membership—

Warden, Thomas Wilkie Muir (*London*).

To Associateship—

Dorrell, William John (*Kisumu, Kenya*).
Ellis, William John (*Greymouth, New Zealand*).
Foley, Thomas Michael (*London*).
Fletcher, Frank Frederick Flitcroft (*Gravesend, Kent*).
Harding, Ernest Fleetwood Stringfellow (*Kitwe, Northern Rhodesia*).
King, Andrew Ian (*Barberton, Transvaal*).
Kirkup, John Leadbitter (*Alston, Cumberland*).
Mann, Alfred (*Bukuru, Northern Nigeria*).
Mann, Marshall Douglas Scott (*Wankie, Southern Rhodesia*).
Morris, Donald Henry (*Ashby-de-la-Zouch, Leicestershire*).
Nixon, William James (*Mufulira, Northern Rhodesia*).

TRANSFERS AND ELECTIONS—*continued.*

Polglase, Philip Henry (*Sheffield, Yorkshire*).
Simon, René Oakley (*Tonbridge, Kent*) (reinstatement).

To Studentship—

Dogan, Mustafa Zelşi (*London*).
Hallé, Charles Edwin Henton (*Manchester, Lancashire*).
Pegg, Charles William (*London*).
Spence, Wilfrid Ian (*Vereeniging, Transvaal*) (reinstatement).
Stephens, John Nolan (*Mosgiel, New Zealand*).
Taute, Andries Hendrik (*Erfpacht, Cape Province*).

MEMBERS ON SERVICE WITH H.M. FORCES.

A full list was published in Bulletin *No.* 456, *September*, 1942. *The following additions or changes are supplementary to those already published.*

Associates.

Major J. Bowen M.C., *Royal Engineers* (Promoted).
Lieutenant E. B. Davies, *Royal Artillery.*
Sergeant L. Dobson, *Royal Army Medical Corps* (Promoted).
Colonel D. T. Hudson, D.S.O. *General List* (Awarded the O.B.E.).
Major F. J. B. Somerset, *Royal Engineers.*

NEWS OF MEMBERS.

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

Professor F. Stuart Atkinson, *Member*, has resigned his position at Leeds University to take up a post with Messrs. Shaw, Wallace & Co., Calcutta.

Mr. A. Bean, *Member*, has left England to resume his former duties at Kuala Lumpur with the Mines Department, Malaya.

Mr. P. Best, *Associate*, is returning to England on furlough from Mysore.

Mr. C. E. Blackett, *Member*, has resigned the position of general manager, New Occidental Gold Mines, N.L., Cobar, N.S.W.

Lt.-Col. H. H. W. Boyes, *M.C.*, *Member*, has arrived in England from Nigeria.

Mr. J. K. Broadhurst, *Associate*, has been released from the Australian Imperial Forces and expects to arrive in England shortly on leave.

Mr. M. A. Brooke, *Associate*, has been demobilized from the Australian Imperial Forces and has returned to Malaya.

Mr. H. F. Burton, *Student*, has joined the staff of the Bisichi Tin Co. (Nigeria), Ltd.

Mr. F. A. Campbell, *Student*, has completed his engagement in Fiji and has resumed work with Gold Mines of Australia, Ltd.

Mr. W. J. Carlyle, *Student*, has been released from the R.A.F. and is now on demobilization leave.

Mr. F. M. Chapman, *Associate*, has left Australia and has joined the staff of Anglo-Oriental (Malaya), Ltd., at Kuala Lumpur.

Mr. L. A. Crozier, *Associate*, has arrived in Bolivia from Australia and has joined the staff of Patiño Mines & Enterprises Consolidated (Inc.).

NEWS OF MEMBERS—*continued.*

Mr. A. Ctvrtecka, *Associate*, expects to leave England soon to join the staff of Frontino Gold Mines, Ltd.

Mr. K. V. Cunliffe, *Associate*, formerly of Randfontein, has joined the staff of African Associated Mines, Ltd., Bulawayo.

Mr. E. V. Dabb, *Associate*, is leaving England for Malaya.

Mr. E. J. Daniel, *Associate*, has been released from the Indian Engineers and has resumed his appointment with Iraq Petroleum Co., Ltd.

Mr. K. H. Davison, *Associate*, has arrived in England on three months' leave from Argentina.

Mr. W. Douch, *Associate*, has joined the staff of Mawchi Mines, Ltd.

Mr. G. H. Fairmaid, *Member*, has left England on his return to Pahang Consolidated Co., Ltd. Malaya.

Mr. F. H. Fitch, *Student*, has returned to England from Australia where he was on leave after release from a Japanese prisoner-of-war camp. He expects to resume work shortly with the Geological Survey of Malaya.

Mr. R. Fleischman, *Student*, has joined the staff of New Occidental Gold Mines, N.L., Cobar, N.S.W., as assistant metallurgist.

Mr. A. S. Fong, *Student*, having been released from internment, has rejoined the staff of Anglo-Oriental (Malaya), Ltd.

Mr. W. A. A. Freeman, *Student*, has been demobilized from the Australian Imperial Forces and is taking up employment in Colombia with Placer Development, Ltd.

Mr. W. P. Gaskell, *Associate*, has returned to England on leave from Nigeria.

Mr. K. B. Goode, *Member*, has returned to Mawchi Mines, Ltd., Burma.

Mr. R. C. J. Goode, *Associate*, has been demobilized and has returned to Geduld Proprietary Mines, Ltd.

Mr. P. G. J. Gray, *Associate*, has been demobilized and has left England for Rio Tinto, Spain.

Mr. S. V. Griffith, *Member*, has left England to join the staff of London and African Mining Trust, Ltd., Gold Coast.

Mr. J. M. B. Gundry, *Student*, has arrived in England on leave from the Transvaal.

Mr. G. L. Hatherly, *D.F.C.*, *Student*, has been demobilized from the R.A.F. and is in England.

Mr. J. Hays, *Student*, has been demobilized from the Royal Artillery.

Mr. H. Hocking, *Associate*, has arrived in England from Australia where he has been recuperating since his release from internment.

Mr. R. C. A. Hooper, *Student*, is returning to England on leave from the Gold Coast.

Mr. S. Ramplen Jones, *Associate*, who was interned at Singapore, has resumed his consulting practice at Kuala Lumpur.

Mr. H. E. Josselyn, *D.S.C.*, *Associate*, has been released from the R.N.V.R. and has been transferred to the Malayan Civil Service.

Dr. N. R. Junner, *O.B.E.*, *M.C.*, *Member*, has retired from the Gold Coast Geological Survey and has been appointed Consulting Geologist to Selection Trust, Ltd.

Mr. R. John Lemmon, *Member*, has terminated his appointment as H.Q. Inspector, Non-Ferrous Section of the Inspectorate of Fighting Vehicles.

NEWS OF MEMBERS—*continued.*

Mr. BEN LIGHTFOOT, *Associate*, has retired from the Geological Survey f Rhodesia and is returning to England.

Mr. C. W. LOCH, *Member*, has relinquished his post as Assistant Manager t a Royal Ordnance Factory in Lancashire.

Dr. R. B. McCONNELL, *Associate*, has returned to England on leave rom Tanganyika.

Mr. J. A. McINTYRE, *Associate*, formerly a prisoner of war in Singapore, as returned to Sydney, N.S.W., for recuperation and a holiday.

Mr. G. A. P. MOORHEAD, M.B.E., *Associate*, has been appointed Com- nissioner of Lands and Mines, British Guiana.

Mr. D. A. O. MORGAN, *Student*, has been demobilized and has resumed his tudies at the Royal School of Mines.

Mr. R. MURRAY-HUGHES, *Member*, has left England on a short visit to Yugoslavia.

Mr. P. I. A. NARAYANAN, *Associate*, has left the Geological Survey of ndia to take up the post of mill superintendent in the Zawar lead-zinc nines.

Mr. A. W. NORRIE, *Associate*, has been discharged from the R.A.A.F., nd after visiting New Zealand expects to go to Canada.

Mr. W. P. H. PARKINSON, *Student*, has joined the staff of the United teel Co., Ltd., at their Risehow colliery.

Mr. C. H. RICHARDS, *Member*, hopes to be in England shortly on leave rom Uruwira Minerals, Ltd., Tanganyika.

Mr. J. A. RICHARDSON, *Associate*, has joined the staff of N. V. de ataafsche Petroleum Maatschappij at Lensbury, preparatory to going broad later this year.

Mr. BRYAN ROE, *Associate*, is leaving England to join the staff of the onsolidated Tin Mines of Burma, Ltd.

Mr. N. R. SCHINDLER, *Associate*, has discontinued his private consulting ork and is taking up an appointment with the General Mining and Finance orporation of Johannesburg at their West Rand Consolidated mine.

Mr. G. A. SCHNELLMANN, *Associate*, expects to be demobilized shortly nd is returning to England from India.

Mr. F. G. SHARP, *Associate*, who has been assisting Champion Reef old Mines of India, Ltd., has now returned to Mysore Gold Mining Co., Ltd.

Mr. H. SIMON, *Associate*, has left Crown Mines, Ltd., on his appointment manager, Eastern Section, Consolidated Main Reef Mines & Estate, Ltd.

Mr. R. H. SKELTON, *Member*, has left England on a visit to Australia.

Mr. H. W. SMITH, *Associate*, who has been seconded from U.N.R.R.A. to e British Economic Mission to Greece, expects to return to England on ave in July.

Mr. R. SMYTHE, *Associate*, has been demobilized and after leave he will turn to Nundydroog Mines, Ltd., Mysore State.

Mr. G. STARMANS, *Associate*, has recently been appointed assistant gineer-hydrologist to the Public Works Department, Kenya.

Brig. R. S. G. STOKES, *C.B.E.*, *D.S.O.*, *M.C.*, *Member*, has left England a two-months' visit to the United States, the West Indies and Venezuela.

Mr. W. E. STOREY, *Associate*, having been demobilized, has joined the aff of Rose Deep, Ltd., Transvaal.

NEWS OF MEMBERS—*continued.*

Mr. J. E. STRUTHERS, *Associate*, has joined the Department of Mines and Works, Gwelo, as District Metallurgist.

Mr. R. TEALE, *Student*, has arrived in England on his return from Sierra Leone.

Mr. K. L. G. TERRELL, *Associate*, has left England to join the staff of Frontino Gold Mines, Ltd.

Mr. C. T. THOMAS, *Associate*, has left England on his return to Malaya.

Mr. L. VAUGHAN, *Member*, has returned to England from Malaya, where he has been employed on the Chinese mining rehabilitation scheme.

Mr. F. H. WAY, *Associate*, formerly with the Pahang Consolidated Co., Ltd., and interned by the Japanese, has arrived in England.

Mr. J. S. WEBB, *Associate*, was awarded a Beit Scientific Research Fellowship at the Royal School of Mines in 1945.

Mr. J. WEEKLEY, *O.B.E.*, *Member*, expects to return to Malaya shortly.

Mr. P. WESTERBERG, *Associate*, has been released from the Army to take up a temporary position as Inspector in the Lands, Mines and Surveys Department of Kenya.

Mr. J. D. WILLSON, *Associate*, has left England to take up an appointment with Beralt Tin & Wolfram, Ltd., Portugal.

Mr. L. WILTON, *Student*, has left England to join the staff of Frontino Gold Mines, Ltd.

INFORMATION REQUIRED.

The Secretary would be glad to receive news of the following Members, Associates and Students from whom no communications have reached the Institution for several years :

A.—*Reported as having been prisoners of war or internees.*

BURMA : Sgt. A. J. Bell, *Associate.*

GERMANY : Lt. the Hon. D. C. Feilding, R.E., *Student* ; Lt. R. H. S. Haydon, R.E., *Associate* ; Lt. J. E. Kernick, R.E., *Student.*

JAVA : M. C. FitzHerbert, R.A.F., *Student.*

MALAYA : H. A. Coates, *Member* ; J. D. Kemp, *Associate* ; Pte. L. I. Watt, F.M.S.V.F., *Associate.*

SIAM : L/Cpl. W. L. H. Morrison, F.M.S.V.F., *Associate.*

B.—*Members whose last address was in an enemy or enemy-occupied country.*

BELGIUM : J. G. J. Parfondry, *Associate.*

BURMA : F. W. Jackson, *Associate*; G. L. Loonba, *Student* ; C. J. J. Reed, *Associate* ; G. C. J. Rotter, *Student* ; G. C. Walters, *Associate.*

CHANNEL ISLANDS : R. B. D. Jackson, *Student* ; C. P. Journeaux, *Student.*

CHINA : K. Y. Kwang, *Member.*

CZECHO-SLOVAKIA : Prof. B. Stoces, *Member.*

DENMARK : R. Underwood Jarvis, *Member.*

GERMANY : P. Trotzig, *Member.*

HOLLAND : M. A. Provily, *Student.*

ITALY : A. Lheraud, *Associate* ; J. A. Nogara, *Associate* ; F. J. Wydler-Hollis, *Associate.*

JAPAN : H. Hunter, *Associate.*

INFORMATION REQUIRED—*continued.*

KOREA : K. I. Yun, *Member.*

MALAYA : R. P. Brash, *Member* ; A. Burns, *Student* ; H. B. Hall, *Associate* ; W. J. D. Kloezeman, *Associate* ; P. O. Shiel, *Associate* ; J. Thomson, *Student* ; E. J. Vallentine, *Member* (believed to have been in New Zealand recently) ; C. W. Wicks, *Associate* ; A. T. Wood, *Student.*

MANCHURIA : B. J. Bryner, *Associate.*

NETHERLANDS EAST INDIES : T. Haden, *Associate* ; P. J. Jansen, *Member* ; N. J. Kuiper, *Associate* ; A. B. M. Meyer, *Student.*

NORWAY : S. Blekum, *Associate* ; N. E. Lenander, *Member.*

PHILIPPINE ISLANDS : M. M. Aycardo, Jr., *Student* ; R. G. Bergman, *Associate* ; F. Garrido, *Member* ; A. B. Rowe, *Associate* ; A. Wellhaven, *Member.*

SIAM : A. Buranasiri, *Student* ; A. L. Fredericks, *Associate* ; R. M. Hannah, *Student* ; C. C. W. Liddelow, *Member*; P. Sukhum, *Associate*; R. J. Sumner, *Associate.*

OBITUARY.

Horace Percival Allwright died at the Royal Infirmary, Derby, on December, 26th 1945, at the age of 45. He served in the Forces in the 1914-18 War and was subsequently employed in Government administrative service in Baghdad. In 1922 he went to Nigeria for Messrs. John Holt & Co. (Liverpool), Ltd., and later worked both there and on the Gold Coast for a firm of motor engineers. In 1933 he began alluvial gold mining activities on his own account and in 1935 he formed Potashi Gold Estates, Ltd., of which he was general manager and director. He continued in this work, with the exception of a period of war work on Nigerian tin mines, until March, 1945, when he became very ill and spent some months in hospital before being invalided home to England.

He was elected to Associateship of the Institution in 1945.

Jehu Berry, a Rand Pioneer, died in Johannesburg on February 17th, 1946, at the age of 79. Born in Rochdale, Lancashire, he served an engineering apprenticeship at the Vulcan Foundry, Newton-le-Willows, and then spent five years in the U.S.A., for three-and-a-half of which he was working at the Union Iron Works, San Francisco. He then went to South Africa and was at first engaged as an assistant engineer during the erection of plant at the City and Suburban and Robinson Gold Mining Companies. From 1893 to 1905 he was chief mechanical engineer to various gold mines, including Crown Deep, Jupiter, Simmer and Jack West, South Geldenhuis, and Robinson Deep, and was a member of the Board of Examiners of the Transvaal Mines Department. In 1905 he went to the Gold Coast as general manager of the Abbontiakoon, Effuenta and Fanti Mines in the Tarkwa district, and subsequently took a similar position at Prestea Block A mine. After several years he returned to South Africa and from 1912 to 1914 conducted a consulting practice in Johannesburg. In the latter year he came to England and discovered oil-shale deposits at Kilve, Somerset. During the period 1915–19 he was general manager of the Wantage Engineering Works in Berkshire, and on leaving this appointment he erected con-

OBITUARY—*continued.*

denser plant at Kilve and extracted oil from the shale he had previously found there. He subsequently designed and erected rotary oil retorts at Hythe for the Estonia oilfields. Mr. Berry travelled extensively in Japan, China and Canada, and finally retired and lived in South Africa, retaining a keen interest in mining ventures. He was a Member of the Institution from 1908 until his retirement in 1943.

Maxwell Campbell Corbett died in Mexico City on June 23rd, 1944, at the age of 58. He was born in New Zealand and received his training at Auckland University College School of Mines. He went to Perak, Malaya, at the beginning of 1910, on his appointment as assistant manager to Separators, Ltd. In November, 1911, he also took up the position of assistant manager of the New Leh Chin Mine, Ltd., which he left a year later to become assistant to Jabus (Tin) Hydraulic Elevators, Ltd., six months afterwards accepting the position of manager and attorney. He was also at that time manager of the Chenderiang Valley Tin Dredging Co., Ltd., and on the amalgamation of this company and Jabus (Tin) Hydraulic Elevators in 1914 he relinquished his appointments with them, remaining with Separators, Ltd., as general manager and attorney and taking up the position of manager and attorney of the New Leh Chin Mine, Ltd. For two years, from 1913 to 1915, he was engaged in the temporary management of Ipoh Tin Dredging, Ltd., entailing the general preparation of the company's site. In November, 1914, he was appointed general manager of the Ulu Piah Co., Ltd., in Perak, which position he held for four years until ill health caused him to resign and visit Australia, leaving the employ of Separators, Ltd., at the same time, but continuing to hold his position with the New Leh Chin Mine until November, 1919. He then turned his attention to a property for the development of which a company called Johan Tin Dredging, Ltd., was formed, with Mr. Corbett as manager. Details of his subsequent work are not known, but it is believed that he remained in Malaya until the Japanese invasion. Mr. Corbett was elected to Studentship in 1909, and was transferred to Associateship in 1919.

Hugh Ostle Crighton died on February 15th, 1946, at Kampala European Hospital, Uganda, at the age of 67. He was educated at Bedford Grammar School and Lausanne, Switzerland. In 1899 he took up employment as assistant to the general manager of Burrakur Coal Co., Ltd., at their Gourangdi colliery, Bengal, three years later being appointed assistant in charge of their Jamgram colliery. In 1904 he returned to England and took a course in Mining at the Camborne School of Mines, obtaining a first class Diploma in 1906. He went to Perak, F.M.S., in August of that year, where he held the positions of assistant manager, and subsequently manager, of the Pusing Lama tin mines, leaving in 1909 to become general manager in Nigeria for the Nigerian Tin Corporation, Ltd. In 1915 he was commissioned in the Royal Engineers and served with a Field Company in Mesopotamia; he was appointed acting adjutant, R.E., 17th Indian Division, and, in 1918, adjutant, R.E., in Persia. He returned to civilian life in 1919, joining the Niger Co., Ltd., as assistant engineer in the mining department, where he remained for two years, and in 1927 and 1928 was engineer in charge of operations in Uganda for Ankole Tinfields, Ltd.

OBITUARY—*continued.*

Mr. Crighton went to Ecuador in 1928 in the position of chief assistant, Santiago Properties, Ltd., and from 1932 to 1933 was assayer and prospector at Taquah and Abosso Gold Mines, Ltd., West Africa. Since 1933 he had been operating his own tin mine at Kikagati, Uganda.

Mr. Crighton was elected a Student of the Institution in 1905 and was transferred to Associateship in 1907.

Gerald Leo Devitt died on October 30th, 1945, at the early age of 42. He was born in Bulawayo, and received his metallurgical training at Falcon mine laboratories, Umvuma, Southern Rhodesia, from January, 1921, to July, 1925. The following month he joined the staff of Northern Rhodesia Co., Ltd., on a geological expedition to Central Africa lasting three months, and in January, 1926, he took up an appointment as assayer at the Alaska and Copper Queen mines, Lomagundi District, Southern Rhodesia, where he remained for two years. From February, 1928, to January, 1930, he did underground contract work on the Mabekwe asbestos mines, Fort Victoria District, and was subsequently appointed assistant assayer at The Standard Bank of South Africa, Ltd., at Bulawayo, being transferred nearly five years later to Gwelo as chief assayer. Mr. Devitt went to the Dokuripe mine, Northern Territories, Gold Coast, in 1938, but returned to England in the following year. He was elected an Associate of the Institution in 1936.

Brigadier George Maitland Edwards, D.S.O., died at his home in London on March 27th, 1946, at the age of 63. He received his mining training in Cumberland, where he served an apprenticeship with Brayton Domain Collieries, Ltd., and studied mining subjects at Aspatria Science College, Carlisle. In 1900 he went to Queensland, where he worked on various gold mines until 1902, when he left for Siberia. After spending two years at the gold mines of Katchkar, he was appointed manager of the Baku Zabrat Petroleum Co., Ltd., in 1904, and when that work was interrupted by revolution in the following year he took over the management of the Caucasus Asphalt Co., Ltd. After a visit to England in 1906 he returned to Siberia and worked on copper and gold mines in the Urals until 1908, when he decided to specialize in petroleum. For the next three years he visited oilfields in Burma, Java, Sumatra and Borneo, and in June, 1911, returned to London and set up in practice as a consulting oil engineer. A year later, however, he took up the appointment of adviser to the Mines Department at Constantinople and some of his observations were published in a paper entitled ' Notes on Mines of the Ottoman Empire ' (*Transactions*, vol. xxiii, 1913–14). In 1913 he returned home, and later reported on oil lands in Oklahoma. He joined the Royal Artillery on the outbreak of war in 1914, and after serving in France was sent to Gallipoli, where he volunteered for mining operations under Colonel H. W. Laws. On the evacuation of Gallipoli he took his unit to France, and was given command of 254 Tunnelling Company, R.E., and in 1917 was sent to Russia as a staff officer, where he served until 1918. He was demobilized in 1919 with the rank of lieutenant-colonel, having been awarded the D.S.O., Croix de Guerre and other foreign decorations.

OBITUARY—*continued.*

Owing to ill-health, Brigadier Edwards was unable to resume professional work until 1924, when he took up the management of the Phoenicia Tin Mines, Spain, and in 1926 he restarted his consulting practice. During the following thirteen years he reported on deposits in Greece, Portugal and Morocco, and was consultant to several mining companies. When the War Office decided to re-form Tunnelling Companies of the Royal Engineers late in 1939, he was commissioned as O.C. of the first Company, which went to France in 1939. Early in 1940 it was used as the nucleus of a group of Companies, with Edwards, now lieutenant-colonel again, in command. Further expansion was intended, and Edwards and several officers selected as C.O.s of new groups were sent to England just before the German break-through, and were not permitted to return. In the Army re-organization after Dunkirk, Edwards was promoted to the rank of brigadier, and appointed Chief Engineer of an Army Corps, where he remained until his retirement under the age limit. He was thereupon appointed Deputy Director (Prospecting) of Opencast Coal Production, Ministry of Supply, and in that capacity organized a large team of mining engineers, nearly all members of the Institution. He resigned from this position, owing to ill health, towards the end of 1945.

Brigadier Edwards was elected an Associate of the Institution in 1908.

Lawrence Lyon Edwards was killed while on active service in Italy on May 21st, 1944. He began his mining experience at the Gaika gold mine, Southern Rhodesia, in 1930, and came to London in 1932, where he was a student at the Royal School of Mines, graduating in 1938 with the A.R.S.M. and degree of B.Sc. (Eng.). He thereupon took up a position as surveyor at the Geduld mine, Transvaal. Details of Mr. Edward's military service are unfortunately not known. He was elected to Studentship of the Institution in 1936.

Charles Edward Hutchings died in New York on May 27th, 1944, at the age of 49. He was born at Corrientes, Argentine, received his education at Castle Hill School, Ealing, and entered the Royal School of Mines in 1911. His studies were interrupted in August, 1914, when he left to join the Royal Marine Light Infantry, and he saw active service on the Gallipoli Peninsula. He was later commissioned to the 5th Dorsetshire Regiment, serving in Egypt and France, and on his discharge from the Army in May, 1918, he held the rank of captain. Mr. Hutchings resumed his studies at the Royal School of Mines in July of that year, and graduated two years later with an Associateship in Mining.

His first mining appointment was that of junior engineer to the Caracoles Tin Company of Bolivia, where he was employed in topographical surveying and prospecting. He remained with the same company for seven years, spending a year as section engineer in charge of a group of mines, then acting as assistant mine superintendent for nineteen months, and from 1923 to 1927 holding the position of mine superintendent. He was in charge of the general administration of the Caracoles group and outlying mines covering a length of forty miles. After seven years of continuous employment at an altitude of 16,000 ft. he returned to England on leave of absence. Mr. Hutchings left England again in 1928 to become manager to the

OBITUARY—*continued.*

Anglo-Greek Magnesite Co., Ltd., in Macedonia, Greece, and in 1931 he returned to South America to take up an appointment with New Goldfields of Venezuela, Ltd.

He was elected a Member of the Institution in 1928.

Arthur Cecil Perry is presumed to have died on or about February 13th, 1942, as a result of the sinking of the s.s. 'Giang Bee', no favourable news having been received since the cessation of hostilities and the subsequent liberation of prisoners of war and internees in the Far East. He was about 55 years of age.

He received his professional training at the Camborne School of Mines from 1903–1907, and then spent a year at the Botallack mine, Cornwall. Practically the whole of his subsequent career was spent in Malaya, beginning when he travelled there early in 1909 to take up a position at Bundi tin mine for Messrs, Guthrie & Co. After fourteen months he joined the staff of Rambutan, Ltd., and Pengkalen, Ltd., as assistant on the Gopeng tin mine under the direction of Messrs, Osborne & Chappel. He remained at Ipoh for some years, subsequently working at Batu Gajah from 1928 to 1933. He was then for two years employed by Minerals Exploration, Ltd., at Penang, and in 1936 went to Malim Nawar for Anglo-Oriental (Malaya), Ltd. Mr. Perry then spent a year at Bhuket, West Siam, on the staff of Tongkah Harbour Tin Dredging, Ltd., and some months with Tavoy Prospectors, Ltd., at Mergui, Lower Burma. He came to England in 1939, but rejoined Anglo-Oriental (Malaya), Ltd. at Kuala Lumpur in September, 1939. No news of his whereabouts could be obtained throughout the war, and notification of his presumed death was received from the Colonial Office in March, 1946.

Mr. Perry was elected a Student of the Institution in 1908 and was transferred to Associateship in 1912.

Major Cyril Harrison Russell, was killed on active service in Burma. He was a student at the Royal School of Mines from 1925 to 1928, graduating with the A.R.S.M. in Mining, and took up his first mining appointment in October, 1928, as assistant millman in the mill and cyanide plant at the Great Nurupi mine of the Chosen Corporation, Ltd., at Taiyudong, Korea. He was subsequently promoted to the position of assistant mill superintendent, and in 1932 became chief surveyor and engineer of the mine. From 1935 to 1936, Major Russell was geologist to South Banket Areas, Ltd., in the Gold Coast, leaving on his appointment as underground manager to the Consolidated Tin Mines of Burma, Ltd., Tavoy. He was transferred in 1937 to the position of mine superintendent at the Hermyingyi mine, and joined the Army after the outbreak of war in 1939.

Major Russell was elected to Studentship of the Institution in 1926 and transferred to Associateship in 1933.

Walter Edgar Segsworth died suddenly at his home in Toronto on July 20th, 1945, at the age of 65. He was born in Canada, and began his mining career in 1897 in the mill and assay office of Bannockburn gold mine, Ontario, two years later taking a position underground at the Poorman mine, Nelson, B.C. In 1900 he was appointed compressor man, and then

OBITUARY—*continued.*

worked underground and as millman at the Granit mine and later as assayer at the Venus mine, and in November of that year passed the examination as British Columbia Provincial Assayer. After working as assistant assayer to the Standard Pyretic Smelter, B.C., in November, 1901, he purchased an assay office in Greenwood, B.C., where for over two years he carried on a general practice as assayer and chemist. In October, 1904, he entered Michigan College of Mines, Houghton, U.S.A., and graduated in 1906, when for a few months he was employed as assistant engineer at the Allowez and Centennial copper mines, Calumet, Mich. In 1907 he returned to Canada as manager of the Nonsuch silver mines, Cobalt, Ont. When those operations were discontinued, he opened an office in Toronto for general engineering practice, and was engaged on various examinations and reports until 1912, when he took charge of the Seneca-Superior silver mine, in Cobalt. His subsequent activities included the full control in Canada of the rehabilitation of disabled soldiers after the 1914–18 war, and he was actively interested in the Canadian Institute for the Blind. He was responsible for the successful development and operation of the Moneta gold mines for five-and-a-half years, and as consulting engineer he played an important part in the development of Amulet, Central Patricia, Preston East Dome, Leitch and Canadian Industrial Minerals. At the time of his death he held executive positions on the boards of several Canadian mining companies.

Mr. Segsworth was elected to Membership of the Institution in 1916. He was also a member of the Canadian Institute of Mining and Metallurgy from 1907 and a Member of Council of that Institute for the period 1916–19.

The Council regret to report the death of **Lewis Percy Cazalet,** *Member,* on 25th April, 1946; **John Henry Cordner-James,** *Member,* on 20th April, 1946; and **Arthur S. Dwight,** D.S.O., *Honorary Member,* on 1st April, 1946. Obituary notices will appear in a later *Bulletin.*

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

ANNUAL REPORT OF THE IMPERIAL INSTITUTE, 1945, by Sir Harry Lindsay (Director). Cambridge: Heffer, 1946. 74 p.

ANNUAL REPORTS OF THE SOCIETY OF CHEMICAL INDUSTRY ON THE PROGRESS OF APPLIED CHEMISTRY. Vol. 29, 1944. Cambridge: Heffer, 1945. 573 p.

BRITISH STANDARDS:

No. 230, 1945: Visual type portable photometers.

No. 341, 1945: Valve fittings for compressed gas cylinders.

No. 354, 1945: Photometric integrators.

No. 667, 1945: Photoelectric type portable photometers.

No. 832, 1945: Bell transformers excluding transformers for use in mines.

No. 953, 1941: Strength tests for the protective toe-caps of boots used for industrial purposes (with amendments to 1945).

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No. 1225, 1945: Recommended methods for polarographic and spectrographic analysis of high purity zinc and zinc alloys for die casting.

Nos. 1265–68, 1945: Drawing Boards and Tee squares.

No. 1269, 1945: Titanium pigments (rutile type) for paints and titanium white types 6 and 7.

No. 1282, 1945: Classification of wood preservatives.

No. 1293, 1945: Screen analysis of coal (other than pulverized coal) with performance and efficiency tests on industrial plant.

London: British Standards Institution, 1945. 2s. each.

COAL MINES MECHANISATION. Report on wartime development and the general position at the present time, by G. M. Gullick. Ministry of Fuel and Power, September, 1945. 12 p.

FRENCH NORTH AFRICA. REPORT OF THE MINERALS DIVISION, BRITISH ECONOMIC MISSION, ALGIERS, by A. H. Steedman, D. Gill, A. Choubersky. (Together with APPENDICES, and coloured map showing the positions of the principal mines). Sept., 1945. (*Presented by the authors*).

GAS INDUSTRY. Report of the Committee of Enquiry, December, 1945. (With maps). Ministry of Fuel and Power. London: H.M.S.O., 1945. 2s. 57 p.

GOLD COAST GEOLOGICAL SURVEY: REPORTS ON THE GEOLOGY AND HYDROLOGY OF THE COASTAL AREA EAST OF THE AKWAPIN RANGE, by N. R. Junner and D. A. Bates (with 2 col. maps, scale 1 in.= 3.944 mls.). Memoir No. 7. Accra: Govt. Printing Dept., 1945. 23 p. 6s.

LIST OF ADDITIONS TO THE JOINT LIBRARY—*continued.*

HANDBOOK OF NON-FERROUS METALLURGY. 2nd Edn. VOL. 2.—RECOVERY OF METALS. D. M. Liddell, *ed.-in-chief.* New York: McGraw-Hill, 1945. 721 p. £2 2s.

JOURNAL OF THE IRON AND STEEL INSTITUTE. Vol. 150, No. 2., 1944. London: Iron and Steel Institute, 1945. 476+170 p.

KENYA GEOLOGICAL SURVEY: GEOLOGY OF THE NYERI AREA, by R. M. Shackleton. (With col. geological map, scale 1/125,000). Report No. 12. Nairobi: Govt. Printer, 1945. 26 p. 2s.

LAW OF TRADE UNIONS. By H. Samuels. London: Stevens and Sons, 1946. 93 p. (*Presented by the publishers to the Institution of Mining Engineers*).

LIST OF MINES IN GREAT BRITAIN AND THE ISLE OF MAN, 1944. Ministry of Fuel and Power, 1946. 295 p.

MINERAL RESOURCES, by A. M. Heron. Oxford: U.P., 1945. (Oxford pamphlets on Indian affairs). 31 p. 9d.

MONMOUTHSHIRE AND SOUTH WALES COAL OWNERS' ASSOCIATION. 18th Report of the Coal Dust Research Committee: The use of wetting agents for the suppression of airborne dust—Further report. January, 1946. 20 p. (*Presented by T. David Jones, University College, Cardiff, to the Institution of Mining Engineers*).

NOVA SCOTIA DEPARTMENT OF MINES: ANNUAL REPORTS ON MINES FOR 1943 AND 1944. Halifax: King's Printer, 1944 and 1945. 189 p. and 219 p.

PROVISION OF EMPLOYMENT IN SOUTH WALES FOR PERSONS SUSPENDED FROM THE MINING INDUSTRY ON ACCOUNT OF SILICOSIS AND PNEUMOCONIOSIS. Board of Trade. London: H.M.S.O., 1946. 1d. 6p.

PROVISIONAL STATEMENT OF NUMBER OF DEATHS BY ACCIDENTS AT MINES AND QUARRIES IN GREAT BRITAIN AND ISLE OF MAN DURING 1945. Ministry of Fuel and Power. London: H.M.S.O., 1946. 6 p. 1d.

QUIN'S METAL HANDBOOK AND STATISTICS, 1945. Metal Information Bureau, London. 1945. 367 p. 10s.

RECENT EXPLORATORY DEEP WELL DRILLING IN MACKENZIE RIVER VALLEY, NORTHWEST TERRITORIES. (Report, map, and figure). Geological Survey Paper, 45-29. Ottawa: Canada Mines and Geology Branch, 1945.

REPORT UPON LIGHTING PERFORMANCE OF SAFETY LAMPS IN COAL MINES IN LANCASHIRE AND NORTH WALES. Ministry of Fuel and Power. London: H.M.S.O., 1946. 13 p. 3d.

LIST OF ADDITIONS TO THE JOINT LIBRARY—*continued.*

Reports of the British and Combined Intelligence Objectives Sub-Committees. Various reports have been presented by the Department of Scientific and Industrial Research, and are listed under the Index of Recent Articles.

Second Progress Report of Technical Panel One on Utilisation of Low Grade Fuels. The Coal Industry Joint Fuel Efficiency Committee. Mining Association of Great Britain. July, 1945. 31 p.

Selected Bibliography on Coal in New Mexico, by R. L. Bates. Socorro, New Mexico: School of Mines and State Bureau of Mines and Mineral Resources, February, 1946. 3 p.

Smithsonian Institute: Annual Report of the Board of Regents for 1944. Washington, D.C.: Government Printing Office, 1945. 503 p.

South Australia Department of Mines: Mining Review for the Half-year ended 31st December, 1944—No. 81. Adelaide: Govt. Printer, 1945. 111 p.

Swaziland Geological Survey Department: Annual Report for 1944. 1945. 26 p. 3s.

Tests on Volumetric Glassware. Metrology Division, The National Physical Laboratory, January, 1946. 38 p.

The Scientific Civil Service—Reorganisation and Recruitment During the Reconstruction Period. London: H.M.S.O., September, 1945. 16 p. 3d.

Transactions, American Institute of Mining and Metallurgical Engineers, Volume 160: Petroleum Development and Technology, 1945. N.Y.: 1945.

Western Australia Department of Mines, Report of the Mineralogist, Analyst and Chemist for the year 1944. Perth: Govt. Printer, 1945. 32 p.

MAPS:

Coal and Iron—Great Britain—Sheets 1 and 2. Scale 1/625,000. Compiled by the Ministry of Town and Country Planning. Ordnance Survey, 1945. (*Presented by the Ministry of Town and Country Planning*).

Iron and Steel—Great Britain—Sheets 1 and 2. Scale 1/625,000. Compiled by the Ministry of Town and Country Planning. Ordnance Survey, 1945. (*Presented by the Ministry of Town and Country Planning*).

Pedley West of 5th Meridian, Alberta. Geological Survey Map 838A. Scale 1 in. = 1 ml. Ottawa: Canada Bureau of Geology and Topography, 1945.

LIST OF ADDITIONS TO THE JOINT LIBRARY—*continued.*

Shinimikas, Cumberland County, Nova Scotia. Geological Survey Map 842A. Scale 1 in. = 1 ml. Ottawa: Canada Bureau of Geology and Topography, 1945.

Taku River, Cassiar District, British Columbia. Preliminary map, Geological Survey. Scale 1 in. = 1 ml. Ottawa: Canada Bureau of Geology and Topography, 1945.

Tay River West of 5th Meridian, Alberta. Geological Survey Map 840A. 1 in. = 1 ml. Ottawa: Canada Bureau of Geology and Topography, 1945.

Victoria: Geological Map. Scale 16 mls. = 1 in. Geological Survey Branch, Department of Mines, Victoria, 1909. (*Presented by the Agent-General for Victoria*).

Victoria: Mineral Map. Scale 16 mls. = 1 in. Geological Survey Branch Department of Mines, Victoria, 1909. (*Presented by the Agent-General for Victoria*).

INDEX OF RECENT ARTICLES.

NOTE.—All Articles indexed are contained in the Library of the Institution.

INDEX OF RECENT ARTICLES—*continued.*

INDEX OF RECENT ARTICLES—*continued*.

COAL—*continued*.

DISTILLATION — HYDROGENATION — PETROL—GERMANY.—Gelsenberg Hydrogenation Plant.—Combined Intelligence Objectives Sub-Committee, Item No. 30, File No. XXX-105.

DISTILLATION — HYDROGENATION — PETROL—GERMANY.—Scholven Hydrogenation Plant.—Combined Intelligence Objectives Sub-Committee, Item No. 30, File No. XXX-102.

ECONOMICS — CANADA. — Research and the coal industry in Canada. W. A. Lang.—*Trans*. Canad. Inst. Min. Metall., Vol. 49, 1946, pp. 51-62. *Canad. Min. Metall. Bull.*, Montreal, No. 406, Feb., 1946. $1.00.)

ECONOMICS — LABOUR — EFFICIENCY. —Administration of collieries—setting the work standards (with task diagrams.) F. Colin Swallow.—*Colliery Engng.*, Lond., Vol. 23, March, 1946, pp. 56-9. 1s.

ECONOMICS — MANAGEMENT. — Administration of collieries: progress planning in relation to standards. F. Colin Swallow.—*Colliery Engng.*, Lond., Vol. 23, pp. 94-6. 1s.

EUROPE—N.W. EUROPEAN CAMPAIGN, 1944-45.—Coal in the N.W. European campaign. W. R. Gordon.—*Colliery Guard.*, Lond., Vol. 172, March 1-April 8, 1946, pp. 281-4, 316-7, 349-53, 421-3, 459-60. 10d. each.

FUEL ECONOMY — GERMANY. — Fuel technology and the Reichsvereinigung Kohle (organization of the German fuel economy, and allocation directorate.)—Combined Intelligence Objectives Sub-Committee, Item No. 28, File No. XXXI-28.

FUELS—GENERAL.—Fuel. R. A. Mott.—*Ann. Repts., Soc. Chem. Ind.*, Lond., Vol. 29, 1944, pp. 22-40.

FUELS — PEAT — INDUSTRIAL USES.—Peat combustion for industrial steam heating. A. J. Woods.—*Engineer*, Lond., Vol. 181, March 8, 1946, pp. 228-31. 1s.

FUELS — RESEARCH — GERMANY. — Coal and coke research, H. Koppers G.M.B.H., Essen.—Combined Intelligence Objectives Sub-Committee, Item No. 30, File No. XXXI-31.

FUELS — RESEARCH — GERMANY. — Fuel research activities of the A. G. der Kohlenwertsoff-Verbande Bochum.—Combined Intelligence Objectives Sub-Committee, Item No. 30, File No. XXXI-29.

FUELS — RESEARCH — GERMANY. — Fuel research and technology Bergbau-Verein Essen-Heisingen.—Combined Intelligence Objectives Sub-Committee, Item No. 30, File No. XXXI-25.

FUELS — SYNTHETIC. — Wartime research on synthetic fuels, Kaiser Wilhelm Institut für Kohlenforschung.—Combined Intelligence Objectives Sub-Committee, Item No. 30, File No. XXV-27.

COAL—*continued*.

GAS MANUFACTURE — THYSSEN — GALOCSY PROCESS.—Thyssen'sche Gas und Wasser-werke G.M.B.H. Duisberg-Hamborn Krupp Treipstoffwerk Wanne-Eickel (report on the Thyssen-Galocsy slagging gas producer process.)—Combined Intelligence Objectives Sub-Committee, Item No. 30, File No. XXIX-51.

GAS MANUFACTURE — WATER GAS — WINTERSHALL-SCHMALFELDT PROCESS.—The Wintershall-Smalfeldt water-gas process.—*Colliery Engng.*, Lond., Vol. 23, March, 1946, pp. 69-70. 1s.

GREAT BRITAIN.—Britain's coal mining industry—its rise and decline.—*S. Afr. Min. Engng. J.*, J'burg, Vol. 57, March 9, 1946, pp. 29-30. 6d.

HAULAGE — LOADING. — Machinery for winning and loading of bituminous coal underground.—Combined Intelligence Objectives Sub-Committee, Item No. 30, File No. XXX-24.

HAULAGE — LOADING — HUWOOD LOADERS.—Ashington colliery: working short faces on a retreating panel by means of Huwood loaders.—*Colliery Guard.*, Lond., Vol. 172, March 29, 1946, pp. 417-20. 10d.

HAULAGE — LOADING — SCRAPER LOADER.—Operation of pneumatic packing machine and scraper loader with planer.—Combined Intelligence Objectives Sub-Committee, Item No. 31, File No. XXX-30.

HOISTING — BELT HOISTS.—Belt hoists handle coal smoothly and economically.—*Coal Age*, N.Y., Vol. 51, March, 1946, pp. 93-8. 35 cents.

INDIA.—Indian mining practice. F. Darley.—*Colliery Guard.*, Lond., Vol. 172, April 5, 1946, pp. 479-84. 1s.

KENTUCKY — LEATHERWOOD. — Leatherwood mine being devloped into Kentucky's largest.—*Coal Age*, N.Y., Vol. 51 Feb., 1946, pp. 113-6. 35 cents.

LIGNITE — GERMANY. — I.G. Farbenindustrie A. G. für Stickstoff-Duenger Knapsack (with brief description of the brown coal deposit.)—Combined Intelligence Objectives Sub-Committee, Item No. 22, File No. XXVI-54.

MINE SUPPORT—Rock pressure and roof support. Z. S. Beyl.—*Colliery Engng.*, Vol. 23, April, 1946, pp. 84-8. 1s.

MINING METHODS—GREAT BRITAIN. —Modern trend in mining (coal). E. P. Dodsworth.—*Min. Elect. Mech. Engr.*, Manchester, Vol. 26, Feb., 1946, pp. 279-84. 2s. 6d.

MINING METHODS — MECHANICAL — U.S.A.—La mécanisation dans les houillères américaines. Louis Dupret.—*Annales des Mines et des Carburants.—Memoires*, Paris, No. 5, Sept.-Oct., 1945, pp. 589-621.

INDEX OF RECENT ARTICLES—*continued.*

COAL—*continued.*

MINING METHODS — OPENCAST. — Stripping efficiency raised by new methods and equipment.—*Coal Age*, N.Y., Vol. 51, March, 1946, pp. 71–5. 35 cents.

MINING METHODS—PITCHING SEAMS. —Mining method for a five foot seam pitch 25° to 50° mine Fritz, Hoesch A. G., Essen.—Combined Intelligence Objectives Sub-Committee, Item No. 30, File No. XXX-29.

MINING METHODS—PITCHING SEAMS. —Mining method for a five foot seam pitch 60°.—Combined Intelligence Objectives Sub-Committee, Item No. 30, File No. XXX-27.

NOVA SCOTIA—HALIFAX COUNTY.—Joggins-River Herbert coal district. M. G. Goudge.—*Nova Scotia Dept. Mines, Ann. Rept.*, 1944, pp. 152–81.

NOVA SCOTIA—INVERNESS COUNTY. —Mabou coal area. G. Vibert Douglas.—*Nova Scotia Dept. Mines, Ann. Rept.*, 1944, pp. 124–31.

NOVA SCOTIA—INVERNESS COUNTY. —The St. Rose-Chimney Corner coalfield, Inverness County, Nova Scotia. W. A. Bell. —*Nova Scotia Dept. Mines, Ann. Rept.*, 1943, pp. 106–22.

NORTHUMBERLAND—ASHINGTON COLLIERY.—Ashington colliery: working short faces on a retreating panel by means of Huwood loaders.—*Colliery Guard.*, Lond., Vol. 172, March 29, 1946, pp. 417–20. 10d.

OPENCAST — INDIANA. — 25 cu. yd. drag-line heads modern units at New Maumee Pit. —*Coal Age*, N.Y., Vol. 51, Feb., 1946, pp. 101–5. 35 cents.

OREGON — COOS — SOUTHPORT MINE. Oregon coal mine steps up production (history and development). Ralph S. Mason.—*Min. Congr. J.*, Wash., D.C., Vol. 32, Jan., 1946, pp. 32–5. 20 cents.

PREPARATION — MAGNETIC SEPARATORS.—Magnetic separators solve tramp-iron problem in preparation. E. H. Neyman. —*Coal Age*, N.Y., Vol. 51, March, 1946, pp. 90–2. 35 cents.

PREPARATION—WASHING—GERMANY. —The preparation of ultra-clean coal at the Königin Elizabeth colliery Essen-Frillendorf (Mannesmannrohnen Werke A.G.)—Combined Intelligence Objectives Sub-Committee, Item No. 30, File No. XXIII-6.

QUEENSLAND — DAWSON VALLEY — BARALABA COALFIELD.—Baralaba coalfield. J. H. Reid.—*Qd. Govt. Min. J.*, Brisbane, Vol. 46, Dec. 20, 1945, pp. 354–63. 6d.

SOUTH AUSTRALIA — LEIGH CREEK COALFIELD.—The Leigh Creek coalfield.—*S. Aust. Dept. Mines, Min. Rev.*, No. 81, 1944, pp. 27–80.

COAL—*continued.*

SYSTEMS — BORD AND PILLAR.—Bord and pillar mining (a historical survey of the development of this system of coal mining). Pts. III and IV. A. Roberts.—*Colliery Engng.*, Lond., Vol. 23, 1946; March, pp. 73–5; April, pp. 100–102. 1s. each.

TENNESSEE — CHATTANOOGA. — Rock Cove mine—built for long life and low mining cost. J. H. Edwards.—*Coal Age*, N.Y., Vol. 51, March, 1946, pp. 81–9. 35 cents.

U.S.A.—L'industrie houillère américaine. Louis Dupret.—*Annales des Mines et des Carburants—Memoires*, Paris, No. 5, Sept.–Oct., 1945, pp. 536–568.

WASHING—MIDDLINGS.—The disintegration of middlings. K. F. Tromp.—*J. Inst. Fuel*, Lond., Vol. 19, Feb., 1946, pp. 115–118.

WASHING—MIDDLINGS.—The economic limits of separation of middlings in dense medium washers. K. F. Tromp.—*J. Inst. Fuel*, Lond., Vol. 19, Feb., 1946, pp. 112–115.

WEST VIRGINIA—BOONE COUNTY—ROBIN HOOD MINE.—Low-cost mining over many years goal at Robin Hood mine. J. H. Edwards.—*Coal Age*, N.Y. Vol. 51, Jan., 1946, pp. 68–74. 35 cents.

WEST VIRGINIA—NICHOLAS COUNTY —NEW SEWELL No. 1 MINE.—High efficiency marks operation at New Sewell No. 1 Mine. J. H. Edwards.—*Coal Age*, N.Y., Vol. 51, Feb., 1946, pp. 106–9. 35 cents.

COPPER.

METALLURGY — REFINING — ELECTROLYTIC. — Electrical developments in modern copper refineries. C. C. Downie.—*Min. J.*, Lond., Vol. 226, March 2, 1946, pp. 168–70. 8d.

NORTHERN RHODESIA.—Copper mining in the war years (review of Rhokana activities.) —*S. Afr. Min. Engng. J.*, J'burg, Vol. 56, 1946; Feb. 16, pp. 603–7; Feb. 23, pp. 627–32. 6d. each.

NOVA SCOTIA—CUMBERLAND COUNTY.—Copper deposits, Wentworth District. M. F. Bancroft.—*Nova Scotia Dept. Mines, Ann. Rept.*, 1943, pp. 94–105.

ECONOMICS.

ACCOUNTS — STOCK RECORDS.—How Hudson Bay runs its mine warehouse. Parts III and IV. P. G. Bishop.—*Engng. Min. J.*, N.Y., Vol. 147, 1946; Jan., pp. 79–85; March, pp. 85–8. 50 cents each.

ANHYDRITE — INDUSTRIAL USES. — Industrial uses of anhydrite. A. E. Flynn.—*Nova Scotia Dept. Mines, Ann. Rept.*, 1943, pp. 147–51.

INDEX OF RECENT ARTICLES—*continued.*

ECONOMICS—*continued.*

COSTS — SOUTH AFRICA. — The cost structure in industry—how and why it should be lowered. Dr. Bernard Price.—*S. Afr. Min. Engng. J.*, J'burg, Vol. 57, March 2, 1946, pp. 3-5. 6d.

EDUCATION — GERMANY. — Activities of the Westfalische Berggewerkschaftskasse during wartime.—Combined Intelligence Objectives Sub-Committee, Item No. 30, File No. XXX-41.

HEALTH OF MINERS—COLOUR BLINDNESS.—Colour blindness in mining.—*Min. J.*, Lond., Vol. 226, April 13, 1946, pp. 288. 8d.

HEALTH OF MINERS—GERMANY.—Kaiser-Wilhelm-Institut für Arbeitsphysiologie (Physiology of exercise and application to industry, particularly mining.)—British Intelligence Objectives Sub-Committee, Final Report No. 83, Item No. 24.

HEALTH OF MINERS—NOISE.—Noise injuries to miners. W. Schweischeimer.—*Min. Mag.*, Lond., Vol. 74, March, 1946, pp. 156-7. 1s.

HEALTH OF MINERS — SILICOSIS — PREVENTION.—The 'Know-how' in fighting silicosis with aluminum. A. W. Jacob.—*Engng. Min. J.*, N.Y., Vol. 147, March, 1946, pp. 70-5. 50 cents.

TAXATION — S. AFRICA.—The State and gold mining—the effects of indirect taxation (Part V).—*S. Afr. Min. Eng. J.*, J'burg, Vol. 56, Jan. 26, 1946, pp. 533-5. 6d.

TECHNICAL SOCIETIES.—The work of the B.C. Industrial and Scientific Research Council as regards mining and metallurgy. S. E. Maddigan.—*W. Miner*, Vancouver, B.C., Vol. 19, Feb., 1946, pp. 43-5. 25 cents.

PRODUCTION — BRITISH COLUMBIA. —Review of mining in British Columbia in 1945. Fred. H. Stephens.—*Canad. Min. J.*, Gardenvale, P.Q., Vol. 67, Feb., 1946, pp. 84-90. 25 cents.

PRODUCTION — CANADA — GENERAL. —Canada's mineral industry in 1945. W. H. Losee.—*Canad. Min. J.*, Gardenvale, P.Q., Vol. 67, Feb., 1946, pp. 76-83. 25 cents.

PRODUCTION—LEAD.—The world's lead production—keeping up with heavier consumption.—*S. Afr. Min. Engng. J.*, J'burg, Vol. 57, March 9, 1946, pp. 35-6. 6d.

PRODUCTION — MANITOBA. — Mining in Manitoba in 1945; widespread prospecting and exploratory activity. J. P. de Wet.—*Canad. Min. J.*, Gardenvale, P.Q., Vol. 67, Feb., 1946, pp. 97-100. 25 cents.

PRODUCTION — MINERALS — CALIFORNIA.—California mineral production for 1944. Henry H. Symons.—*Calif. Min. J.*, San Francisco, Vol. 41, No. 3, July, 1945, pp. 192-5.

ECONOMICS—*continued.*

PRODUCTION — NOVA SCOTIA.—The mineral industry of Nova Scotia: diamond drilling and geological investigations. D. L. Currie.—*Canad. Min. J.*, Gardenvale, P.Q., Vol. 67, Feb., 1946, pp. 113-14. 25 cents.

PRODUCTION — ONTARIO. — Ontario Annual Review: increased production follows return of Army personnel. J. C. Browning. —*Canad. Min. J.*, Gardenvale, P.Q., Vol. 67, Feb., 1946, pp. 101-6. 25 cents.

PRODUCTION — QUEBEC. — Quebec Annual Review: activity on new prospects features the current situation. J. C. Browning.—*Canad. Min. J.*, Gardenvale, P.Q., Vol. 67, Feb., 1946, pp. 107-11. 25 cents.

GEOLOGY AND ORE DEPOSITS.

BELGIUM.—Minor elements in sphalerites from Belgium. Pierre Errard.—*Econ. Geol.*, Lancaster, Pa., Vol. 40, Dec., 1945, pp. 568-74. $1.00.

CALIFORNIA — RIVERSIDE COUNTY.—Mineral resources of Riverside County. W. B. Tucker and R. J. Sampson.—*Calif. J. Min.*, San Francisco, Vol. 41, No. 3, July, 1945, pp. 121-82.

DOLOMITE — NOVA SCOTIA.—Dolomite deposit, upper Musquodoboit. J. P. Messervey.—*Nova Scotia Dept. Mines Ann. Rept.*, 1943, pp. 62-70.

EIRE.—Minerals in Eire.—*Chem. Tr. J.*, Lond., Vol. 118, Feb. 15, 1946, pp. 207-8. 8d.

GOLD COAST.—Reports on the geology and hydrology of the coastal area east of the Akwapim Range. N. R. Junner and D. A. Bates.—*Geol. Surv. Gold Coast Colony*, Memoir No. 7, 1945. 23 pp. 6s.

HYDROLOGY.—Hydrology in relation to economic geology. Oscar E. Meinzer.—*Econ. Geol.*, Lancaster, Pa., Vol. 41, Jan.-Feb., 1946, pp. 1-12. $1.00.

KENYA—MERU DISTRICT.—A suite of hypersthene-bearing plutonic rocks in the Meru District, Kenya. William Pulfrey.—*Geol. Mag.*, Lond., Vol. 83, March-April, 1946, pp. 67-88. 2s. 6d.

KENYA — NYERI AREA. — Geology of the Nyeri area (with coloured geological map). R. M. Shackleton.—Geol. Surv. Kenya, Rept. No. 12, 1945. 26 pp. 2s.

NIGERIA.—Nigeria, geology and mineral resources. F. Dixey.—*Imp. Inst. Bull.*, Vol. 43, Oct.-Dec., 1945, pp. 300-16. 2s. 6d.

ONTARIO—COCHRANE AND THUNDER BAY.—Geology along the Trans-Canada highway between Hearst and Longlac. E. L. Evans.—*Ont. Dept. Mines, 51st Ann. Rept.*, Vol. 51, Pt. 9, 1942.

INDEX OF RECENT ARTICLES—*continued*.

GEOLOGY AND ORE DEPOSITS—*continued*.

ONTARIO—THUNDER BAY—LITTLE LONG LAC.—The hardrock 'porphyry' of Little Long Lac. J. A. Reid.—*Econ. Geol.*, Lancaster, Pa., Vol. 40, Dec., 1945, pp. 509-16. $1.00.

ORE DEPOSITION.—The control of impounding structures on ore deposition. Robert A. Mackay.—*Econ. Geol.*, Lancaster, Pa., Vol. 41, Jan.-Feb., 1946, pp. 13-46. $1.00.

PENNSYLVANIA.—Notes on the derivatives of the triassic diabases of Pennsylvania. W. Harold Tomlinson.—*Econ. Geol.*, Lancaster, Pa., Vol. 40, Dec., 1945, pp. 526-36. $1.00.

PETROGRAPHY — MICROSCOPIC EXAMINATION.—Petrographic analysis by fragment counting. Felix Chayes.—*Econ. Geol.*, Lancaster, Pa., Vol. 40, Dec., 1945, pp. 517-25. $1.00.

QUEBEC — ABITIBI REGION. — Major faults, Abitibi region, Quebec. G. W. H. Norman.—*Canad. Min. Metall. Bull.*, Montreal, No. 406, Feb., 1946, pp. 129-44. $1.00.

VENEZUELA—LLANOS. — Venezuela, The Llanos. John C. Davey.—*Min. Mag.*, Lond., Vol. 74, March, 1946, pp. 141-55. 1s.

WATER SUPPLY.—Natural waters: a survey of the characteristics of waters from various sources for drinking and for boiler feeding. Arthur Bray.—*Colliery Engng.*, Lond., Vol. 23, April, 1946, pp. 91-3. 1s.

GOLD.

BRITISH GUIANA—CUYUNI RIVER.—Notes on the Aurora goldfield, Cuyuni River, British Guiana. S. Bracewell.—*Min. Mag.*, Lond., Vol. 74, Feb., 1946, pp. 82-87. 1s.

ECONOMICS—CURRENCY.—The monetary future of gold. J. N. Reedman.—*Rhod. Min. J.*, J'burg, Vol. 18, Jan., 1946, pp. 13-17. 6d. *Min. Industr. Mag. S. Afr.*, J'burg, Vol. 34, Feb., 1946, pp. 87-91. 1s.

ECONOMICS — TAXATION — S. AFRICA. —Gold mining taxation—the case for the gold mining industry.—*Min. Industr. Mag. S. Afr.*, J'burg, Vol. 34, Jan., 1946, pp. 17-23. 1s.

METALLURGY — FINENESS.—The fineness of gold, with special reference to the Morobe goldfield, New Guinea. N. H. Fisher. —*Econ. Geol.*, Lancaster, Pa., Vol. 40, Dec., 1945, pp. 537-63. $1.00.

METALLURGY — WESTERN AUSTRALIA.—Investigation of treatment of gold ores: cyaniding copper sands—zinc and copper precipitation compared—charcoal in tailings—gold in flotation tailings.—*Chem. Engr. Min. Rev.*, Melb., Vol. 38, Jan. 10, 1946, pp. 121-4. 1s.

TRANSVAAL — WITWATERSRAND. — The Rand's gold industry (a survey of 1945 operations.— *S. Afr. Min. Engng. J.*, J'burg, Vol. 56, Feb. 9, 1946, pp. 373-7. 6d.

GOLD—*continued*.

WESTERN AUSTRALIA — YILGARN. — Notes on finds by Western Australian prospectors—II—conclusion of review of important Western Australian gold finds—the Yilgarn goldfield. Charles M. Harris.—*Chem. Engng. Min. Rev.*, Melb., Vol. 38, Dec. 10, 1945, pp. 91-94. 1s.

IRON AND STEEL.

ALLOYS — IRON — GERMANY. — Elektrowerk-Weisweiler (a plant operating on ferro-alloys of silicon, manganese, chrome, molybdenum and tungsten.)—Combined Intelligence Objectives Sub-Committee, Item No. 21, File No. XI-13.

IRON POWDER — GERMANY. — I.G. Farbenindustrie—Oppau Works Ludwigshafen (manufacture of carbonyl nickel and carbonyl iron powder.)—Combined Intelligence Objectives Sub-Committee, Item No. 21, File No. XXIV-12.

NOVA SCOTIA—BRIDGEVILLE.—Iron ore occurrences, Bridgeville area. J. P. Messervey.—*Nova Scotia Dept. Mines Ann. Rept.*, 1943, pp. 71-81.

ONTARIO — MICHIPICOTEN — HELEN MINE.—Mining and geology at the Helen mine. S. J. Kidder and G. C. McCartney.—(A.I.M.E. Tech. Pub. 1971) *Min. Tech.*, N.Y., Vol. 10, March, 1946, 24 pp. $1.35.

PORTUGAL.—Porque importamos ferro? Siderurgia Nacional (Review of iron mining in Portugal.) Manuel Aboim Ascensao de Sande Lemos and Abilino Vicente.—*Revista da Ordem dos Engenheiros*, Lisbon, No. 25, Jan., 1946, pp. 2-19.

PROPERTIES — MAGNETIC. — The magnetization of polycrystalline iron and iron-silicon alloys. G. C. Richer.—*J. Iron Steel Inst.*, Lond., Vol. 150, 1944, pp. 93-142.

SMELTING — BLAST FURNACES — GENERAL.— Considerations on blast-furnace practice. T. P. Colclough.—*J. Iron Steel Inst.*, Lond., Vol. 150, 1944, pp. 359-74.

SMELTING—BLAST FURNACES—GENERAL.—Discussion on blast-furnace operation and problems.—*J. Iron Steel Inst.*, Lond., Vol. 150, 1944, pp. 375-452.

SMELTING—BLAST FURNACES—SLAGS. —The constitution of some basic open-hearth slags. Brian Mason.—*J. Iron Steel Inst.*, Lond., Vol. 150, 1944, pp. 69-80.

SMELTING — DIRECT PROCESS — KRUPP-RENN PROCESS.—The Krupp-Renn process: continuous reduction of iron ore.—*Iron Coal Tr. Rev.*, Lond., Vol. 152, March 15, 1946, pp. 485-91. 9d.

INDEX OF RECENT ARTICLES—*continued.*

INDEX OF RECENT ARTICLES—*continued.*

INDEX OF RECENT ARTICLES—*continued.*

MINING (General)—*continued.*

BLASTING—DIAMOND DRILLS.—Blast-hole diamond drilling at the Sullivan mine. C. D. M. Chisholm.—*Trans.* Canad. Inst. Min. Metall., Vol. 49, 1946, pp. 63–78. (*Canad. Min. Metall. Bull.*, Montreal, No. 406, Feb., 1946. $1.00.)

BLASTING — SASKATCHEWAN — FLIN FLON.—Secondary blasting and car loading at Flin Flon. H. A. Kramer and G. M. Proudfoot.—*Trans.* Canad. Inst. Min. Metall., Vol. 49, 1946, pp. 22–31. (*Canad. Inst. Min. Metall. Bull.*, Montreal, No. 405, Jan., 1946. $1.00.)

BORING — CORE DRILLS — DIAMOND. —New double tube drill barrel—success of South African invention.—*S. Afr. Min. Engng. J.*, J'burg, Vol. 57, March 2, 1946, pp. 17–18. 6d.

BORING—DIAMOND DRILLS.—Diamond drilling as applied to ore production.—*Engng. Min. J.*, N.Y., Vol. 147, 1946; Jan., pp. 58–64; March, pp. 88–93. 50 cents each.

DEVELOPMENT — PIPE LAYING. —The Greenlee hydraulic pipe pusher—a description of a machine with wide general uses (for pushing pipes up to 4 in. dia. below ground level). S. V. Griffith.—*Min. Mag.*, Lond., Vol. 74, Feb., 1946, pp. 87–91. 1s.

DEVELOPMENT — PLANNING. — The time factor and the mining engineer. (Use of progress graphs in development schemes.) A. Nelson.—*Mine and Quarry Engng.*, Lond., Vol. 11, March, 1946, pp. 59–62. 1s.

DEVELOPMENT — WATER SUPPLY — TREATMENT.—The design, construction, and operation of a water-treatment plant at Mufulira mine, Northern Rhodesia. Kenneth Paterson.— *J. Instn. Civil Engrs.*, Lond., No. 4, February, 1946, pp. 281–300. 2s. 6d.

DRAINAGE — PUMPS — EFFICIENCY.— The economic pumping of underground mine waters. F. C. Johnson.—*J. S. Afr. Instn. Engrs.*, J'burg, Vol. 44, Dec., 1945–Jan., 1946, pp. 66–79. 3s. 6d.

DRAINAGE — PUMPS — HIGH LIFT.— Testing of pumps and pump columns. R. G. Sutton.—*J. S. Afr. Instn. Engrs.*, J'burg, Vol. 44, Dec., 1945–Jan., 1946, pp. 80–6. 3s. 6d.

DRAINAGE — PUMPS — SLUDGE. — Sludge pumping problems. R. H. Thomas.— *J. S. Afr. Instn. Engrs.*, J'burg, Vol. 44, Dec., 1945–Jan., 1946, pp. 86–94. 3s. 6d.

DREDGING — TASMANIA. — New dredge in operation at Dorset Flats, Tasmania— proving of alluvial area—description of dredge —details of operating results.—*Chem. Engng. Min. Rev.*, Melb., Vol. 38, Dec. 10, 1945, pp. 77–82. 1s.

EQUIPMENT — STOPE-FILLING MACHINES.—The Brieden pneumatic packing machine Karl Brieden & Co., Bochum.— Combined Intelligence Objectives Sub-Committee, Item No. 31, File No. XXX–25.

MINING (General)—*continued.*

FRENCH EQUATORIAL AFRICA.— Mining position in French Equatorial Africa. —*Rhod. Min. J.*, J'burg, Vol. 17, Dec., 1945, pp. 259. 6d.

GEOPHYSICAL PROSPECTING — BIBLIOGRAPHY. — Geophysical abstracts 121, April–June, 1945. W. Ayvazaglou and V. Skitsky.—U.S. Bureau of Mines *Information Circular* 7338, Nov., 1945. 48 pp.

GEOPHYSICAL PROSPECTING — COLORADO. — Geophysical surveys at the Malachite mine, Jefferson County, Colorado. C. A. Heiland, R. M. Tripp, and Dart Wantland.—*A.I.M.E. Tech. Pub.* 1947, 1945.

GEOPHYSICAL PROSPECTING—RADIOACTIVE METHODS.—Some practical aspects of radioactivity well logging. Warren J. Jackson and John L. P. Campbell.—*A.I.M.E. Tech. Pub.* No. 1943, 1945.

GEOPHYSICAL PROSPECTING—WASHINGTON.—A geological and geophysical study of the Chelan nickel deposit, near Winesap, Washington. Ernest N. Patty and Sherwin F. Kelly.—*A.I.M.E. Tech. Pub.* No. 1953.

HAULAGE — LOCOMOTIVES — DIESEL. —The Diesel locomotive underground (an examination of the potentialities and difficulties with special reference to British mining and locomotive practice). Brian Reed.—*Colliery Engng.*, Lond., Vol. 23, 1946; March, pp. 62–4; April, pp. 97–9. 1s. each.

HAULAGE — LOCOMOTIVES — DIESEL. —Design of mine-diesel locomotive.—Combined Intelligence Objectives Sub-Committee, Item No. 31, File No. XXX–43.

HOISTING — CONTROL. — The control of mine winders (Parts II and III). Rolt Hammond.—*Mine and Quarry Engng.*, Lond., Vol. 11, 1946; March, pp. 63–66; April, pp. 89–96. 1s. each.

HOISTING — GERMANY — KOEPE SYSTEM.—Investigation of Koepe winding practice in Ruhr district of Germany and Limburg district of Holland (illus.) — British Intelligence Objectives Sub-Committee Final Report No. 136, Item 31.

ILLUMINATION — SURFACE. — Colliery surface lighting (improved methods of illumination). C. S. Chubb.—*S. Afr. Min. Engng. J.*, J'burg, Vol. 56, Feb. 16, 1946, pp. 609–10. 6d.

NORWAY. — Bergverksdriften (Norwegian mining industry, 1930–1940). H. H. Smith.— Reprinted from *Tidsskrift Kjemi Bergvesen Og Metallurgi*, Nov., 1945, pp. 153–61. (*Presented by the author.*)

QUARRYING — MECHANIZATION. — Mechanization in quarries. J. P. Hanscombe. —*Quarry Mangrs. J.*, Lond., Vol. 29, April, 1946, pp. 494–520. 1s.

INDEX OF RECENT ARTICLES—*continued.*

MINING (General)—*continued.*

ROCK DRILLS — BITS — DETACHABLE BITS.—An investigation of detachable bits. Ralph E. Sinke.—*W. Miner*, Vancouver, B.C., Vol. 19, March, 1946, pp. 46–53. 25 cents.

ROCK DRILLS — DEVELOPMENT. — Rock-drill high lights: development of the mechanical drill.—*Compr. Air Mag.*, N.Y., Vol. 51, March, 1946, pp. 71–6. 35 cents.

TUNNELLING—U.S.A.—Machine mucking speeds Hudson coal tunnel-driving work. R. R. Richart.—*Coal Age*, N.Y., Vol. 51, Jan., 1946, pp. 79–83. 35 cents.

VENTILATION — AIR DUCTS.—Air duct design for industrial ventilating systems. C. C. Hermann.—*Chem. Met .Engng.*, N.Y., Vol. 53, March, 1946, pp. 118–21. 35 cents.

VENTILATION — COOLING. — Air cooling plant, No. 4 Shaft, City Deep, Ltd.—a review of layout and results achieved. H. W. Unwin, A. Malcolm, and R. E. Campbell Pitt. —*J. Chem. Met. Min. Soc. S. Afr.*, J'burg, Vol. 46, Sept.–Oct., 1945, pp. 69–119. 3s. 6d.

VENTILATION — DUST — MICROSCOPIC EXAMINATION. — Micrometer-graticule combination for particle determination. S. C. Blacktin.—*Chem. Ind.*, Lond., April 13, 1946, pp. 154–6. 1s. 9d.

VENTILATION — DUST — PARTICLE SIZE.—A comparison of methods of measuring microscopical particles. Harold Heywood.—*Instn. Min. Metall. Bull.*, Lond., No. 477, March, 1946, 14 pp. 1s. 6d.

VENTILATION — DUST — PARTICLE SIZE.—Meaning of particle size: methods of expressing particle size—need for specific definition. H. H. Dunkin.—*Chem. Engr. Min. Rev.*, Melb., Vol. 38, Jan. 10, 1946, pp. 129–31. 1s.

VENTILATION — ELECTRIC FAN.—Fan failure de-energizes underground power circuits. J. G. Pollock and R. R. Richart.—*Coal Age*, N.Y., Vol. 51, Jan., 1946, pp. 89–93. 35 cents.

VENTILATION — HUMIDITY AND TEMPERATURE.—Control of humidity in industry. M. R. Morton.—*Industr. Heating Engr.*, Lond., Vol. 8, April, 1946, pp. 50–4. 1s.

MINOR METALS.

CHROMIUM.—Le chrome dans le monde. N. Demassieux.—*Chimie et Industrie*, Paris, Vol. 55, Jan., 1946, pp. 58–64.

MANGANESE — MEXICO — LOWER CALIFORNIA.—Manganese from Mexico. Roger V. Pierce.—*Compr. Air Mag.*, N.Y., Vol. 51, Feb., 1946, pp. 33–8. 35 cents.

MANGANESE — TREATMENT — ELECTROLYTIC METHODS. — Electro-process developed to produce metallic manganese (with flow sheet). Glenn L. Allen, J. H. Jacobs and J. W. Hunter.—*Chem. Met. Engng.*, N.Y., Vol. 53, March, 1946, pp. 106–8. 35 cents.

MINOR METALS—*continued.*

MERCURY—U.S.A.—Mercury meanderings —Part I. Worthen Bradley.—*Min. Congr. J.*, Wash., D.C., Vol. 32, Jan., 1946, pp. 36–7. 20 cents.

NICKEL — GERMANY. — I. G. Farbenindustrie-Oppau Works, Ludwigshafen (manufacture of carbonyl nickel and carbonyl iron powder).—Combined Intelligence Objectives Sub-Committee, Item No. 21, File No. XXIV–12.

RADIOACTIVE — WITWATERSRAND.—Radio active substances (research on the Witwatersrand).—*S. Afr. Min. Engng. J.*, J'burg, Vol. 56, Feb. 2, 1946, pp. 565–8. 6d.

RARE METALS — INDIUM. — Indium.—*Min. J.*, Lond., Vol. 226, March 16, 1946, pp. 208–9. 8d.

TANTALITE — WESTERN AUSTRALIA. —Report on a visit to Wodgina district. H. P. Rowledge.—*W. Australia Dept. Mines Rept.*, 1945, pp. 20–21.

TANTALUM — WESTERN AUSTRALIA. —Mineralogical examination of tantalite concentrates from Strelley, Tabba Tabba and Pilgangoora. H. P. Rowledge.—*W. Australia Dept. Mines Rept.*, 1945, pp. 18–19.

TANTALUM.—Tantalite and tantalum—the future utilization of a little-known mineral. John H. Hohnen.—*Min. Mag.*, Lond., Vol. 74, Feb., 1946, pp. 72–82. 1s.

TITANIUM.—Preparation and properties of ductile titanium. R. S. Dean, J. R. Long, F. S. Wartman and E. L. Anderson.—(A.I.M.E. Tech. Pub. 1961) *Metals Tech.*, N.Y., Vol. 13, Feb., 1946, 17 pp. $1.25.

TUNGSTEN — SCHEELITE — FLUORESCENCE PROPERTIES.—A review of flourescence as applied to minerals, with special reference to scheelite. J. W. Vanderwilt.—(A.I.M.E. Tech. Pub. 1967) *Min. Tech.*, N.Y., Vol. 10, March, 1946, 14 pp. $1.35.

NON-METALLIC MINERALS.

ABRASIVES — GERMANY. — The German abrasive industry.—British Intelligence Objectives Sub-Committee, Final Report No. 113, Items No. 21, 31.

BARIUM—NOVIA SCOTIA—INVERNESS COUNTY.—Barytes and fluorspar deposits at Lake Ainslie. G. Vibert Douglas.—*Nova Scotia Dept. Mines Ann. Rept.*, 1944, pp. 132–51.

CEMENT—PORTLAND.—Improvements on Portland cement and concrete—past, present, and future. M. Spindel and R. T. Quinn.—*J. Soc. Engrs.*, Lond., Vol. 36, July–Dec., 1945, pp. 140–64. 3s. 6d.

CLAYS—KAOLIN.—China clay working (with diagrams). W. O. Meade-King.—*Trans., Cornish Inst. Min. Mech. Met. Engrs.*, Camborne, Vol. 1, Oct.–Dec., 1945, pp. 9–16.

INDEX OF RECENT ARTICLES—*continued.*

NON-METALLIC MINERALS—*continued.*

DIATOMITE — NOVA SCOTIA — COLCHESTER COUNTY.—Diatomite, Oxford and Tatamagouche areas. M. F. Bancroft.—*Nova Scotia Dept. Mines Ann. Rept.*, 1944, pp. 107–23.

FLUORSPAR — NOVA SCOTIA — INVERNESS COUNTY.—Barytes and fluorspar deposits at Lake Ainslie. G. Vibert Douglas. —*Nova Scotia Dept. Mines Ann. Rept.*, 1944, pp. 132–51.

LIMESTONE — NOVA SCOTIA — HANTS COUNTY.—Limestone deposit, Riverside Corner. J. P. Messervey.—*Nova Scotia Dept. Mines, Ann. Rept.*, 1943, pp. 82–7.

MARL — NOVA SCOTIA — ANTIGONISH COUNTY.—Marl deposit, Lanark. J. P. Messervey.—*Nova Scotia Dept. Mines Ann. Rept.*, 1943, pp. 88–98.

MICA—COLOUR CLASSIFICATION.—Color standard for ruby mica. Deane B. Judd.—*J. Res. U.S. Nat. Bur. Stand.*, Wash., D.C., Vol. 35, Oct., 1945, pp. 245–56. 30 cents.

MICA — PHYSICAL PROPERTIES. — Some physical properties of mica. Peter Hidnert and George Dixon.—*J. Res. U.S. Nat. Bur. Stand.*, Wash., D.C., Vol. 35, Oct., 1945, pp. 309–53. 30 cents.

MICA — SYNTHETIC — GERMANY. — Synthetic mica process, Ostheim, Germany.—Combined Intelligence Objectives Sub-Committee, Item No. 22, File No. XXII–11.

SILICA — QUARTZ — BRAZIL.—Deposits of quartz crystal in Esperito Santo and eastern Minas Gerais, Brazil. F. L. Knouse.—(A.I.M.E. Tech. Pub. 1962) *Min. Tech.*, N.Y., Vol. 10, March, 1946, 12 pp. $1.35.

SILICA — MOULDING SANDS — SYNTHETIC.—The effect of grain shape on the moulding properties of synthetic moulding sands. W. Davies.—*J. Iron Steel Inst.*, Lond., Vol. 150, 1944, pp. 19–47.

PETROLEUM.

ENGLAND.—The English oilfields. R. K. Dickie.—*Royal Inst. Chem. Gt. Britain & Ireland, J. & Proc.*, Feb., 1946, pp. 43–4. 2s.

GEOLOGICAL — GREAT BRITAIN.—The geological results of the search for oilfields in Great Britain. G. M. Lees and A. H. Taitt.—*Quart. J. Geol. Soc.*, Lond., Vol. 101, Feb., 1946, pp. 255–317. 7s. 6d.

NEW MEXICO.—Future oil possibilities of New Mexico. Robert L. Bates.—*State Bureau Mines & Mineral Resources*, New Mexico, Circular 12. Feb., 1946.

OIL SHALE—DISTILLATION—FRANCE. —French oil shale industry.—Combined Intelligence Objectives Sub-Committee, Item No. 30, File No. XXVI–78.

PETROLEUM—*continued.*

OIL SHALE — DISTILLATION — GERMANY.—Oil recovery from Wurttemberg shale.—Combined Intelligence Objectives Sub-Committee, Item No. 30, File No. XXX–18.

PETROLEUM—GENERAL.—Mineral oils. W. W. Goulston.—*Ann. Repts., Soc. Chem. Ind.*, Lond., Vol. 29, 1944, pp. 62–72.

U.S.A.—Our petroleum resources. Wallace E. Pratt.—*Smithsonian Instn.*, Ann. Rept., 1944, pp. 297–306.

WELL DRILLING — SHOOTING.—Some problems encountered during well shooting operations in the Nottinghamshire oilfields. J. F. Waters.—*J. Inst. Petrol.*, Lond., Vol. 32, March, 1946, pp. 119–26. 7s. 6d.

PLANT AND POWER.

COMPRESSED AIR—COAL MINES.—Compressed air and its application to coal mining. D. Y. Marshall.—*Min. Elec. Mech. Engr.*, Manchester, Vol. 26, March, 1946, pp. 333–41. 2s. 6d.

ELECTRICITY — PLANT — PERSONNEL. —The training of an electrical craftsman for mining. C. Brennan.—*Min. Elect. Mech. Engr.*, Manchester, Vol. 26, Feb., 1946, pp. 298–301. 2s. 6d.

ELECTRICITY — SWITCHGEAR. — Airbreak and airblast switchgear. J. H. C. Peters.—*Min. Elect. Mech. Engr.*, Manchester, Vol. 26, March, 1946, pp. 314–22. 2s. 6d.

PRECIOUS STONES.

DIAMONDS.—The world's diamond industry —a review of 1944 production. Sydney H. Ball.—*S. Afr. Min. Engng. J.*, Vol. 56, Pt. 2, Jan. 19, 1946, pp. 523–26. 6d.

EMERALDS — SOUTH AFRICA. — South African emeralds—mining operations to be expanded.—*S. Afr. Min. Engng. J.*, J'burg, Vol. 57, March 16, 1946, pp. 59–61. 6d.

SILVER.

METALLURGY.—The treatment of silver ores (with flow sheets of various mills). F. B. Michell.—*Mine & Quarry Engng.*, Lond., Vol. 11, April, 1946, pp. 81–6. 1s.

TIN.

CORNWALL—CARN BREA AREA.—A survey of the deeper tin zones in a part of the Carn Brea area, Cornwall. Brian Llewellyn.—*Instn. Min. Metall. Bull.*, Lond., No. 477, March, 1946, 19 pp. 1s. 6d.

CORNWALL — GEEVOR MILL.—Geevor Mill (with detailed flow sheet). A. D. Bennetts. —*Trans., Cornish Inst. Min. Mech. Met. Engrs.*, Camborne, Vol. 1, Oct.-Dec., 1945, pp. 17–27.

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 Sessional year 1945–46, and the Statement of Accounts for the calendar year ended December 31st, 1945. As in previous Reports, the statistics given refer to the calendar year.

Death of the President.

As previously announced in the *Bulletin*, Lt.-Colonel Edgar Pam died at his home at Virginia Water on December 20th, 1945, after a long illness. The Council wish to record their sense of the severe loss to the Institution, and their sympathy with Mrs. Pam in her bereavement. Many members of the Institution showed their respect for the memory of the late President by attending a Memorial Service held in London in January.

The Post-War Position.

The Council are glad to report that, thanks to the loyalty and co-operation of members, the position of the Institution after six years of war is reasonably good. At the end of 1945 the total membership was only 124 less than in 1938, notwithstanding the fact that during the war years the intake of students into the various Schools of Mines and Metallurgy was greatly reduced, resulting in 140 fewer elections to Studentship of the Institution than during the six years preceding the war. The roll of membership at the end of 1945 includes 9 members who had been reported prisoners-of-war or civilian internees and a further 45 members who were working in countries overrun by the enemy, from all of whom nothing has been heard recently. Their names are published in the May, 1946, *Bulletin*, in the hope that other members may have some information concerning them. Three Members, 27 Associates and 23 Students are known to have lost their lives, either on service with the Armed Forces, or by enemy action.

The exigencies of war have stressed the importance of the Institution to the profession and to the mining and metallurgical industries. The Institution has been consulted by many branches of Government, and took a leading part in establishing the Mining Taxation Committee and thus in securing a more reasonable application of Income Tax Law to mining companies. There is no doubt that in the next few years the Institution will be called upon to undertake new tasks, and the Council look forward with confidence to the whole-hearted support of members.

Roll of the Institution.

During 1945 there were 87 admissions into the Institution (including 5 reinstatements), and 41 transfers to Associateship or Membership. Statistics of membership at December 31st, 1945, with corresponding figures for the previous year, are as follows:

	1945.	1944.
Honorary Members	10	10
Members	628	633
Associates	1,095	1,110
Students	389	388
Totals	2,122	2,141

The names of 5 Members, 20 Associates and 12 Students have been removed from the Roll owing to non-payment of subscription, and the resignations of 10 Members, 7 Associates and 2 Students were received and accepted. The name of one Student was removed from the Roll under the provisions of Section II, Clause 3, of the By-Laws.

The Council record with deep regret the decease of 21 Members, 19 Associates and 9 Students. Their names are as follows:

Members (21): Vincent Brice Carew Baker, William John Barnett, Walter Henry Beasley, Edward Leslie Gilbert Clegg, Eric Roy Davis, George Stewart Scott Duns, Walter John Joseph Franks, Edmund Lionel Gay-Roberts, George Wynter Gray, Edmund William Janson, William Henry Johnson, Reginald Frank Krall, William Patrick Leal, John O'Malley Lyons (died on active service), John Taylor Marriner, Frank Oates, Leonard Camroux Oliver, Edgar Pam, Hans Pirow, Philip James Arthur Plummer, Alexander Robert Thomson.

Associates (19): Richard Ashley Atkinson (killed on flying operations), John Ewart Trounce Barbary, Nicholas Boonin (killed in action), Tom Caplen, Paul Charles Cuthbert Cayley (died on active service), Richard Arthur Ormerod Claudet, William Alfred Corkill (died in prisoner-of-war camp), John Langhorne Francis, Cyril Frazier, Ernest George Houghton, Frederick William Kendall, Donald Rennie Pengilly, Wolfram Hermann Albert Penseler (died in prisoner-of-war camp), Cyril Harrison Russell (killed on active service), William Robert Wilson Ronald Scott, Ronald John Coulson Telford (died on active service), Frederick Percy Tremble, Brian Frank Tyson (died in prisoner-of-war camp), Thomas Weir.

Students (9): John Paul Barkell Bennett (killed in action), William Robert Davis (died in prisoner-of-war camp), Lawrence Lyon Edwards (killed on active service), William Eric Shirres German (died of injuries received in action), Peter Donald Hopkins (died on active service), Augustus Huws Rocyn Jones (killed in action), Reginald Arthur Montague Lemmon (killed on flying operations), Philip John Nice (killed on active service), Carlos René Pullinger (died of wounds received in action).

The above list represents a serious loss to the Institution and to the mining and metallurgical professions. It includes the name of Mr. G. W. Gray, who was President in the Session 1933–34 and was subsequently Chairman of the Committee on Dust Sampling Investigations. The Council offer their sincere sympathy to the relatives of all those who have died.

The list of Members, Associates and Students admitted to the Institution during 1945 is as follows:

Members (8): Leigh Watson Bladon, Arthur Savile Davis, Antoine Julien Gonon, Bernard John Hastings, William Ewart Hosking, Edward Henry Jones, Albert Edward Phaup, Frederick Trostler.

Associates (36): Horace Percival Allwright, Robert Bernard Allwright, Albert Edward Andrews, Arnold Armstrong, Thomas Hamilton Blair, Gerald Leslie Brown Douglas, Claude Arnold Creelman, Yule Herbert Martin Crosby (reinstatement), Eric George Cutbush, Samuel Howard Davies, Tom Price Elwell, Kenneth Arthur Fern, Laurence Goudie, Reginald Ronald Eric Jacobson, Frederick Louis Jameson, Maurice Colin Kain, Jacobus de Villiers Lambrechts, Nigel Bertie Locke, James Richard Michell, Elvert Richard Mitchell (reinstatement), Ernest George Vincent Newman, John Gibson Nichols, Owen Arthur Mellish Nicolle, Albert Philip Notzing, Robert Murray Park, Ronald Alfred George Peck, Joseph Penn Rosslyn Polkinghorne, Peter Foreman Ransby, Henry Angwin Roberts, Norman Rudolf Schindler, Francis John Burchell Somerset, Richard Stackhouse Stead (reinstatement), John Wood Thorburn, Joseph James Timmins, Chester John Wilton, William Olive Wood.

Students (43): Hamish Johnston Alexander, Sadreddin Alpan, Royston Ernest Arch, Anthony Macaulay Babington, George William Bellman, Malcolm Anthony Bowley, Robert Laurence Carlton, Philip Alan Clive, Joseph Albert Cocking, Geoffrey Arthur William Dove, John Dyer, Michael Gawthrop, William Henry Goldsworthy,

Stanley Gray, Senih Gürel, Frederick Evan Roy Hamblin, David Jenner Hogg, Douglas Hosking, John Roger Hosking, R. Krishnaswamy, Edward Michael Paterson Lee, Michael Elliott Lewis, Louis William McNamara, John Scott Matheson, Arthur Theodore Max Mehliss, Derrick Morris, George Edwin Osborne, Tom Penhale, Petros Georgiou Petropoulos, Daniel Hague Phillips, Frederic Roger Phillips, Richard Athanasius Pryor, M. Nagesa Rao, Maurice Smith (reinstatement), Derek Anthony Temple, Peter Allen William Thuell, William Haydn Tinker, Mustafa Orhan Ural, Peter William Watson, Peter Christopher Mackain Weston (reinstatement), Leslie Wilton, Kenneth Paul Wright, Alfred Charles Patrick Wynne.

The following have been transferred :

To *Membership* (23) : Harry Arnall, Harry Archibald Cochran, William Sinclair Davidson, Michael Falcon, John Adams Fawdry, Oakley George Howard Gale, Michael Duhan Garretty, William Albert Hardy, Victor William George Hodgson, Cecil John Irving, Edgar Charles Knuckey, Thomas Hope Brendan Lawther, Clive Maxwell Norman, Reginald Paul Opie, Edmund James Pryor, Douglas John Rogers, William Sutherland, Walter Wood Tullis, Edgar Aruba Walker, William Morley Warren, Henry Carlyon Webb, Jesse West, Arthur Howell Williams.

To *Associateship* (18) : Angus George Forbes Alexander, George Peter Bennett, Arthur William Boustred, Richard John Trewhitt Caney, Crichton Colquhoun Cullen, Jack Lane Harvey, Jan Jacobi, Noel Spencer Kirby, Lotfollah Nahai, Oliver Douglass Paterson, John Douglas Phillips, Coonigal Ranganna Ramaswamy, Thomas Patrick Simpson, Wilfred Leslie Stapleton, Arthur Hugh Enfield Taylor, George Edward Thomson, John Thomson-Jacob, Eric William Warner.

Honours and Distinctions.

Reports of the award of honours and distinctions to the following members have been received during the year, and the Council wish to extend their congratulations to the recipients :

Order of the British Empire—

C.B.E. (Civil Division)—

Colin Raeburn, O.B.E. (*Member*).

O.B.E. (Military Division)—

Colonel D. T. Hudson, D.S.O. (*Associate*).

Lieutenant-Colonel J. S. Sheppard, *Royal Engineers* (*Associate*).

Lieutenant-Colonel John Weekley, *Perak Local Defence Force* (*Member*).

Lieutenant-Colonel G. A. Whitworth, *Home Guard* (*Member*).

M.B.E. (Military Division)—

Major G. J. Mortimer, *Royal Engineers* (*Student*).

Military Cross—

Captain J. P. B. Bennett, *Royal Engineers* (*Student*) (since killed in action).

Bar to Military Cross—

Captain J. L. HALLS, M.C., *South African Artillery* (*Student*).

Distinguished Flying Cross—

Flying Officer G. G. RUNNALLS, *Royal Air Force* (*Student*).

Squadron Leader R. C. PARGETER, *Royal Air Force Volunteer Reserve* (*Student*).

OFFICERS AND COUNCIL.

On the death of Lt.-Colonel Edgar Pam, Mr. G. F. Laycock, who had been Acting President during Colonel Pam's illness, was elected President for the remainder of the Session 1945–46. He was also elected President for the Session 1946–47.

Dr. J. G. Lawn has intimated his desire to resign from the office of Honorary Treasurer, and the Council wish to record their sincere appreciation of the great service he has rendered in this capacity since 1938. The Council are glad that his wide knowledge and experience will still be available, as he will continue to serve on the Council as a Past-President. The Institution has been fortunate to secure a worthy successor to the office of Honorary Treasurer in Mr. Robert Annan, who has consented to serve for the Session 1946–47.

Owing to the death of Mr. L. C. Stuckey, Mr. E. G. Lawford was elected a Vice-President for the Session 1945–46. Messrs. G. Keith Allen, H. R. Holmes, E. G. Lawford, E. A. Loring, Andrew Pearson, and S. E. Taylor have been elected or re-elected Vice-Presidents for the Session 1946–47.

The vacancy for a Member of Council caused by Mr. Lawford's election as Vice-President was filled by the election of Mr. T. Eastwood. Mr. A. Broughton Edge resigned his seat on the Council during the past Session, and Mr. A. J. G. Smout was elected in his place. Two Members of Council, Professor S. J. Truscott and Mr. G. W. M. Eaton Turner, expressed their wish not to be included in the ballot for the election of Council for 1946–47, and the Council record their thanks for past services.

MEETINGS.

Since the last Annual General Meeting, seven General Meetings have been held in the Apartments of the Geological Society, Burlington House, and the Institution renews its thanks to the Society for their continued hospitality. Towards the end of the Session, General Meetings were again held monthly, thus affording more opportunities for members to meet each other after, in many cases, long absences abroad. Attendances have reached pre-war proportions. During 1945, thirty Council and Committee Meetings were held.

PUBLICATIONS.

The *Bulletin* has been published every two months, and there is unfortunately still no prospect of resuming monthly publication, owing to the meagre ration of paper and shortage of labour at the printers. Vol. LIII (1943–44) of the *Transactions* has only recently been published, after being with the printers for many months. Vol. LIV has now been put in hand, and will be issued as soon as the difficulties of the printing trade will allow.

The Council thank all those who have written papers and joined in discussions, and they trust that prospective authors will not be deterred by the unavoidable delay in publication.

INSTITUTION AWARDS.

The Gold Medal of the Institution for 1945 has been awarded to Mr. Carl Raymond Davis, E.M., M.Inst.M.M., in recognition of his services to the Gold Mining Industry in South Africa and West Africa, to Mining in Northern Rhodesia, and to Mining Education as the Representative of the Institution for eight years on the Governing Body of the Imperial College of Science and Technology.

Dr. James Gunson Lawn, *C.B.E.*, A.R.S.M., Hon. D.Sc.(Eng.), M.Inst.M.M., *Past-President*, has been elected an Honorary Member of the Institution in recognition of his services to the Institution as Honorary Treasurer from 1938 to 1946.

'The Consolidated Gold Fields of South Africa, Limited' Gold Medal and Premium of Forty Guineas have been awarded to Mr. Gavin Hildick-Smith, B.Sc.(Min.), M.Inst.M.M., for his Paper on 'Shaft Pillars and Shaft Spaces' (*Transactions*, Vol. LIV).

EDUCATION.

The scheme for awarding National Certificates in Metallurgy is now in operation, and so far 10 technical schools and colleges have applied for the approval of courses of instruction. Dr. S. W. Smith, who was one of the Institution's representatives on the Joint Committee administering the scheme, has been appointed an Assessor, and his place on the Committee has been taken by Mr. E. G. Lawford; Mr. W. A. C. Newman continues to serve on the Committee.

The Joint Committee on Metallurgical Education, mentioned in the last Report, has met several times, and will shortly issue a *brochure* on 'Metallurgy as a Career'. Other projects include the formation of a film library on metallurgical subjects. Mr. Lawford has joined this Committee as an additional representative of the Institution, the others being Mr. E. D. McDermott and Mr. Newman. The Committee now comprises representatives of five metallurgical institutions: the Iron and Steel Institute, the Institution of Mining and Metallurgy, the Institute of British Foundrymen, the Institute of Metals, and the Institution of Metallurgists.

The Council have pleasure in reporting the generous offer of the Mond Nickel Company, through its Chairman, Dr. W. T. Griffiths, to make available during the next seven years a sum of £50,000 for the establishment of Post-Graduate Fellowships in Metallurgy, to be administered jointly by the five metallurgical institutions. The Council have gratefully accepted this offer on behalf of the Institution, and have expressed their willingness to co-operate with the other four institutions.

Library.

The Library was brought back to Salisbury House from Derbyshire in September, 1945, and Flying Officer W. G. Watts, R.A.F., was appointed Librarian. Mr. Watts has since been demobilized and assumed duties early in 1946. During the year 270 books were borrowed from the Library, and the total number of books and pamphlets added was 402.

Representation on Other Bodies.

Members of Council have continued to represent the Institution on other bodies. Additional representation during the year includes Dr. Wm. Cullen on the Advisory Committee on Metallurgy of King's College, Newcastle-on-Tyne, and Prof. J. A. S. Ritson on the Committee organizing the Ores and Minerals Team for collecting industrial intelligence from Germany (British Intelligence Objectives Sub-Committee—B.I.O.S.). Mr. Laycock has taken the place of the late Colonel Pam as the Institution's representative on the Executive of the Parliamentary and Scientific Committee.

Accounts.

The Statement of Receipts and Expenditure for the year ended December 31st, 1945, submitted herewith, shows a surplus for the year of £2,100 5s. 4d., of which £1,500 has been placed to the reserve for *post-war activities* and for contingencies.

Expenditure on publications is less by approximately £400, due mainly to the slow progress in the preparation of Vol. LIII of the *Transactions*. Two new items of expenditure will be noted, namely, National Certificates in Metallurgy and the Joint Committee on Metallurgical Education. The former item, of £162, will be increased to approximately £300 in 1945, and in addition the Council have agreed to make three annual donations of £100 each to a Prize Fund established in connection with the National Certificates.

General.

Housing.—It is regretted that no progress can be reported in the search for a fitting home for the Institution. As is well known, the war has left its mark on London, and the type of accommodation required is at present difficult to obtain.

Appointments (Information) Register.—Over 50 vacancies were brought to the notice of the Institution, and 47 members whose names were on the Register obtained appointments. A number of vacancies still exist for recent graduates.

Silicosis.—Arrangements are being discussed for a Joint Meeting with the Institution of Mining Engineers on the subject of silicosis and dust control in mines, at which papers on various aspects of the problem will be submitted.

New Professional Bodies.—During the year the Council were officially notified of the foundation of the Institution of Metallurgists, the Rhodesian Society of Engineers and the East African Association of Engineers. The Council have expressed their good wishes for the success and prosperity of these bodies.

Proposed programme of Boring by the Geological Survey of Great Britain.—Early in 1945 the Council were invited by the Chairman of the Geological Survey Board to express their views on a recommendation of the Advisory Council of Scientific and Industrial Research that the Board should consider the desirability of a programme of boring forming part of the normal work of the Geological Survey. This recommendation was warmly supported by the Council in a written reply to the Chairman, which was amplified by a deputation from the Council at a Meeting of the Board in April, 1945.

Colonial Mines Service.—In October last a deputation from the Council was received at the Colonial Office by Sir Charles Jeffries and other officials. The deputation expressed the view of the Council on the need for Inspectors of Mines in the Colonial Service to have had practical experience, preferably underground, before appointment, and that starting salaries should in consequence not necessarily be at the minimum rate. Study leave for Inspectors was recommended, and other points made were that experienced older men joining the Service should be given non-pensionable salary on a higher scale, that the disparity in salary between Geologists and Inspectors of Mines should be removed, and that the status of the Inspectorate should be improved by ensuring that the Chief Inspector has direct access to the Governor of his colony.

Misuse of Initials Signifying Membership of the Institution.—Three cases of the misuse of initials signifying membership of the Institution were reported to the Council during the year, and appropriate action was taken.

In conclusion, the Council wish to record their appreciation of the efficient service of the Secretary and staff of the Institution during the past year, which included the arduous work of re-arranging the Library in its new accommodation in Salisbury House.

By Order of the Council,

G. F. LAYCOCK, *President.*
W. J. FELTON, *Secretary.*

17th April, 1946.

DR. BALANCE SH

	£	s.	d.	£	s.
To Sir Julius Wernher Memorial Fund				10,147	0
,, Endowment Fund				46,933	4
,, Post Graduate Grants Fund				557	15
,, Life Compositions Account				3,664	10
,, Sundry Creditors				1,016	4
,, Reserve for Contingencies and Post-War Expenditure				7,000	0
,, Staff Superannuation Fund as at 31st December, 1944	8,565	9	8		
Add Transfer from Receipts and Expenditure Account	400	0	0		
Income from Investments	201	10	0		
				9,166	19
,, 'A. G. Charleton Prize' Trust Fund, as at 31st December, 1944	699	15	9		
Add Income for 1945	23	9	0		
				723	4
,, Subscriptions, etc., in Suspense				193	11
,, Accumulated Fund: Balance as at 31st December, 1944	37,766	4	3		
Less Transferred to Life Compositions Account	126	0	0		
	37,640	4	3		
Add Balance of Receipts and Expenditure Account for the year ended 31st December, 1945	600	5	4		
				38,240	9
				£117,642	19

J. G. LAWN, *Hon. Treasurer.*

W. J. FELTON, *Secretary.*

We have examined the foregoing Balance Sheet with the Books, Vouchers a ***of the Transactions*** **in hand, and subscriptions in arrear, are not inch**

LONDON, 29th April, **1946.**

ST DECEMBER, 1945. CR.

	£	s.	d.	£	s.	d.	£	s.	d.
y Sir Julius Wernher Memorial Fund:									
Investments, at cost:									
£376 Southern Railway 4% Debenture Stock				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			
£20,650 3% Local Loans				19,953	14	2			
£1,500 3½% War Stock				1,608	19	0			
£16,700 4% Consolidated Stock ...				17,734	4	3			
							46,933	4	3
,, Investments, at cost:									
£5,032 9s. 2½% Consolidated Stock ...				4,060	7	3			
£1,000 2½% National War Bonds, 1951/53				1,000	0	0			
£3,000 3% Savings Bonds, 1960/70 ...				3,000	0	0			
£750 3% Local Loans				730	17	6			
£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			
							46,298	5	10
,, Freehold Property, Mill Close, Darley Dale, Derbyshire, at cost, less amount written off							500	0	0
,, Staff Superannuation Fund (*commenced* 1924):									
Investments, at cost:									
£300 3½% War Stock				305	16	3			
£1,500 3% Local Loans				1,341	8	6			
£1,000 3% Defence Bonds				1,000	0	0			
£450 3% Savings Bonds, 1960/70 ...				450	0	0			
£2,850 2½% National War Bonds, 1946/48				2,850	0	0			
£950 2½% Consolidated Stock ...				762	4	7			
Insurance Policies: Premiums Paid ...				1,506	4	0			
Post Office Savings Bank				457	10	0			
Cash at bank				493	16	4			
							9,166	19	8
,, 'A. G. Charleton Prize' Trust Fund:									
£670 3½% War Stock, at cost ...				676	6	9			
Cash at Bank				46	18	0			
							723	4	9
,, Furniture and Library, as at 31st Dec., 1944				602	6	10			
Additions during 1945				49	7	1			
				651	13	11			
Less Depreciation at 12½% ...				81	9	3			
							570	4	8
,, Sundry Debtors							94	0	0
,, Cash at Bank				3,550	15	0			
Less on account of—									
Staff Superannuation Fund	£493	16	4						
'A. G. Charleton Prize' Trust Fund	46	18	0						
				540	14	4			
							3,010	0	8
,, Cash in hands of Secretary							200	0	0
							£117,642	19	10

...ankers' Certificates, and certify it to be in accordance therewith. The value ...mong the *assets*.

WOODTHORPE, BEVAN & CO.,
CHARTERED ACCOUNTANTS, } Auditors.

Dr. *Receipts and Expenditure f*

1944. £	s.	d.		Expenditure.	£	s.	d.	£
			To	Cost of Printing Publications: *Transactions*, *Bulletin*, advance copies of Papers, reports of Discussions and on account of Volume 53, *less* receipts for the Advertisement Section of the *Bulletin*				
1,278	5	6						885
248	15	11	„	Printing and Stationery ...				210
36	7	2	„	Expenses of Meetings				43
33	15	8	„	Insurance				26
1,126	2	4	„	Rent, Heating, Lighting, Cleaning, etc.				1,265
22	3	3	„	Telephone...				25
89	5	0	„	Audit Fee...				89
4,161	10	0	„	Salaries and Retiring Allowances				4,449
204	4	6	„	Postage and Receipt Stamps ...				234
156	2	7	„	Sundry Accounts...				303
			„	Library at Mill Close:				
				Salaries, Rates, etc.	£283	7	3	
				Less Contribution received from the Institution of Mining Engineers ...	96	0	0	
356	8	6						187
			„	Subscriptions:				
105	0	0		Ross Institute of Tropical Hygiene				52
50	0	0		British Standards Institution				25
–	–	–	„	National Certificates in Metallurgy:				
–	–	–		Proportion of Expenses ...				162
			„	Joint Committee on Metallurgical Education:				
–	–	–		Proportion of Expenses ...				25
14	18	0	„	Repairs and Renewals				38
–	–	–	„	Cost of Removal of Library ...	264	6	10	
				Less Proportion received from the Institution of Mining Engineers ...	132	2	11	
								132
			„	'Consolidated Gold Fields of South Africa Ltd.', Premium for 1944				
42	0	0						42
400	0	0	„	Staff Superannuation Fund ...				400
16	12	6	„	Interest on Post Graduate Grants Fund				16
86	1	0	„	Depreciation — Furniture and Library				81
2,000	0	0	„	Reserve for Contingencies and Post-War Expenditure ...				1,500
288	19	4	„	Excess of Receipts over Expenditure				600
£10,716	11	3						£10,796

Year ended 31st December, 1945. Cr.

Receipts.	£	s.	d.	£	s.	d.	1944. £	s.	d.
By Entrance Fees and Subscriptions:									
Entrance Fees	288	15	0				*287*	*14*	*0*
Subscriptions (including £486 4s. 6d. received in advance)	6,445	2	6				*6,474*	*5*	*9*
Life Compositions	126	0	0				*63*	*0*	*0*
				6,859	17	6			
" Sales of *Transactions*				330	6	3	*339*	*18*	*0*
" Interest and Dividends, including repayment of Income Tax for 1944–45				3,564	6	0	*3,509*	*13*	*6*
" 'Consolidated Gold Fields of South Africa Ltd.', Premium for 1944				42	0	0	*42*	*0*	*0*
				£10,796	9	9	*£10,716*	*11*	*3*

Subject to revision.] [*A Paper issued on May 9th,* 1946, *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, June 20th,* **at 5.0 o'clock p.m.**

Sandfilling at Mufulira.

By A. C. Turton.*

Introduction

Mufulira mine is situated in the Copperbelt of Northern Rhodesia and lies in latitude 12° 32′ south, longitude 28° 25′ east, at an elevation of 4,100 ft. above sea level. The topography of the district is one of low relief and the vegetation is of the savannah type, with an open growth of trees 40 ft. to 60 ft. high. The climate is subtropical, with regular wet and dry seasons. The average rainfall in the wet season from November to April is about 50 in.

The Mufulira mine covers a strike (N.W.–S.E.) of 7,000 ft. to 8,000 ft. and consists of three separate, flat, lenticular, superimposed ore-bodies ranging from 10 ft. to 60 ft. in true thickness and separated from each other by barren rock varying from 0 ft. to 50 ft. in true thickness. The two lower ore-bodies (B and C) merge in the central part of the mine for a strike distance of 3,000 ft. to 4,000 ft. and down the dip for upwards of 2,000 ft. The dip (N.E.) varies from 30° to 55°.

The ore occurs in hard medium- to fine-grained quartzite, the ore-bodies being conformable to the bedding; the only noteworthy features from a ground-support point of view are:—

(*a*) *The Mudseam*, which forms the hanging-wall of the C, or lower, ore-body. This is a soft mineralized sandstone, 3 ft. to 6 ft. thick, which is of little value in a supporting pillar.

(*b*) *The Lower Dolomite* (15 ft. thick), which forms the hanging-wall of the B., or middle, ore-body. This is completely weathered to black manganese wad to depths of from 250 ft. to 900 ft. below the surface.

The ore-bodies are not mined above the 250-ft. level from the surface, as leaching has removed the copper content.

The foot-wall country rock is a cross-bedded sandstone, generally poorly consolidated to a depth of 1,000 ft.

*Formerly Sandfilling Foreman, Mufulira Copper Mines, Ltd.

The hanging-wall country rocks are quartzites, dolomites, and shales, in general softer than the ore-bodies, weathered to a depth of over 1,400 ft., and containing large quantities of water.

Reasons for Adopting Sandfilling

In the upper portion of the mine—i.e., above the 460-ft. level—the ore-bodies were mined separately (*vide* Fig. 1, Plate I), using the sub-level open stoping method. It was found that the A, or upper, ore-body could be mined in some instances for a stope-strike width of over 100 ft. without signs of collapse of the hanging-wall and over 1,000 ft. of strike was mined, with stope widths varying from 60 ft. to 100 ft. and pillars of 10 ft. to 15 ft., without collapse of the hanging-wall.

In the B ore-body, where a skin of 6 ft. was left in the hanging to support the manganese wad, a stope strike width of 60 ft. was found to be the maximum to permit completion of the stope before collapse of the hanging-wall skin.

In the C ore-body it was found that the mining of two adjacent 60-ft. strike-width stopes was the maximum to permit completion before collapse of the hanging-wall, which in this case generally involved the overlying ore-bodies and caved to the surface.

The need for some method of support for the hanging-wall in mined stopes was therefore obvious for the following reasons :—

(1) To prevent flooding of the mine by caving into the overlying water-bearing formation.

(2) To prevent air blasts. (With three superimposed openings totalling a thickness of 120 ft. to 170 ft. and a possible almost instantaneous collapse the hazard to life is considerable.)

(3) To attain predictable ore extraction.

The only suitable material readily available for hanging-wall support is the sand from mill tailings, so that experiments in this direction were carried out between 1935 and 1937, when sandfilling of stopes was started on a major scale. This has now attained what is probably the maximum required—i.e., 100,000 to 130,000 tons of sand placed per month.

Mining System

The general system of mining at Mufulira from the 260-ft. level to the 900-ft. level is the sub-level open-stope bench and trail method, with haulages at the 460-ft., 660-ft., and 900-ft. levels. The stopes are 60 ft. wide along the strike, with a back of 350 ft. down the dip, and a 15-ft. pillar between stopes.

Sequence of Stoping.—At the commencement of stoping at the mine the sequence of mining was to stope out the ore-bodies in the order A, B, then C, or A, then B–C (if the latter were mined

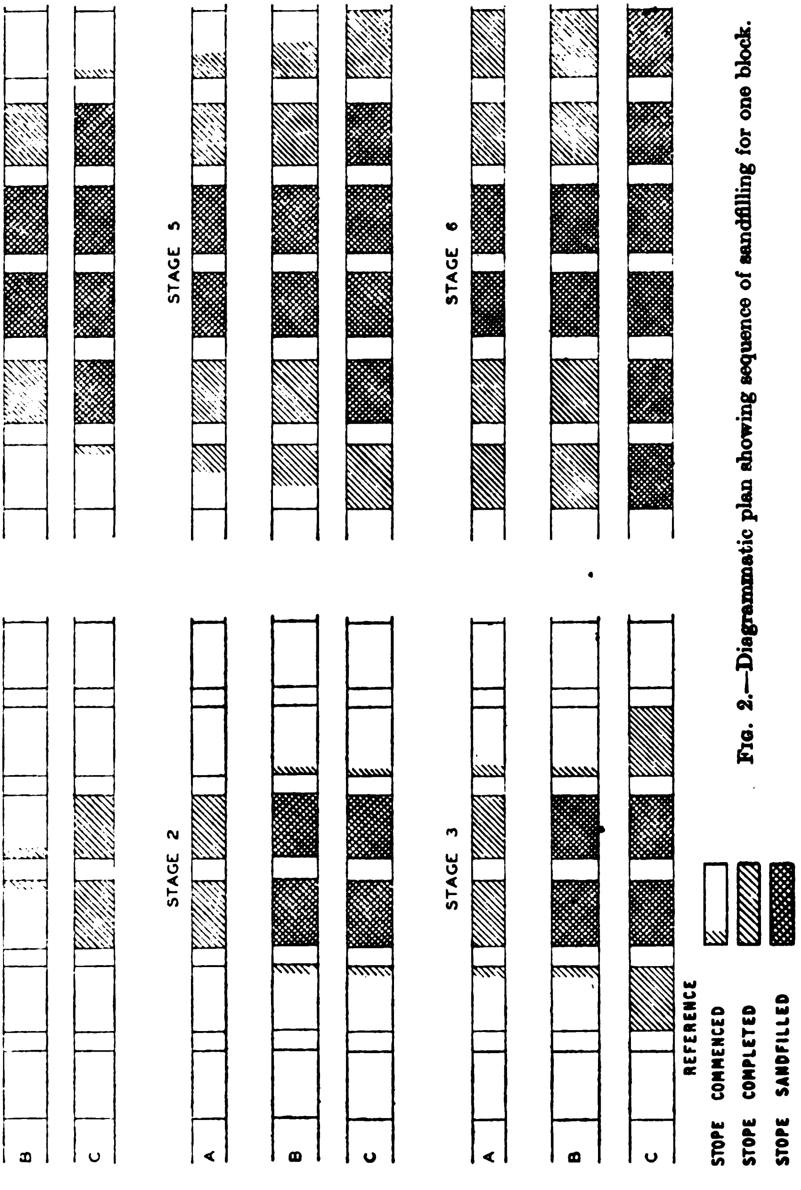


FIG. 2.—Diagrammatic plan showing sequence of sandfilling for one block.

together), but later the sequence of stoping was reversed—the order being now C, B, A.

Fig. 2 shows in graphic form the stoping and filling sequence. In a developed block the two centre C stopes are mined first, commencing from the centre pillar and retreating towards the ladderway. When the two C stopes are complete the overlying B stopes begin breaking ore, and bulkheading of the drawpoints and sub-levels in the pillars of the C stopes starts immediately ; when

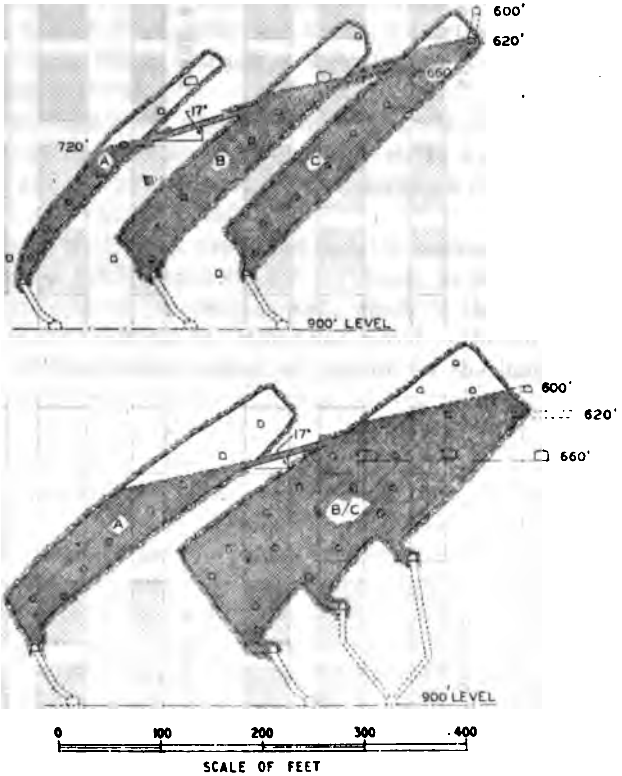


FIG. 3.—Section through ore-bodies showing typical sandfilling sequence.

the concrete is sufficiently set the sand is run in from the top levels. As filling proceeds the slotting of the two adjacent C stopes follows the sand level. When the C ore-body stopes are about filled the B stopes are usually mined out and bulkheaded and the sand flows through raises at 15° to 17° grade into the B stopes, and so on, to the A ore-body stopes later (Fig. 3).

When the sand-filling programme was laid out it was assumed that 40 per cent of the ore mined would be recovered as sand for filling and as one ton of sand—settled and drained—occupies 20 cu. ft., as compared with solid rock at 13·3 cu. ft. per ton :—

100 tons of ore at 13·3 cu. ft./ton = 1,330 cu. ft., and
40 per cent recovery of sand at 20 cu. ft./ton = 800 cu. ft.,

therefore the sand recovered would occupy 60 per cent of the space of the ore mined. (Note: The short ton of 2,000 lbs. is used throughout the paper.)

With this point in view the programme was laid out to fill the whole of the third ore-body, or C, stopes and two stopes of the B ore-body and two of the superimposed A stopes in each block where they were mined separately, as well as all the B–C stopes *plus* two A stopes in each block where the former are mined together. In this method there would be a solid pillar of sand 150 ft. wide extending from the foot-wall to the hanging-wall of the ore series.

At the present time the recovery of sand has increased to 55 per cent; thus approximately 75 per cent of the volume of the stopes can be filled, allowing for loss of sand through shutdowns. Therefore practically all of the B stopes can be filled as well as the C stopes.

Experimental Work on Sandfilling

The gangue is quartzite and graywacke with an average silica content of 75 per cent, and a typical screen analysis of the tailings in 1935 was :—

Solids per cent	+65	+100	+150	+200	+325	—325
16·8	1	6	12	14	20	47

The most important requirement of a sand-filling material is that it must leach rapidly, since it is placed in a fluid condition and hydraulic pressure must be reduced. For this purpose in 1935 a Dorr classifier with 16-ft. bowl was installed at 2 Shaft and a pipeline 3 in. in diameter was laid down the shaft (40° incline) to a small stope, B. 19E, above the 360-ft. level and adjacent to the shaft. Some 2,800 tons of sand were duly placed in this stope and showed conclusively that the mill tailings, deslimed and dewatered, made an efficient support for the hanging of the stopes, having the requisite qualities for leaching.

A typical screen analysis of this experiment is as follows :

Tailings Feed	*Solids per cent*	*Screen Gradings*					
		+65	+100	+150	+200	+325	—325
41 cu. ft./min.	16·8	1	6	12	14	20	47
Sand discharge	70·6	6	18	25	27	8	16
Overflow	12·6	—	1	6	7	32	54

Stopes C.20 and C.21 east of 2 Shaft and above the 460-ft.

level were then filled and the classifier moved to 8 Shaft in November, 1937, for filling stopes C.1, C.2, C.3 East, and B.1, B.2, B.3 East, and B.0 and B.1 West.

Meanwhile a churn-drill bore-hole was put down from the surface to the 260-ft. level at C.7 West and cased with 6-in. casing to solid at 170 ft., and a single steel dewatering cone 9 ft. in diameter and 9 ft. deep erected over the bore-hole. This cone, with a tailings

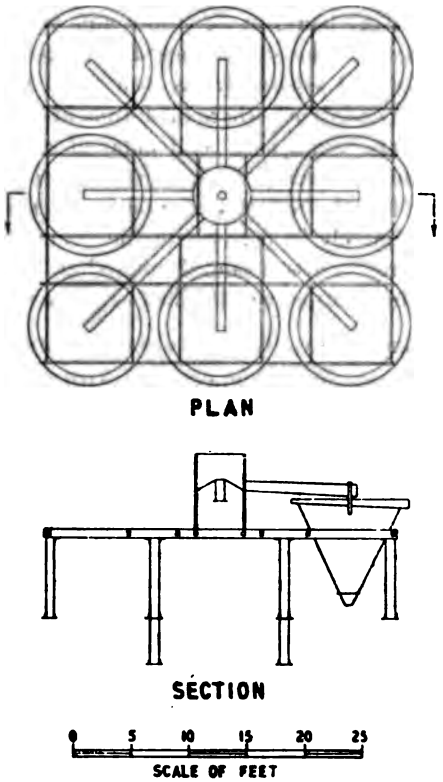


FIG. 4.—Structure for 8-cone battery.

feed of 24 cu. ft./min., and density of 36 per cent solids, proved to recover more of the finer sands than the Dorr classifier, having the same density of feed. This loss from the classifier was due to the agitation of the rakes.

Bore-holes were then sunk to the upper service level (240 ft. to 260 ft.) at the intersection of the ladderways, which are raised in the foot-wall. Owing to the overburden of approximately 100 ft. and the loose friable nature of the rock above the 240-ft. level, these bore-holes were drilled 12 in. in diameter and lined throughout with 4-in. I.D. reinforced concrete pipe. The whole string of concrete pipes, each 6 ft. in length with bell and spigot joint caulked with cement and sand as lowered, is inserted into the bore-hole suspended on N diamond drill rods. The bore-hole at the holing of the ladderway is caulked round with coconut matting and the space between the pipe and the wall of the hole filled up with concrete—1:3:5 mixture (the crushed stone of *minus* ½-in. mesh). Steel lining of the bore-holes has a very short life, owing to the abrasive action of the sand.

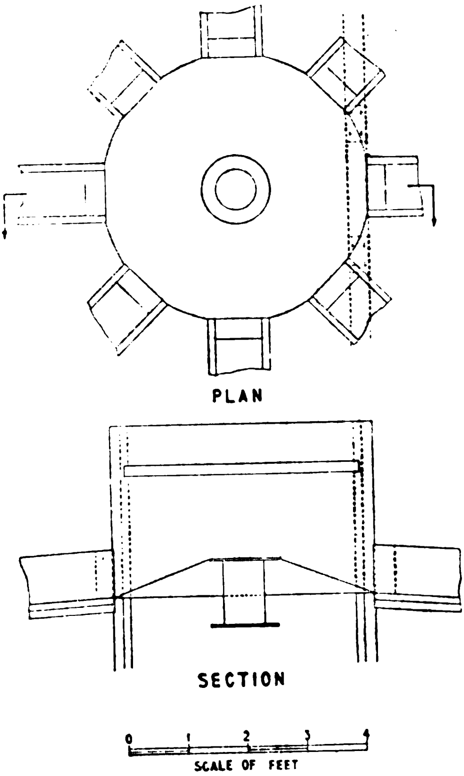


FIG. 4A.—Distributor for 8-cone battery.

Four batteries, each of eight steel cones, with battery capacity of 1,000 tons of sand per 24 hours (Figs. 4 and 4A) were erected and run simultaneously with the classifier at 3 Shaft, the programme being to move these cone batteries from ladderway to ladderway as the stopes came in for filling, but, in practice, it was found that time taken to move a battery (approximately 30 days) was time lost and sand lost to stopes. Therefore, a desliming battery of

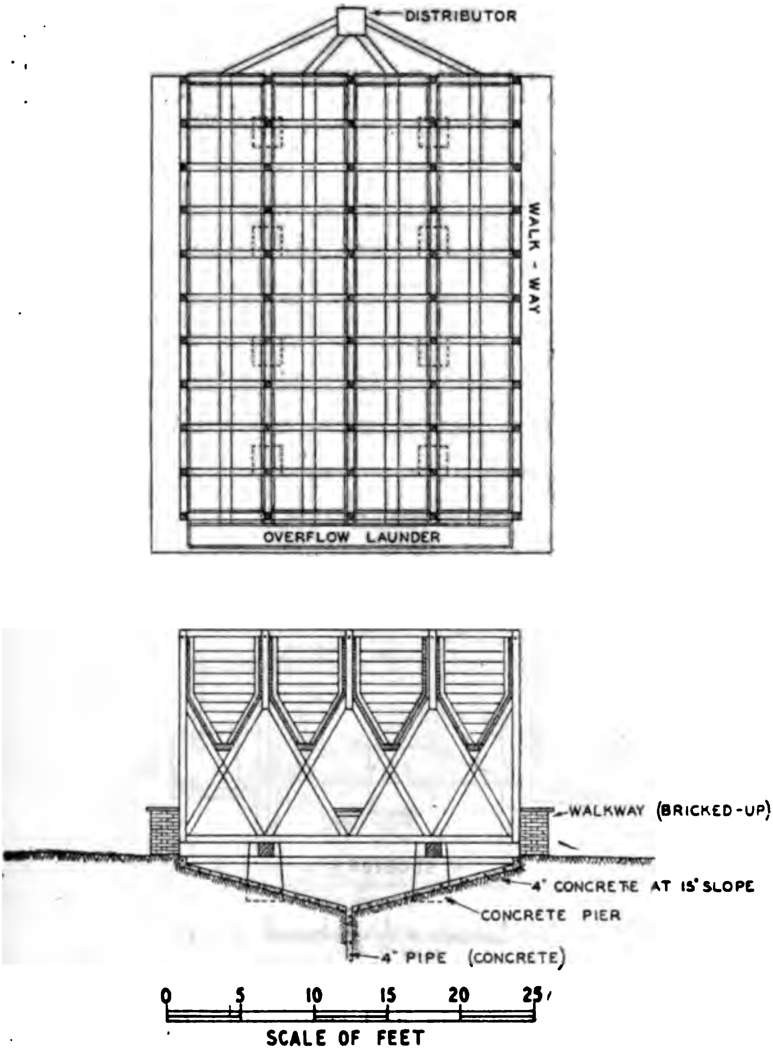


FIG. 5.—De-watering cones.

cones is now erected at each bore-hole, in order that any stopes can commence filling immediately stoping has been completed and the drawpoints and lower sublevels bulkheaded.

The desliming cones shown (Fig. 5), with either two or four compartments, depending on the sand requirements of the section to be filled, were evolved. These are constructed from local timber

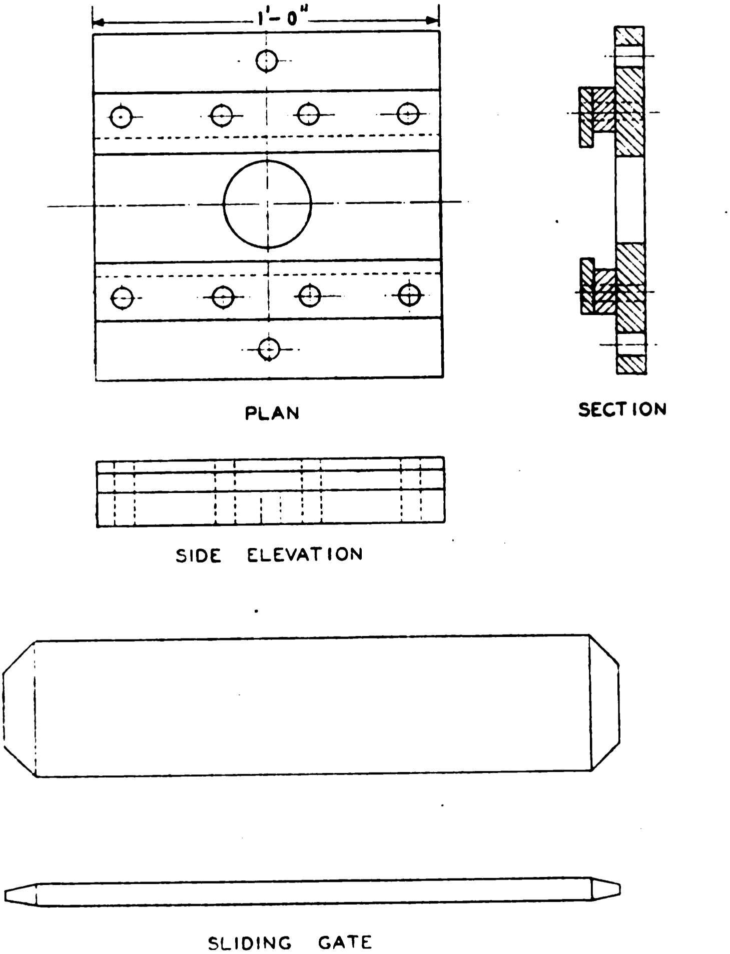


Fig. 6.—Discharge door.

(6-in. by 6-in. framing) and Congo (9-in. by 2-in.) tongue-and-grooved planking, with discharge gates (Fig. 6) at the bottom of each compartment. These desliming cones have proved very satisfactory, with a recovery of 58 per cent of total solids, and a capacity of 1,400 to 1,500 tons of sand per 24 hours.

An average screen analysis of a battery of four desliming cones is as follows :—

Tailings Feed	*Solids per cent*	*Screen Gradings*				
		+100	+150	+200	+325	—325
70 to 90 cu. ft./min. ...	34	14	19	16	14	37
Underflow................	72	30	29	20	11	10
Slimes overflow	19	1	3	9	13	74

Surface Layout

The general layout of the sand-filling plant at present is shown in Fig. 7. A length of 2,000 ft. of 15-in. I.D. concrete pipe-line, reinforced for 30 lb. internal pressure, is laid from the mill thickener to 3 Shaft at 40 ladderway, with four cast-iron tees, spaced at approximately 500-ft. intervals, for the tailings feed to the batteries west of 3 Shaft. At this point an 8-in. Wilfley sand pump with an 8-in. Victaulic pipe column and a 6-in. Wilfley pump with 6-in. pipe column laid on surface are used to boost the tailings east of 3 Shaft. The 6-in. pump is used as far as 22 ladderway and the 8-in. to the eastern fringe of the mine. Both columns are tied in to the batteries at 22, 28, and 34 ladderways. This system is arranged for flexibility and either one or both pumps run, depending on the volume of tailings available.

The branch lines to the batteries are 6-in. flanged standard pipe with a ½-in. steel plate faced with conveyor belting, having an aperture of a diameter to feed 70 cu. ft. per minute of tailings to each battery. Very little wear takes place in the 8-in. and 6-in. tailings pipes, as a furring $\frac{1}{16}$-in. to ⅛-in. in thickness forms from the slimes and lime, giving a lining of a vitreous nature to the metal. The 8-in. column has been in constant use for six years.

An 8-in. Sulzer pump (low head) delivering 1,200 gallons of water per minute is installed where the 15-in. concrete tailings-main crosses the mine water furrow—just north of the Dorr thickener—and pumps water directly into the tailings pipe in order to reduce the density of the tailings from 56 per cent to 34 per cent solids.

It might be noted that in 1941 the mill increased the density of the tailings from around 37 per cent to over 56 per cent, with the *consequence* that desliming was bad and precipitation of sand

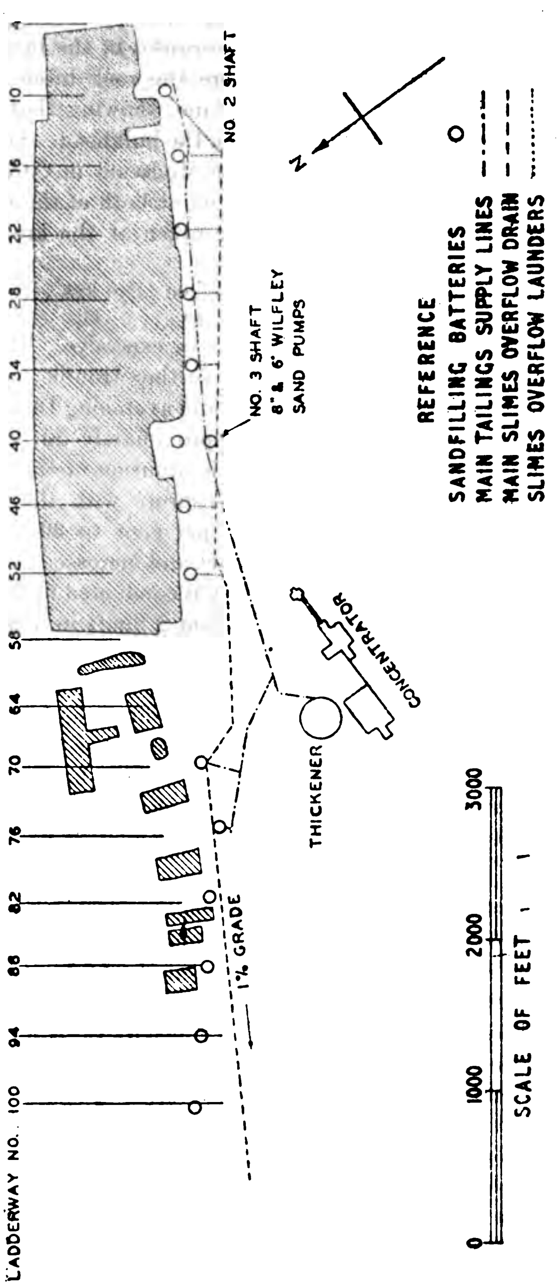


FIG. 7.—Surface layout of sandfilling plant.

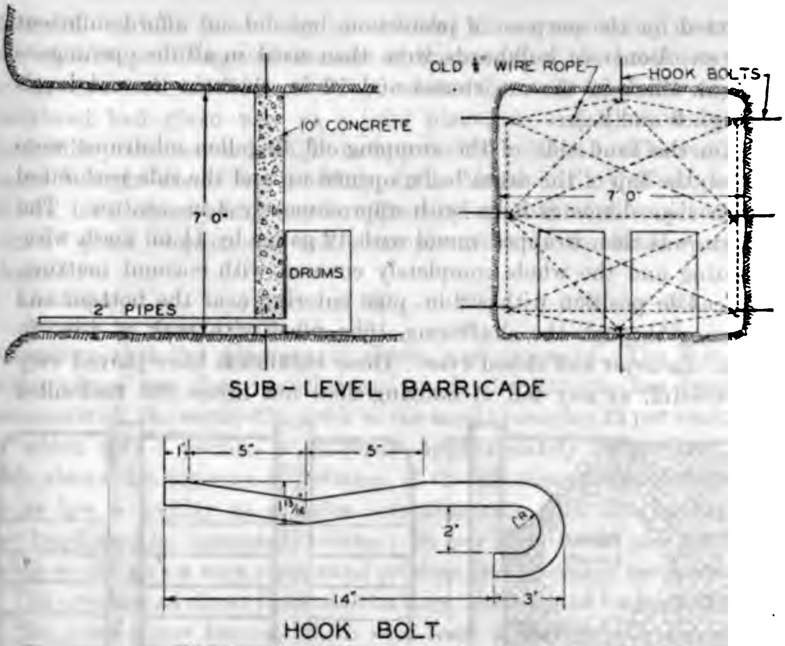


Fig. 9.—Sandfilling barricade—detailed.

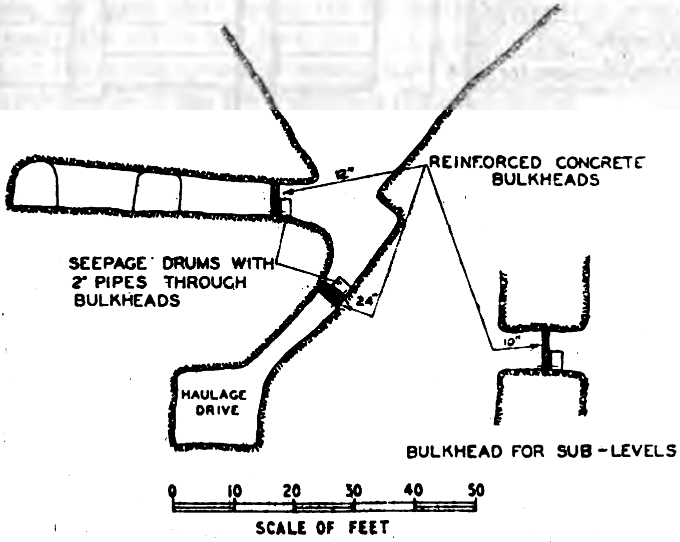


Fig. 9a.—Sandfilling barricades.

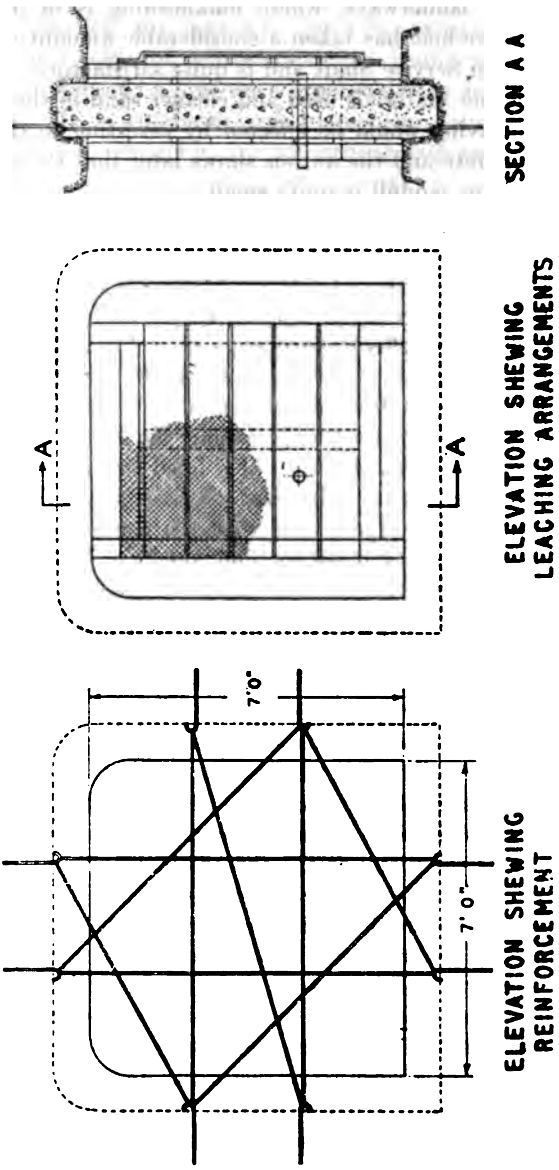


Fig. 10.—Sandfilling bulkhead.

(sand and crushed stone) is dropped down the bore-hole on alternate days. From the bore-hole at the 660-ft. level the material is trammed to the ladderways, where bulkheading is in progress. This material bore-hole has taken a considerable amount of extra work off the Main Service Shaft and is quite satisfactory.

With the cleaner materials used and cleaner sand in the stopes, a considerable saving would be effected by reverting to the 1:3:5 mixture for concrete and the author shows later that the pressure on bulkheads from sandfill is quite small.

Sand Delivery Pipes

As stated previously the sand-filling bore-holes are sunk to the 260-ft. level at the ladderways. At this point 6-in. by 45° cast-iron laterals are placed to effect the change over from the concrete lining of the bore-holes to the 4-in. sand-delivery pipes, which are laid down the raises to the delivery points; in the case of the 900 to 660-ft. level stopes these will be along the 600-ft. level.

Originally some 5,000 ft. of 4-in. rubber-lined pipe in 8-ft. lengths was purchased (the rubber vulcanized to the metal of the pipe) and the majority of this pipe is still in use; it is pre-eminently the most suitable pipe to stand up to sand abrasion. There is also less friction owing to the smooth rubber surface. One string of rubber-lined pipe (down 3 Shaft—40° incline) has passed 750,000 tons of sand and a section cut across a test pipe shows a $\frac{3}{16}$-in. thickness of rubber wall left from the original $\frac{1}{4}$-in. thickness.

When further rubber-lined pipe was unobtainable, owing to the rubber situation, recourse was made to ordinary 4-in. standard pipe. These were random lengths of 18 ft. to 20 ft., screwed and flanged with standard 4-hole flanges. These proved unsuitable, for although the pipes were turned a quarter turn every week to equalize the wear the metal at the threads had been cut through after one month's use—30,000 to 40,000 tons of sand—and although there was further life left in the barrel of the pipe the walls were too thin for re-threading. A consignment of Spang piping then came forward. This has a $\frac{9}{32}$-in. wall and is cut into 10-ft. lengths; 8-hole flanges are skimmed out and welded to the pipe and 10-ft. lengths are easily handled in raises, etc. These pipes are turned $\frac{1}{8}$ of a turn every two weeks and this goes on till the wall is too thin to continue their use. Their life is from 4 to 5 months—i.e., 150,000 tons of sand. All bends are rubber-lined; not greater than 45° and as great a radius as possible.

It is the practice at Mufulira to keep sufficient rubber-lined pipes of 8-ft. length and to bend these with pipe benders to the required curve, using two lengths to complete a curve of 90°. Ordinary standard bends of 45° cut through after three or four days' wear and are continually replaced. An excellent first-aid device is coconut matting cut into 9-ft. by 1-ft. strips; this bandage is steeped in a paste of neat cement and bound tightly round the bend or pipe should a hole wear through due to denting with rough handling. These bandages have frequently outlived the pipe.

A line of 4-in. diameter pipe will deliver with ease 2,000 dry tons

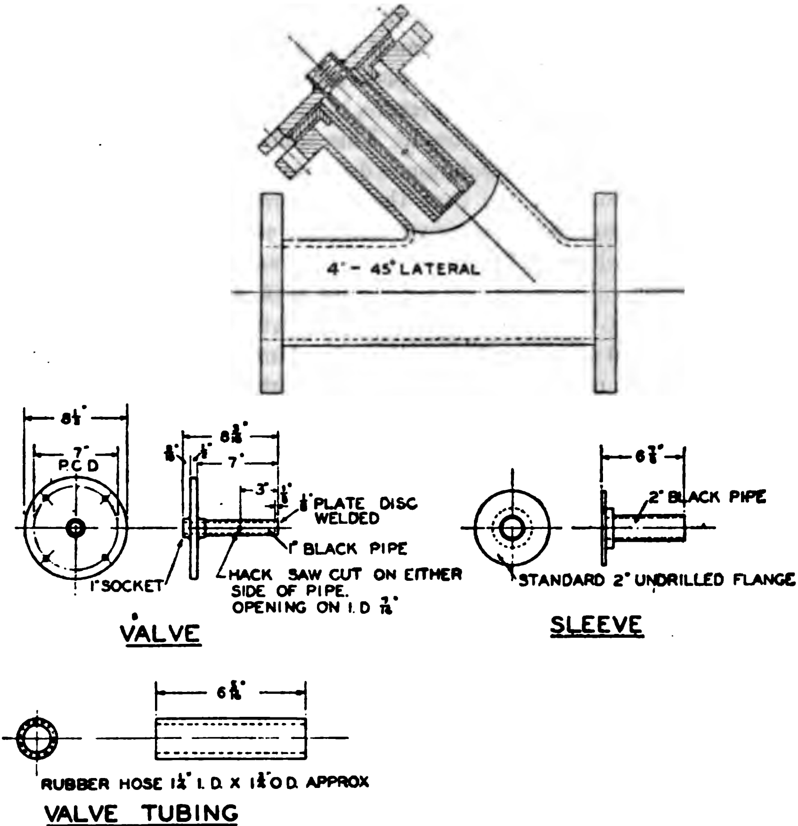


FIG. 11.—Non-return valve for compressed air into 4-in. sand line.

of sand, having a moisture content of 32 per cent in 24 hours.

Frequently the sand has to be boosted considerable distances along the horizontal sub-drive. After considerable experimentation a compressed-air non-return valve has been evolved. It is simplicity itself, very economical on air, and efficient. The only care required is a weekly check over to see that the rubber tubes have their elasticity and the air vents are clear. All the material, except the rubber hose, is readily obtainable. The rubber tubing is $1\frac{1}{4}$-in. inside diameter and $1\frac{3}{4}$-in. outside, soft rubber with no reinforcing fabric. The hose is renewed every six weeks. Some 40 tons of sand per hour have been boosted continuously day after day with an air consumption of 2,880 cu. ft. per hour for 1,200 ft. horizontally, or 72 cu. ft. of air per ton of sand. This is the greatest distance it has been necessary to transport the sand heretofore. The possibility is that this distance can be doubled.

The 4-in. lateral for the air valve is placed in the sand line 80 ft. to 100 ft. from the bottom of the ladderway along the sub-drive, and a diaphragm of $\frac{1}{2}$-in. plate faced with conveyor belting, both having a 3-in. diameter aperture, is placed in the sand line on the approach side of the lateral. This baffle ensures the bore of the pipe being full for a minimum distance up the raise of 50 ft. If the air is inserted too near the ladderway it will blow back towards the surface, dewatering the sand and causing a choked pipe and bore-hole. Fig. 11 shows the air valve in detail.

Air pressure below 70 lb. per square inch is not satisfactory, so a 300-cu. ft. Holman compressor is held as a stand-by should the pressure of air from the main power plant drop, as it usually does after the mining shift. An independent 2-in. air line is laid down 40 ladderway to the 600-ft. level, where it divides and runs east and west, 2 in. for the first portion of the branches, and then reduced to 1 in. pipe with tees at ladderways to supply air to the valves. The change over from main air to auxiliary compressor is made immediately and no drop in pressure is allowed.

Since the use of compressed air the 4-in. pipes along the levels are *not* turned, the wear taking place evenly all round the bore. The sand is emitted intermittently—a gush of full bore for a few seconds, then nothing for about the same period.

Personnel

1 foreman.
1 charge hand with five boys; maintenance of cones and surface lines and pumps.
1 carpenter and six boys; erection of cones, launders, etc.
2 pipefitters: six boys each; underground sand lines.
3 operators, one on each shift, each with 20 boys.
1 boss boy.
2 boys per battery.
1 telephone boy.
1 underground leading hand, in charge.
1 pipe boy (examiner).
4 barricade boys.

The erection of sand-filling barricades in the stopes is done by the underground timbermen under the supervision of the mine captains. An average of six timbermen, each with eight boys, is usually employed on barricades.

Operation

Batch classification is now the system on all the batteries of cones—i.e., the tailings are turned into the cones and when the sand is level with the overflow lip the feed is turned into the next and the full cone of sand is discharged down the bore-hole. In this way the slime content is more under control. It will be seen then that with a battery of eight steel cones four will be discharging sand and the other four filling up. Originally automatic discharge valves were fitted, which were operated by sand friction, but were not completely successful owing to occasional sticking. All discharge gates are now operated manually.

Each steel cone when full contains nine tons of sand and each compartment of the wooden desliming cones contains 40 tons of sand. A tally board is kept at each battery and a tally kept of the number of cones discharged on each shift. The sand discharges at the gate with a moisture content of 28 per cent, or 72 per cent solids, which is too heavy to run along the pipes.

A water gauge (Fig. 12) is fixed over the bore-hole and clean water to liquify the sand to 68 per cent solids is added. The 1-in. valve is adjusted to run the requisite number of holes (usually four). This system is easily understood by the native attendants and is entirely satisfactory, the quantity of water discharged from the orifices being: 3 holes, 7 gallons; 4 holes, 9 gallons; and 5 holes, 13 gallons per minute.

When a stope, or succession of stopes, is to be filled, 700 to 1,000 tons of sand are run in, then closed off for two days or so to allow the sand to leach off water thoroughly and set. Filling is then continued to bring the sand up to each succeeding sub-level (30 ft. apart vertical distance) where the free water, which never exceeds 3 ft. at the deepest point, is decanted off through the barricades. This battery is then shut down for three days and the operator starts up another set, which has had a period of rest. At any time—whether the sand level has reached a sub-level or not—if the filling is closed down for six hours the surface of the sand is entirely free of water, and is compact and hard, barely leaving a footprint. Blocks of six stopes are filled with as equal facility as one stope. In fact, the only objection is whether the open stopes will stand up long enough before caving.

There is a small amount of classification of the sand in the stopes.

The sand is poured into the stope at the foot-wall and flows towards the hanging-wall, where the free water collects between sub-levels, and it is in this water that classification takes place—the coarse particles depositing immediately at the water's edge and grading off to the fines at the hanging-wall. This is not as serious as it may at first appear. The sand pours into a stope at one point at the foot-wall and in the majority of cases the stope is 60 ft. along the strike and 170 ft. to 180 ft. from foot to hanging, and from the point of entry the flow is either to the eastern pillar or western pillar or north to the hanging for hours at a time, driving the small pond of water ahead to different positions in the stope.

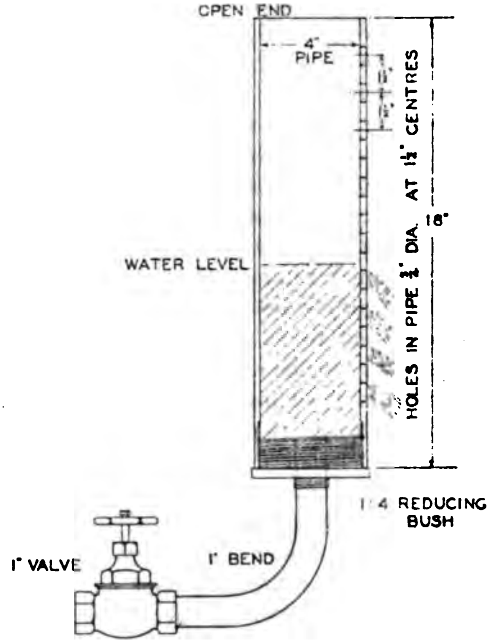


Fig. 12.—Water gauge for adding water to sand.

Owing to inaccessibility it is not possible at Mufulira to fill from the hanging-wall side or to distribute the sand over the whole area by means of hoses.

Drainage of interstitial moisture from sand in stopes ceases in two months or so from the time filling has stopped and the moisture content of sand which has been lying in stopes for two years averages 11 per cent. A considerable amount of initial moisture is absorbed by fissures and cracks in the rock, particularly in the foot-wall. From practical experience in the stopes at Mufulira the angle of repose of sand is : 66 per cent solids, 9° ; 68 per cent solids, 12° ; 70 per cent solids, 15°.

When flowing long distances from stope to stope through the sub-level openings in the pillars the sand is dewatered by absorption *en route* and at 76 per cent solids ceases to flow—this over a distance of 360 ft. to 400 ft.

Percolation Rate

Tests were run on a number of sand samples to determine the rate of percolation. For this purpose a glass tube was used of $\frac{11}{16}$-in. diameter and the charge of sand was 50 grammes, resulting in a column of sand just under 3 in. long. Water was added after the sand and successive 50-c.c. additions were allowed to percolate through the wet sand bed. The figures (Table I) shown on time of draining are the average of eight tests each.

Table I

Product	*Screen Gradings*	*Draining Time*	*Product*	*Screen Gradings*	*Draining Time*
Sand	+ 65 : 9·8		Sand	+ 65 : 11·5	
from	+100 : 31·4		from	+100 : 31·3	
Steel	+150 : 29·6	8·75	Wooden	+150 : 27·6	11·34
Cones	+200 : 13·9	mins.	Cones	+200 : 13·8	mins.
	+325 : 5·3			+325 : 5·7	
	—325 : 10·0			—325 : 10·1	

An average percolation rate of 1 ft. in 40 minutes.

Volumetric Screen Analysis :—A sample of composite final tails (250 g.) was screen sized into the gradings +100 to —325 mesh. Dried fractions were weighed and measured volumetrically in a glass cylinder by tapping until no further consolidation took place.

Table II

Mesh	*Per cent Weight*	*Per cent Volume*
+100	14·5	13·4
+150	15·4	14·9
+200	13·3	12·5
+325	10·2	9·7
—325	46·6	49·5
Whole tails	....	77·2

The results are given in Table II, showing that the volume of the sum of the fractions considerably exceeds the volume of the composite due to interstitial consolidation.

Doming of Sand:—Experiments were carried out in a wooden box 8 ft. square and 10 ft. high to contain 32 tons of sand when full. An opening 12 in. square was cut in the side near the bottom and another opening 18 in. square in the centre of the floor. These openings were closed loosely with doors lined with coconut matting. The box was then filled in 40 minutes with sand of average screen gradings and at 68 per cent solids. When the moisture content dropped to 22 per cent or 78 per cent solids, which was in 90 minutes from the time the box was full, the doors were immediately opened. The sand exposed at the side door did not in any way show signs of sloughing off, but remained perfectly perpendicular with no bulging and, of course, no free moisture.

The bottom door when opened discharged a small amount of sand at once, causing a dome, the top of which was 27 in. above the opening. This dome remained till the following day, when the box was emptied and the experiment repeated with all conditions being equal, with the exception that the bottom opening was increased to 24 in. square, when the height of the doming was 36 in. The following day the opening was increased to 36 in. square and, the other conditions being equal, the doming was 54 in. high.

The sand was then left in the box for 30 days, open to the weather and rainfall, which, during that period, was 15 in. The dome remained the same, but there was considerable dripping of water from the bottom of the box.

From this experiment, it would follow that the doming height of sand is:

$$H = W \times 1{\cdot}5$$

where H = height, and W = width of opening.

Quantities.—Milling 200,000 dry tons per month = 278 tons per hour.

278 tons solids with 66 per cent moisture gives a ratio of solids to solution of 1 : 1·94.

278 tons + 539·3 tons water = 817·3 tons tailings per hour at 25·15 cu. ft./ton.

= 20,555 cu. ft. of tailings per hour—which gives a feed of approximately 70 cu. ft./min. to each of five batteries.

From the average of pulp and sludge density samples, taken every 2 hours throughout, an average of 34 per cent solids in tailings and 20 per cent solids in overflow was obtained.

From the formula below—where F., S. and Q. represent dry tons of feed, sand, and overflow, and f., s. and q., per cent moisture in feed, sand and overflow respectively:

$$S = F \times \frac{100 - s}{100 - f} \times \frac{q - f}{q - s}$$

$$= 278 \times \frac{100 - 28}{100 - 66} \times \frac{80 - 66}{80 - 26}$$

$$= 153 \text{ tons per hour.}$$

A rate of 153 tons per hour operating 600 hours per month gives 91,800 tons per month. Using the tally system on the same tonnages per month gives 102,000 tons, a discrepancy due to various causes.

Costs

The average cost to date of sandfilling since the commencement is 0.57s. per ton of sand placed. This is split into four main headings:

	Shillings Per ton
Surface:	
Includes tailings pipe lines and sand pump spares; cost of cones and erection maintenance; operation of sandfilling cones	0·148
Underground:	
Includes bore-holes and lining, 4-in. sand pipes to stopes, and bulkheading of stopes...	0·405
Pumping:	
Leaching water from stopes	0·015
Overflow slimes disposed by gravitation	0·002
	0·570s.

Notes

The tailings pipe-lines choke occasionally owing to insufficient velocity of tailings, caused by worn pump impellers, in which case the pump should be repaired and the pipe line cleaned at once, otherwise the sand sets and the line will have to be disconnected at every joint. If done at once a disconnection every 100 ft. is sufficient.

Underground 4-in. sand lines choke frequently—mainly owing to holes being worn at bends, or the blowing of gaskets. In this case, it is most important to repair the line and clear at once. The average time taken to clear 1,000 ft. of 4-in. line is 8 hours. If left for more than 24 hours it has taken three shifts.

A combination of water and compressed air is used to clear pipes. Air alone does more harm than good by drying out the sand in the pipe.

When starting up a battery a full 1-in. pipe of water is run down for 15 minutes to flush the sand line before opening up the sand, and after closing down the flushing is repeated, and the borehole plugged.

When wooden desliming tanks are closed down they are always filled to capacity with sand, which prevents the timber drying out, thus avoiding leaks, and preserving the wood.

Coconut matting on the bulkheads should be covered with sand within six months, or rot sets in, due to the warm humid atmosphere of a closed-in stope. If covered with sand, matting lasts practically indefinitely.

Telephonic communication with the surface operators is maintained by means of a manual telephone on the 600-ft. level, which is the main distributing level in the mine, and also by automatic telephone from all the main haulages. The leading hand in charge of the barricade guards reports by phone at least once every two hours to the operator on shift.

TABLE III

ORE REMOVED AND SANDFILLED

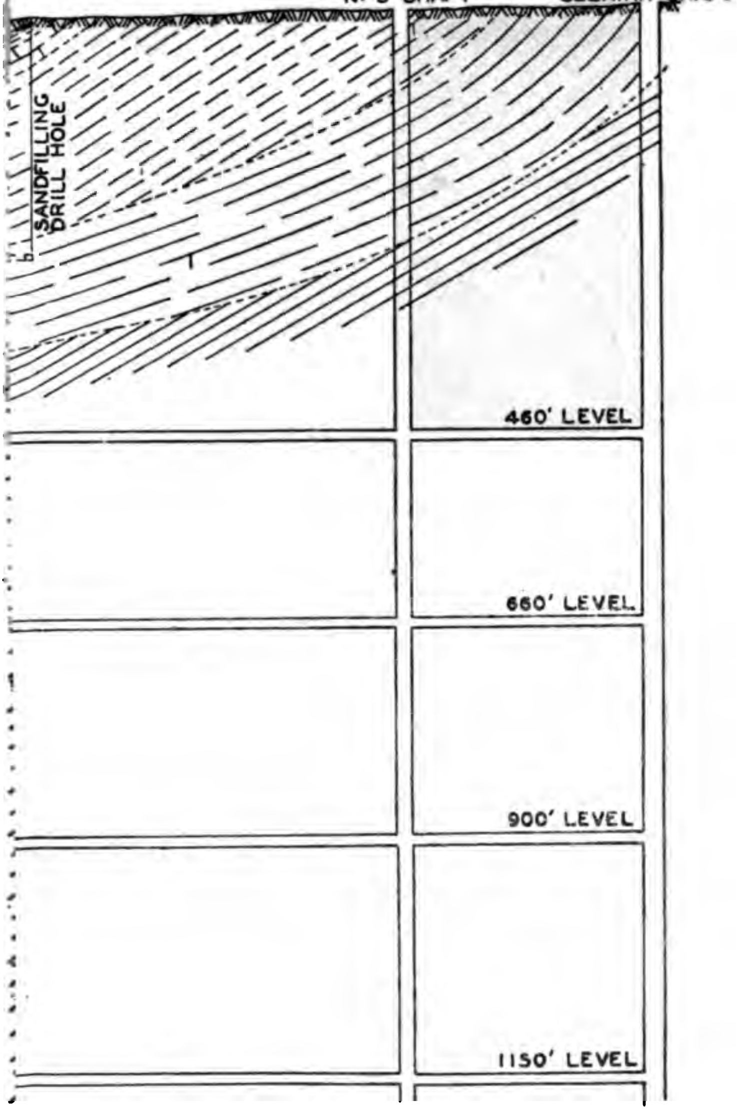
Year ending	*Ore Removed*		*Sandfilled*	
	Tons Milled	*Cubic Feet*	*Tons*	*Cubic Feet*
Start of Operations to June, 1937	2,481,007	32,997,393	2,300	46,000
To June, 1938	1,598,886	21,265,184	299,500	5,990,000
,, ,, 1939	1,782,119	23,702,183	816,649	16,332,980
,, ,, 1940	2,192,340	29,158,122	721,044	14,420,880
,, ,, 1941	2,411,303	32,070,330	801,002	16,020,040
,, ,, 1942	2,517,611	33,484,226	1,159,806	23,196,120
,, ,, 1943	2,594,131	34,501,942	908,656	18,173,120
,, date—March, 1944	1,994,972	26,533,128	983,783	19,675,660
Totals	17,572,369	233,712,508	5,692,740	113,854,800

*** *Extra copies of this paper may be obtained at a cost of 1s. 9d. each, at the office of the Institution, Salisbury House, Finsbury Circus, London, E.C. 2.*

CROWTHER & GOODMAN (*Branch of The Argus Press Ltd.*), 2, Carmelite St., London, E.C.4.

Plate I.

II.


Nº 5 SHAFT
SELKIRK SHAFT
SANDFILLING DRILL HOLE
460' LEVEL
660' LEVEL
900' LEVEL
1150' LEVEL

CE

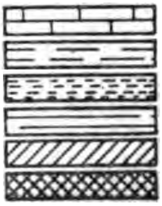


FIG. 14.—A battery of four desliming tanks.

FIG. 15.—A battery of eight steel desliming cones.

FIG. 16—Distributor and launders feeding desliming tanks.

Fig. 17.—Column of sand discharging from ore gate of desliming tank.

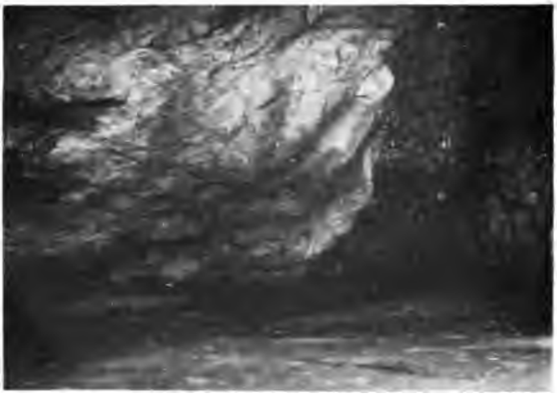


Fig. 18.—View showing back of stope and surface of sand.

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FIFTH ORDINARY GENERAL MEETING

OF THE

FIFTY-FIFTH SESSION

Held in the Rooms of the Geological Society, Burlington House, Piccadilly, W. 1,

ON

Thursday, February 21st, 1946.

Mr. G. F. Laycock, *President*, in the Chair.

DISCUSSION

ON

The San Telmo Ore-Body, Spain.

By J. C. Allan, *Member*.

The President announced that the paper submitted for discussion that evening was on the San Telmo ore-body by J. C. Allan.

This paper covered certain aspects of a subject which had been discussed many times before, but which was always of interest to the mining geologist and mining engineer—namely, the genesis of the large pyritic ore-bodies which are found in the province of Huelva, Spain. In this case, however, the author referred more particularly to the complex ore which occurred in the Santa Barbara mass of the San Telmo group. He had never visited that part of Spain and was unable, therefore, to offer any constructive criticism or comments on the subject, but he was interested to find that the author made several references to the complex ore found at Buchans mine, Newfoundland, with which he was very familiar indeed. The author drew attention to various similarities between the two ores and the speaker had thought it might be of interest to members if he put a few specimens of the Buchans ore on the table for inspection. He could not see himself that the two ores were very similar, the interesting point being that they both contained a high percentage of barytes.

Some of the specimens of the Buchans ore might show evidence of the structures referred to in the paper, but he would prefer to leave it to the geologists to express an opinion on that. Some comments from the staff at Buchans upon the paper would be read later on in the discussion. Unfortunately the author was abroad and unable to introduce the paper in person, but Dr. David Williams, who had an intimate knowledge of the district, had kindly undertaken the task.

Dr. David Williams expressed his regret that Professor W. R. Jones was prevented by indisposition from introducing the paper on behalf of the author. As a background to the subject under discussion the speaker sketched the general geology of the pyritic field of the Huelva Province, describing with the aid of a diagrammatic section across the area the location of the various mineralized zones and their spatial relations to the intrusive porphyries. He then summarized the paper, pointing out that in the author's opinion the sulphides had crystallized mainly from an intrusive hydro-pyritic matte to yield massive pyrite, complex ore, and cupreous pyrite as three successive phases. Colloidal solutions had played an important rôle in ore deposition, especially in the formation of the complex ore, and hydrothermal sulphide replacement had been effective locally and notably near the margins of the ore-body.

The speaker concluded his introduction by reading the following memorandum submitted by the author :

'In putting forward this paper I am very conscious of the fact that the information given falls very far short of the complete data I would have liked to present. When the mine was being worked for complex ore the idea of this paper had not occurred to me and the details presented represent what I have been able to extract from the company's records. I have ventured to put forward such details as are available in this form so that they may be permanently on record, as I hope that even in this limited form they are of interest.

'The association of barytes-bearing fine-grained zinc mineral with bodies of massive pyrite appears to be too widespread not to have some significance. Apart from the examples in the Province of Huelva, the following may be mentioned : (1) Buchans River and the pyritic bodies of Tilt Cove and Little Bay, Newfoundland ;

(2) the Aldermac Moulton Hill deposit and the Eustace mine, Quebec ; and (3) the complex ore at the Parys mine, Anglesey.

'Hydrothermal replacement as I understand it is the molecular replacement of the original rock by mineral-bearing solutions which are, in turn, conceived as dissolving the original for the deposited mineral. I cannot see why under the wide range of conditions that must have obtained such mineral-bearing solutions should be assumed to have a comparatively limited range of concentration.

'I suggest that little is known of the conditions under which such solutions can be formed, so that the degree of concentration must be largely hypothesis. The conception of a hydro-pyritic matte is after all analogous to a highly concentrated solution and it would seem to me that much of the argument can be boiled down to a question of the degree of concentration of the solutions at any one point at any one time. Nature does not work to precise mathematical rules and in the range from hydro-pyritic mattes to hydrothermal replacement as normally understood there is ample room for intermediate effects which mask the criteria conventionally applied to either theory'.

The President then invited **Dr. Williams** to open the discussion on the paper, and Dr. Williams continued by confessing that he had found the paper rather unbalanced, since too much theory had been built on an inadequate foundation of fact. Nevertheless the paper was valuable in drawing attention to a barytes-rich complex ore which was not common in the pyritic deposits of Spain. The account of the geology of the ore-body and of the mineralization was, however, so meagre that to discuss it was almost analogous to tilting at windmills. The description and cross-sections did not make it clear whether the ore-body dipped northwards or southwards, whether it was roughly conformable to the flow-cleavage of the slates or not, or whether the adjacent porphyry was wholly sheared or was massive and ramified by stockwork veinlets of sulphide. It was stated that the apex of the ore-body 'showed the characteristic secondary enrichment of copper values, although, owing to the fact that it did not outcrop, no leaching or erosion of the solid sulphide appears to have taken place'. The term secondary copper enrichment usually implied the deposition of supergene copper sulphides—such as sooty black chalcocite and blue covellite—from downward-percolating sulphate solutions derived by oxidation of the primary sulphides. As there was no mention of chalcocite and covellite in the paper it seemed that the author was misusing the term secondary enrichment by embracing within it the introduction of chalcopyrite deposited at a comparatively late stage of the primary mineralization. True secondary enrichment necessitated some leaching of the primary ore and this leaching could take place anywhere within the zone of oxidation, which might be hundreds of feet in depth, even though the ore-body failed to outcrop.

In the absence of any direct clues given in the context the speaker enquired whether the irregular cross-sections of the ore-body on the lower levels were related to irregularities in the contour of the foot-wall porphyry. He also asked whether the 'horses' within the sulphide mass consisted of porphyry or slate, and whether any of them had been proved unequivocally to be wholly enclosed in the ore-body without connection with the wall-rocks.

Although no instances of zinc, lead, or barytes disseminations had been recognized at San Telmo, 'such as would point to hydrothermal replacement', at the Rio Tinto mines disseminated barytes of hydrothermal origin was quite common in the hanging-wall slates, in horses within the massive ore, and in the porphyry foot-wall, while grains of blende and galena were locally scattered abundantly through the porphyry in the vicinity of stockwork veinlets.

The author might have exaggerated the degree of dependence placed by protagonists of the replacement hypothesis upon banding in ore as a palimpsest structure inherited from replaced rock. They were fully aware that banding could arise in other ways—such as by colloidal deposition, by parallel encrustations in open veins, and by the plastic deformation of sulphides. An obvious contribution to the solution of the problem of ore deposition lay in reasoning back from visible results, and on this score there could be no doubt in very many cases that banded ore was due to the replacement of slates and fissile porphyry. A suite of specimens, mostly collected by the late H. F. Collins, was exhibited to demonstrate various stages in the passage from slate to banded ore. At Rio Tinto it was sometimes possible to trace banded ore into unreplaced horses of slate whose cleavage was parallel to banding in the adjacent sulphide, irrespective of irregularities along the margins of the horses. In addition to such clear evidence of replacement it was often possible to demonstrate passages from massive porphyry into solid ore by increase in sulphide content, so that the limit of massive ore was merely an assay boundary.

The lenticular form of many of the pyritic bodies in the Province of Huelva had been attributed to the infilling of openings formed by *orogenic pressures*, to the distension of the walls by the pressure of mineralizing solutions, to the force of crystallization of the sulphides, and other causes. It was possible, however, that the shape of some ore-bodies had been influenced by post-mineralization plastic deformation. In many areas—such as at Rio Tinto, Esperanza, and Monte Romero—the ore-bodies had been partly deformed by post-mineralization pressures, the early pyrite was often brecciated, and the chalcopyrite frequently exhibited polysynthetic pressure-twinning. Under the influence of pressure the sulphides—especially those of lead, zinc, and copper—would tend to suffer plastic flow. This might result in the development of banding and elongation of the sulphide grains, although complete recrystallization would obliterate such textural evidence of deformation. The speaker considered that if the ore-bodies had been subjected to strong pressure they could undergo what he might call 'metamorphic differentiation', so that the later sulphides would flow and possibly exhibit cross-cutting relationships towards the other sulphides in the manner of intrusive veins.

The wall-rocks of the ore-bodies commonly contained innumerable cubes of pyrite surrounded by eye-shaped lenticular areas, known as 'pressure-shadows', mainly infilled with feather quartz. Although many of these lenses had probably been opened by the force of crystallization of the growing cubes most of them appeared to be due to the linear stretching of the porphyry or slate, so that the matrix was dragged away from the faces of the pyrite crystals to form potential openings that were filled continuously with quartz. It was significant that the orientation of these pressure-shadows was parallel to that of the nearby ore-body. By analogy with the microscopic cubes of pyrite and their enveloping haloes it was possible to visualize an early resistant body of massive pyrite being subjected to continuing earth-pressure so that potential cavities formed at its apex and extremities, towards which the later mineralizing solutions bearing copper, lead, and zinc would tend to flow.

Professor G. V. Douglas had recently offered an explanation as to why some of the Huelva ore-bodies were richer in primary copper towards their apices.* The mechanism envisaged was a softening of the slates by hydrothermal solutions rising through the stockwork porphyry towards lens-shaped cavities opened up along the cleavage of the slates or along the porphyry-slate contacts. On reaching the wider cross-sections of the lenses the upward current of mineralizing solutions would be slackened and the early-

*'The Hydrodynamical Factor in Ore Deposition'. *American Journal of Science*, Vol. 243–A, *Daly Volume*, 1945, pp. 122–134.

crystallizing pyrite would tend to settle towards the bottom of the lenses, the subsequent precipitation of copper, lead, and zinc sulphides taking place in the upper parts of the cavities.

The author seemed to base his support for colloidal ancestry of the complex ore upon its frequent banded structure and the presence of abundant barytes, which, at Buchans mine in Newfoundland, is definitely associated with colloform structures. Although the speaker admitted that the barytes might have been precipitated from a dispersed sol, he saw no adequate reason to invoke the aid of this *deus ex machina*. Colloform structures could also be developed by the precipitation of material as a colloid at the site of deposition, even though it had previously been transported in true solution. The available experimental evidence suggested that hydrothermal alkali sulphide solutions were incapable of carrying large quantities of metal and that iron in particular was not readily soluble in them. For this reason several American writers had advocated the transport of sulphides in a dispersed solid phase, deposition being effected by dispersal of the peptizing agent (possibly H_2S), by coagulation by alumina, or some electrolyte. The speaker was certainly prepared to accept the possibility of such a means of ore deposition, especially if it could be supported by factual evidence.

Barytes-bearing sulphide ore somewhat similar to that of the complex ore at San Telmo had recently been described from the Twin J mine, Vancouver Island, where it had been attributed to the replacement of folded cherty tuffs.* Although the speaker had examined the deposits at Tilt Cove and Little Bay in Newfoundland he had not observed much barytes in them, and he would therefore welcome any further information about the nature of the barytes occurrences at these localities.

Mr. P. F. Whelan congratulated the author on a paper packed with informative detail and with closely reasoned arguments. It was in no spirit of criticism that he wished to point out one aspect in which the presentation might be improved. The new observation that in the Santa Barbara mass all the banding occurred within the mass and could not be considered as pseudomorphic after replaced slate was of such cardinal importance that an endeavour should be made to interpret it in a manner beyond the reach of controversy. On p. 13 the author wrote: ‘Banding would be more likely to have been produced under conditions analogous to a colloidal sol. . . . Thus a colloidal ancestry for the complex ore

*STEVENSON, J. S. ‘Geology of the Twin J Mine’. *Can. Min. Met. Bull.*, May, 1945.

might be deduced from the banded structure, which is a phenomenon known to be associated with colloids'. This argument carried great weight in 1925 when Edge put forward the Liesegang phenomenon as an explanation of regular banding in the pyritic ore-bodies, but there had now to be taken into account a vast amount of research into periodic precipitation since that time and the fact had to be faced that present-day views on rhythmic phenomena ran in some degree counter to the conceptions current in 1925, and repeated in the paper. On the whole, it seemed that the case for explaining these regularities in the ore in terms of periodic precipitation had been strengthened rather than weakened by the new information accumulated with the passage of the years, but it was all the more essential to present the case so that it would escape criticism from modern workers in the field.

Periodic precipitates were usually prepared by allowing a gel—such as gelatine or silicic acid—containing a dilute solution of one reactant to set in the lower part of a test-tube and then pouring a concentrated solution of the other reactant on top. After some time, a series of bands of precipitate formed within the gel, and were called 'Liesegang's rings' after a prolific worker in this field. Between 1890 and 1925 numerous examples of rhythmic structures were prepared by this technique—invariably, however, in a colloidal medium. In 1926, for the first time, a periodic precipitate was prepared in the absence of colloidal material, when Ostwald produced rings of calcium hydrophosphate by interaction of disodium hydrophosphate and calcium chloride in capillary tubes. In the same year Fischer and Schmidt reported rhythmic bands of calcium hydroxide formed in a non-colloidal medium by a slightly different technique.

Following these Continental discoveries Doyle and Ryan in these islands carried out a study of many periodic reactions and published their principal results in 1929. The preparations made by these workers were the first examples of periodic precipitation the speaker had had the opportunity to examine and even to-day, after 18 years, he could vividly recall a small research laboratory where racks of glass tubes containing regular rings of chromates, iodides, and other precipitates made a brilliant and unusual display. Amid this array of colour, however, the real centre of interest lay in some apparently insignificant assemblies of narrow-bore tubing containing calcium chloride solution, so suspended that the open ends of the capillaries dipped into a bath of saturated disodium hydrophosphate, the whole arrangement being carefully maintained at a constant temperature. In these narrow tubes, con-submission of some remarks which are the result of a discussion with our geological staff, in charge of Dr. H. J. MacLean.

'Since the submission of Dr. W. H. Newhouse's paper* a large amount of structural geological study has been completed, mainly in the hope of locating extensions to the known ore-bodies or favourable geological areas which may lead us to new ore-bodies. However, in the mining of the ore-bodies and in the structural research certain theories of genesis have been developed which are not in accordance with the paper by Dr. Newhouse or the paper by Mr. P. W. George.†

'At Buchans the country rock is a series of Ordovician lava flows, agglomerates, and tuffs, varying in composition from quartz-latite to basalt. There are no immediate intrusive bodies other than diabase dykes. Dr. Newhouse refers to rhyolite-porphyry intrusive sheets in the Buchans mine vicinity ; however, further underground development work has shown these sheets to be acid flows and agglomerates. Granodiorite and diorite intrusive masses border the mine on the north ; the nearest outcrops are four miles from the mine workings. Other than the usual break-down products there is no evidence in the country rock of metamorphic recrystallization. Quartz or carbonate veins in the country rock surrounding the mine are very rare.

'The ore solutions are believed to have risen to the favourable horizon through an undiscovered fault or faults and to have advanced up the dip to the location of a favourable sheared and altered zone ; there the hydrothermal solutions replaced the sheared volcanic in scattered centres of crystal deposition, each individual lens enlarging until some of them united to form major ore sheets. The largest single ore sheet has a length down the dip of 1,200 ft., a maximum width of 500 ft., and a maximum thickness of 175 ft. ; this lens is composed of fine-grained sphalerite-galena-barite, with some chalcopyrite-pyrite bands near the foot-wall ; narrower bands and veinlets of chalcopyrite occur irregularly through the ore lens.

'The barite gangue varies from about 15 to 20 per cent in the high-grade ore lenses to about 80 to 90 per cent in the low-grade baritic ore lenses. The ore and barite content of individual lenses is fairly constant throughout the lens, although there are almost barren barite layers present on the hanging-walls or foot-walls of some of the lenses.

* *Econ. Geol.*, Vol. 26, 1931.

†A.I.M.E. Tech. Pub. 816, 1937.

'Most of the ore-bodies are massive sulphide and barite with sharp boundaries, bordered by schist; however, a considerable number of the lenses, especially in the eastern mine, have disseminated boundaries between the high-grade centres and the altered surrounding dacite. In the lenses there are gradual variations from fine to coarse grained, from sphalerite-galena to chalcopyrite, and from high grade to medium grade, but there are no sharp inter-lens boundaries to suggest fissure-infilling of injected ore masses cutting the rest of the ore.

'There is considerable hydrothermal alteration of the enclosing country rock, with the development of several feet or more of chlorite or sericite schist at the ore contacts. So far to date no other type of replacement except "hydrothermal" has been recognized.'

Mr. J. B. Richardson said he would like to express his hearty appreciation of Mr. Whelan's constructive contribution on rhythmic precipitation, with its bearing on the formation of banded ore.

At the Santa Barbara mine, on the upper levels and at the same horizon, it was possible to see a clear-cut contact between cupreous pyrites and complex ore and not many metres away a gradual merging of complex ore into cupreous pyrites, frequently with well-developed banding. This puzzling phenomenon gave rise to much local discussion. In this connection the specimen exhibited on the table, labelled San Telmo, was not as typical of the banded ore found there as the sample from Collins's collection labelled Monte Romero.

He would like to amplify the author's description of the San Telmo complex ore which formed such an important feature of the paper. It had no visible crystalline structure; the texture of the blocks was like a concrete made of equal parts of Portland cement and very fine sand and the usual colour was dark blueish grey. Even when it had remained on the surface for years it did not soften or break up, but was as hard as freshly-mined ore, and only differed in appearance by surface staining—usually streaks of a dark chocolate colour. Complex ore left on the dumps by the Spanish owners many years before was collected and used as mill feed. In spite of much discussion in the mill it was never definitely proved to be inferior to freshly-mined complex ore as feed for the flotation plant. The author did not mention that this ore was successfully treated in the mill from 1927 up to the time of the great slump in metal prices, to produce a large tonnage of zinc, lead, and copper concentrates of saleable grades. At present prices it would be again an economic proposition.

The milling problem was interesting and intricate, as the author hinted, owing to the intimate intergrowth of the constituent minerals. With the highest magnification of the mill microscope it was frequently impossible to distinguish the separate sulphides and only mixed particles of sphalerite and pyrite could be seen. At the outset all the ore was ground to *minus* 200-mesh and at one time, when recoveries were lower than usual, it was suggested that it be ground to *minus* 1,000-mesh to free the separate minerals.

As the Collins collection of specimens of ore exhibited on the table showed, complex ore of a similar type was known to exist on a number of other properties and it was understood at the time that parcels had been sold abroad, but San Telmo was the first company to treat the material in Spain, because of the much larger tonnage of zinc-bearing complex ore found there.

The question much discussed at the time the mill was operating was that if the Santa Barbara mass was from the same parent magma and of the same age as other ore-bodies in the district, some of which had also not reached the surface, then was not the much greater tonnage of zinc and lead-bearing complex ore an indication of an immense mass of pyrite below ? He would like to hear the opinion of geologist members, especially those acquainted with the southern Iberian pyritic ore-bodies, on this point.

From the paper itself and the discussion it would appear that there was wide divergence of opinion on the theory of origin of these ore-bodies. Farther afield, in America, one read there was a school of thought that opposed the assumption that ore-bodies are necessarily related genetically to the nearest igneous intrusion and doubted whether ore-bodies derived from a magma at all.

The author pointed out that at San Telmo there were no known instances of dissemination of zinc, lead, or barytes in the country rock near the Santa Barbara mass, but he understood from Dr. David Williams's spoken contribution that disseminations of zinc and barium minerals were found in the porphyry walls at Rio Tinto. Naturally, if further development was contemplated at San Telmo the nature, origin, structure, and order of deposition of the different types of ore should be taken into account. He did not know if the upper levels at San Telmo were still accessible, but if so it would seem a good idea for a competent economic geologist to investigate further the points under dispute, to collect factual evidence, and bring it for discussion to a meeting of the Institution, in order to see if it shed fresh light on the relation between neighbouring, better-known ore-bodies, and that one which appeared to be unique among Southern Iberian pyritic masses. The main object in so doing would be, of course, to guide local mining engineers in the search for new ore, the proper function of mining geologists.

Mr. W. W. Varvill said that he had recently returned from Germany, where he spent three weeks examining lead-zinc mines, during a fortnight of which he was at the Rammelsberg mine mentioned in the paper, and he felt he might be able to contribute something to the discussion, although he was not a geologist. The Rammelsberg mine had been the happy hunting-ground of German geologists for many generations and the problems as to how the ore got into the place where it was were still unsolved. The mine claimed an uninterrupted record of work since the year 968, and Agricola himself in 1550 gave an account of it in his works. The Germans said that every two geologists who examined the mine went away and produced three entirely new theories and even he, a layman, felt it his duty to produce a new one. The Germans were too polite to make any comment on this, but the author of the paper must not allow the long drawn-out controversy about Rammelsberg to dishearten him in the search for truth about San Telmo, because the party to Germany included two geologists of wide experience who were now engaged in producing a report which would be published in due course and which would prove of interest to the people interested in these geological problems.

The San Telmo ore-body showed considerable similarity to Rammelsberg, although the latter was much richer. The Rammelsberg mine contained 10 per cent lead, 19 per cent zinc, 1 per cent copper, 8 per cent iron, 22 per cent sulphur, 25 per cent barium sulphate, 8 per cent silica, 4 per cent lime and alumina, and 160 grammes silver and 0·8 grammes per ton of gold. In view of the fact that the mine was equipped to recover everything of value, including barytes, not very much was wasted. He could confirm that the banding of the lode was parallel to the walls, but he was not able to observe whether the stratification of the country rock was also parallel. According to the geological sections there was much folding and faulting and the slates were shown parallel in some parts, in others they were not.

Rammelsberg appeared to differ from the description on p. 10 given by the author. There did appear to be a large deposit which seemed to conform to the planes of the country rock, which the Germans called *Kniest.* They described it as siliceous rock, containing 1·3 per cent copper, 1·4 per cent lead, and 3 per cent zinc, which would make it very nearly payable ore. It was 80 metres thick and 30 metres from the lode, but was not worked; there was no reason why it should be, since they had about six million tons of the other high-grade ore, which would last for 80 years. He did not think their British geologists had found any signs of metasomatism in the ore-body, and there seemed to be absolutely clear-cut lines of demarcation between the lode and the country on either side. Another thing which the two ore-bodies had in common was that at Rammelsberg they had two ore-bodies, just as there were at San Telmo, with much the same strike and dip but offset *en echelon* to one another. At Rammelsberg the two ore-bodies were connected by a large mass, termed the 'Grey ore-body', which consisted mostly of barytes. Owing to the existence of a loop haulage road, which was driven round the end of the ore-body, the party was able to examine the line of strike when it extended into the country, but could not see any pronounced fissure. The Germans told them that drill-holes underneath the bottom limit had failed to disclose anything at all, so it looked as if there were no fissure coming from the bottom.

Nevertheless this did not rule out possibilities of the ore solutions having come from below, because there did not seem to have been much development or exploration by drives and cross-cuts at depth, and diamond drill-holes could easily have missed any 'vent'. The speaker had had experience in several mines where quite important ore-bodies had apparently been formed by solutions which ascended through quite small vents.

Mr. R. H. Craven said that Mr. Allan's paper was an interesting study. He had endeavoured to digest it very carefully, as it formed an important addition to the geological research work carried out with a view to explaining the mineral formations around the Mediterranean basin. To be strictly accurate the Huelva group of mineralized deposits were not actually on the Mediterranean coast; however, their similarity to those eastwards were very marked.

The limestone rock of Gibraltar was not really a geological division of the formation of the shores of the inland sea and the Atlantic; it must be looked upon rather as a continuation of the limestone formation found in the Mediterranean, where it was found in the promontory of Portofino; also practically the whole of the railway tunnels on the Italian Riviera were driven through the limestone, though there were a few exceptions due to intrusive rocks. In certain neighbourhoods sandstone was found on the coast, as at Sestri Levante, where it was capped by a slate formation from which Lavagna and Cavi took their names. Further inland the similarity no longer continued, and there were many differences.

Having made only short visits to the Spanish and Portuguese deposits he had had little opportunity of studying the geological formation on the spot. Mr. Allan gave the information that the ore-bodies occurred on the porphyry-slate contact ; the ore deposits further east were chiefly near a contact between serpentine and diabase, the former being the dominating rock on the surface. However, an important point was that both porphyry and slate did sometimes exist in small veins near to and even in contact with the masses of mineral, but what he wished to arrive at was the way in which the hydrothermal formation of copper and iron sulphide had been confirmed by studies carried out both on the surface and in development underground.

During some 40 years of intimate connection with the Libiola mines (these were worked by British capital from the year 1868 until tragic events brought about a break) so much development work was carried out that similar conclusions emerged as those put forward in Mr. Allan's paper—namely, that the sulphide solutions found their way under pressure into crevices, formed at the time or previously, and there crystallized. The reason which brought about this train of thought in connection with the eastern formation was that, whilst the masses of ore were found more or less near the contact of serpentine and diabase, the copper sulphides and iron pyrites were never found impregnated in the latter hard compact rock, whilst in the former, being highly fissured, there were small veins of both sulphides which extended over a large area.

Another important conclusion come to was that the two mineralized solutions were deposited and crystallized at different periods. The small mineralized veins formed in the serpentine rock were either copper sulphide or clean iron pyrites, free from impurities such as zinc, lead, arsenic, etc., though there were traces of gold and silver in both cases. The two sulphides never appeared to intermingle in the same fissures in the serpentine formation.

Further confirmation was given to the theory that the solution bearing the copper followed on to that containing the iron pyrites by the fact that copper sulphides were found embodied in the masses of iron pyrites in crystal form. In fact, separation of the two minerals was very clearly defined as a rule ; by hand picking alone clean copper sulphide was produced, leaving only 0·5 per cent or less of copper in the retaining iron pyrites.

Dr. S. W. Smith said that mention had been made both in the paper and in the discussion of the many difficult and puzzling questions which arose in any attempt to account for the origin and formation of the pyritic ore-bodies of the Huelva Province, of which the San Telmo ore-body was a particularly interesting example. Among those difficulties was that of accounting for the way in which many of those bodies were said to terminate or 'cut out' abruptly in depth. Mr. Varvill had quoted an example from another field—that of the famous Rammelsberg mass—and Dr. Williams had shown on the screen the hemispherical form of the ore-body which had occupied the Salomón open-cast at Rio Tinto. Other examples could be cited, but one of the most remarkable, as it appeared to him, was that described by Collins* as occurring at the abandoned mine of Confesionarios, in the Val de la Musa, not very far away from the San Telmo body. According to his description, that ore-body—consisting of very pure pyrites almost free from copper—was shaped like a dumb-bell in a vertical position, the lower end, however, being oval in section instead of circular. No ore channel was found beneath, although the ore-body was underlain by 'very poor azufron'. Exploration all round its lower half by a ring drive and by inclined rises put up from below until they joined it, failed to disclose a continuation of the ore-body in any lateral direction. The inference which had been drawn, therefore, was that this dumb-bell-shaped mass was entirely enclosed within schist walls, although in fairly close proximity to porphyry dykes to the north and to the south of it—at one point on the north side within 50 metres or less of the ore-body.

The speaker's object in drawing attention to this particular occurrence, differing from the more usual lenticular forms of the ore-bodies of this district, was to point to an analogy between this dumb-bell shaped mass and that assumed by a column of viscous and immiscible liquid when intruded from below into surroundings sufficiently mobile to allow it to acquire, under the influence of surface tension, a form or shape consistent with that of minimum surface area—namely, that of a sphere. Such behaviour could be readily shown experimentally, since a cylindrical column of liquid was always unstable and tended to split up into globular masses.

The dumb-bell form of the Confesionarios ore-body suggested that some such process had operated but had been arrested at an intermediate stage and that the mass had been trapped by con-

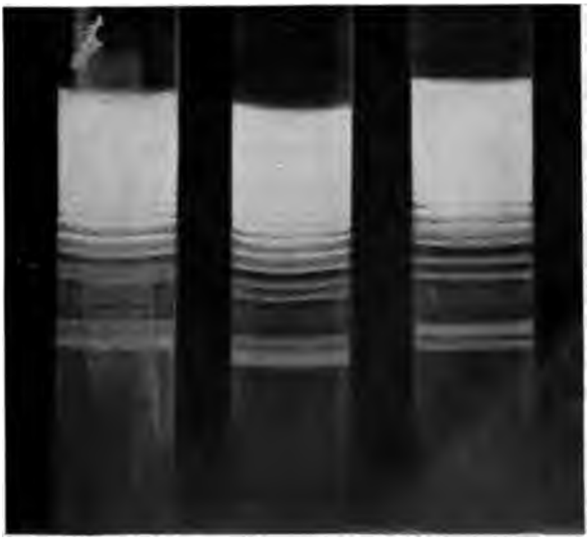
Trans. I.M.M., Vol. XXXI, 1922, p. 101, Fig. 14.

solidation while in the act of sub-dividing into two spheres. In the absence of any other adequate explanation, he had always felt that this unusual occurrence was a 'free gift' to those who favoured the magmatic injection theory of the origin of these pyritic bodies and might be regarded as a case where the more common lenticular form had not been assumed. It was conceivable that under lateral pressure this particular ore-body might have been resolved into two lenticular masses, *en echelon*, an occurrence not uncommon in this field.

Owing to the lateness of the hour **the President** asked other members who wished to contribute to the discussion to send in their remarks in writing. Although perhaps no satisfactory conclusions had been arrived at as regards the mode of origin of these ore-bodies, the discussion had been most interesting and he was sure it would be the wish of the meeting to pass a hearty vote of thanks to Mr. Allan for his paper. This was carried by acclamation.

(1-in. bore test-tubes)

FIG. 6.—Bands of magnesium hydroxide in 5 per cent gelatine gel.



(1 in. bore test-tubes)

FIG. 7.—*Bands of calcium* hydrophosphate in 5 per cent gelatine gel.

(0·5 mm. bore capillary tubing)

FIG. 8.—Bands of calcium hydrophosphate in water.

THE INSTITUTION OF MINING AND METALLURGY.

SIXTH ORDINARY GENERAL MEETING

OF THE

FIFTY-FIFTH SESSION.

Held in the Rooms of the Geological Society, Burlington House, Piccadilly, W.1,

ON

Thursday, March 21st, 1946.

Mr. G. F. Laycock, *President*, in the Chair.

DISCUSSION

ON

A Study of Sizing Analysis by Sieving

AND

A Comparison of Methods of Measuring Microscopical Particles.

By Harold Heywood, Ph.D., M.Sc., A.C.G.I., M.I. Mech.E.

AND

Application of Sizing Analysis to Mill Practice.

By Harold Heywood and E. J. Pryor, *Member*.

Dr. Heywood, in introducing the two first papers, said that developments in methods of measuring particle size had progressed rapidly in recent years, many complex physical phenomena being involved in efforts to determine the size of minute bodies. This rapid advance might be a cause of some bewilderment to those concerned in the practical applications of powdered materials and it would seem advisable to establish as early as possible the basic principles of well-known methods of particle-size measurement and to standardize these as far as possible. The papers on sieving analysis and on microscopical measurement of particles had been presented to the Institution with this object and he desired to express his appreciation of the privilege of being able to discuss the subject before an Institution that had already done so much to promote the study of particle-size measurement as applied to mining.

The two main factors involved in sieving analyses were the standardization of the sieve cloth and of the method of conducting the analysis. The Institution of Mining and Metallurgy took a leading part by standardizing a sieve series in 1907 and was followed by other countries; the British Standards Institution published a standard sieve series in 1931 and a revised specification in 1943. The tolerances in sieve specifications were now reduced to the minimum commensurate with the practical accuracy of weaving and further improvements in the technique of sieving could only be attained by standardization of the method of conducting the analysis. Since sieving was a statistical process and there was always an element of chance whether a 'near mesh' particle would or would not pass the apertures the 'end point' could only be defined by the adoption of a convention—such as a specified time or rate of sieving. The element of chance was increased with irregularly-shaped particles, for flakes would pass diagonally through a square aperture if favourably presented. Thus the apparent size under the microscope of particles that just passed a given size of aperture varied according to the shape, as shown by lantern slides of 10-mesh particles which were projected. Consideration must also be given to the different characteristics of wet and dry sieving; data had been given in the paper showing the extent to which fine dust adhered to the coarser particles even after prolonged dry sieving and photographs taken subsequently showed this effect very clearly.

In spite of all care in the manufacture of sieve cloth and in standardizing the method of sieving, there might still be a discrepancy between sieving analyses made on the same material with different sieves having the same nominal apertures. This might have important consequences in research work or in the purchase of powdered materials to close specification and the disparity could only be avoided by a comparative sieving test with the sieves concerned. An instance of this arose in a co-operative research on coal grinding and in order to correlate the results obtained at seven different laboratories a system of aperture corrections was devised. He had endeavoured to describe clearly in the paper the method of determining these corrections and time did not permit further description, but the essential principle was shown by Fig. 5. For a particular aperture error between two sieves the residue or weight correction would vary according to the fineness of the powder concerned and thus it was preferable to apply aperture corrections rather than residue corrections. Table VIII formed a summary of this section of the paper, the figures given being regarded as a measure of the relative error to be expected for different conditions of sieving.

To turn to the second paper: Microscopical measurement over a wide range of particle size was a laborious process and other methods of size determination should if possible be used. Nevertheless there were occasions—such as when only a minute sample of dust was available—when there was no alternative to its use. It should be emphasized that this paper dealt only with the various methods of assessing size and not with the manipulation of the optical components of the microscope. The graticule introduced by Patterson and Cawood was most frequently used for measurement by visual examination under the microscope, although modifications of the design had recently been proposed. It was perhaps more convenient to use a statistical diameter to represent the size of particle images enlarged and projected on a screen. Such statistical diameters had been defined by Green, Martin, and Feret and were illustrated in Fig. 8 of the paper. Statistical measurements could not be applied to individual particles and only had a real meaning when considered as the average of a large number of measurements.

In the research described comparative measurements were made on photographically-reproduced profiles and the results were summarized in Table III. The conclusions were that there was a tendency to over-estimate sizes with the graticule system of comparison, although for practised observers the error was small; that the average of Martin's statistical diameter was slightly less than the true projected size of the particles; and that the statistical diameter defined by Feret was considerably too high and was not recommended. The latter point was not merely of academic interest, for the graticule recently proposed by K. R. May involved the use of this method.

It was considered that the data presented in the papers showed that some measure of standardization might be attained in the size analysis of particles by sieving and by microscopical measurement. While rigorous standardization could not be applied in all cases it would seem feasible to issue guidance in such matters as weight of sample and time of sieving, the methods to be adopted for wet and dry sieving, and the correlation of analyses. For microscopical measurement the graticule comparison method or the average of statistical intercept diameters were equally within the required limits of practical accuracy. Standardization of these established methods of particle size measurement would leave the way open for an attack on the much more difficult problem of standardizing the methods of sub-sieve particle size analysis mentioned in the joint paper by Mr. Pryor and himself, the third paper under discussion.

Mr. E. J. Pryor introduced the third paper by suggesting that mineral dressing had for some decades been emerging from an art to a science. There was increasing predictability in its application, although they were not yet able to say without first testing an ore that a given method of treatment would succeed at one mine because the same ore had responded to this treatment elsewhere. When they were able to predict the behaviour of an ore after studying its chemical analysis and its structural interlock the 'art' of ore-dressing would have been replaced by its science.

In struggling toward this goal the behaviour of ore during crushing and fine grinding had received much attention, since it had shown itself to be a major factor in the treatment of ores by modern methods. No satisfactory index of grindability had thus far gained general acceptance, since small changes in the assay-values and in crystal interlock set up changes in grinding-response which were too variable to allow of simple tabulation. Technically, grinding was very important in connection with product control, whether that product was to be a selected fraction of an ore intended for sale to a smelter in a definite state of purity, or concerned primarily with the production of graded sizes of finished material. Economically grinding was important because it was an expensive process, which must be performed, but which must be carried through as cheaply as possible. Somewhere an optimum must be agreed between the technologist who insisted on adequate liberation of his values and the accountant who must watch the cost.

'Good grinding' was grinding which processed its raw material to an economic optimum—balancing cost, recovery, and value of product. It was the result of careful control in the mill of the right machinery by a reasonably-skilful operator, backed by first-class sampling and systematic use of the information given by sizing analyses made on the mill samples. In most mills this control was taken down to the *minus* 200-mesh point and left there. Increasingly during the past few years sub-sieve routine checking had been applied in concentrators dealing with complex problems, while on the research side of the industry one could say, with reference to particle size, that more and more attention was being paid to less and less.

The amount of surface exposed in an ore-pulp was a very important factor in its behaviour. They might consider, for example, its response to gravity when finely ground. The forces inducing solids to settle through water were gravitational; those opposing settlement were frictional. Hence, the greater the amount of total surface developed in unit volume of pulp the greater the total resistance to settlement. One practical application of this effect was in the preparation of heavy media for the sink-float process, where stability of the separating medium was affected by the relation between coarser and finer particles used. Others influenced the work done in ball-mills, classifiers, and pulp-transporting systems.

Again they might consider the question of grinding for flotation —a process believed to enter into some 90 per cent of present-day concentrating flow-sheets. It was customary to measure chemicals —such as, for example, the collector reagent—into the pulp with fair precision, so that some such quantity as 0·1 lb. of, say, xanthate, was present per ton of ground solid. This xanthate was carried in the form of molecules, of which a definite number were available for coating the selected mineral particles. By the nature of all chemical reaction there must be some sort of minimum strength below which reaction would become unsatisfactory. If they failed to control the mineral surface it was useless to add their 0·1 lb./ton and to expect consistent results. Table V showed how many molecules of xanthate were available per square millimicron

TABLE V

MOLECULES PER SQUARE MILLIMICRON OF SURFACE

Number available in pulp, calculated for 0·1 lb. of potassium ethyl xanthate per ton of banket ore at S.G. of 2·73. Formula:

$$\text{Molecules}/\mu\mu^2 = \frac{4{\cdot}196 \times 10^3}{\text{Area in sq. cm./gram. (Table I, col. 11).}}$$

Tyler Mesh	*Molecules* $/\mu\mu^2$	*Mesh or Microns*	*Molecules* $/\mu\mu^2$
10	105	200	5
14	74	270	3
20	52	400	2
28	37	30	2
35	26	20	1
48	19	10	0·8
65	13	5	0·4
100	9	1	0·2
150	7	0·5	0·04

Note.—The cross-section of xanthate molecules lies between 0·2 and 0·3 $\mu\mu^2$.

of ore surface. It ignored the question of percentage of selected mineral in the total solid and of dispersion of those molecules due to pulp dilution. Table V showed that while at 200-mesh five molecules, totalling an area of at least 1 $\mu\mu^2$, were available for each $\mu\mu^2$ of surface present, at a five-micron grind only half a molecule was available, to cover at most some 10 per cent of each $\mu\mu^2$. With a high-grade ore carrying easily-slimed sulphides such a pulp would be in danger of collector starvation. The calculation showed an important relationship between development of solid surface and chemical dosage.

In the technology of powder metallurgy the total surface affected the flow of material during hydraulic compaction. Other applications were connected with the study of silicosis, with explosive dusts, powdered fuels, and powdered catalysts. The paper sought to put on record some data on size and shape which might be of help in the study of particles as they affected the surface-physics and surface-chemistry of technical control in modern industry.

Mr. R. L. Brown* said that he was fortunate enough to have been associated with Dr. Heywood some years ago and had always appreciated Dr. Heywood's efforts to establish a firm basis for size determination. In work undertaken in coal breaking, involving detailed size analyses, there appeared to be a great need for standardization of the method of sampling and the method of carrying out sieving tests. A British Standard, No. 1293, had now appeared for large material and it was to be hoped that steps would be taken for extending this standardization to the finer sizes.

Mr. C. N. Davies† desired to raise a point in connection with the joint paper by Dr. Heywood and Mr. Pryor in which the waste of power due to overgrinding was discussed. He wondered if that was really related to the surface area of the unwanted fine particles; whether Rittinger's law was valid or not. It seemed to him that the efficiency of grinding was so very low that the establishment of a large tail of fine particles was not the important thing. Rather did it seem to be a function of the time for which it was necessary to operate the mill in order that the last big particle might be broken into small enough pieces to pass through the mesh. That obviously brought in not so much the fundamental question of surface activity as the question of statistics and probability in the design of the mill and he thought that the low efficiency of mills pointed in the direction of that being the important factor rather than the applicability of Rittinger's law, which no

*British Colliery Owners Research Association.

† *Industrial* Health Research Board, Medical Research Council.

doubt was valid for the shattering of individual particles, but ceased to be important when taking the operation of the mill as a whole.

They came up against the same problem in the atomization of liquids by means of sprays, because there it was necessary to impart a considerable amount of kinetic energy to the liquid, and the work done in operating the spray was dissipated, the active contribution to the surface tension or energy being quite a small proportion—1 or 2 per cent, or even less in some cases.

Dr. M. L. Smith* said that in his second paper Dr. Heywood had shown that observers consistently over-estimated particle size in visual comparison with opaque circles. As this was a matter of great significance in microscopic size analysis some tests had been carried out during the past few months, using test cards with particle profiles on the lines of Fig. 2 of the paper. These cards had been kindly provided by Dr. Heywood.

In applying the test the observer first sized about 70 profiles by eye-comparison, then was allowed to study the key, and then tried a second card with a further 70 profiles. The results were plotted as frequency curves and the 50 per cent point used for comparison as a measure of the personal error. The observers tested (14 in all) over-estimated the particle areas by 5 to 20 per cent, the experienced people having a tendency to the lower score. There were large differences between different observers, both in respect of their initial facility and the improvement after examination of the key, and this suggested that only a proportion of people would be found to be really satisfactory for counting tests. Use of such test cards was recommended both for training people in accurate size assessment in counting and for selection of people for that job.

In a written addition to his remarks Dr. Smith says : In the paper 'Sizing Analysis by Sieving' Dr. Heywood has abundantly emphasized the need for a fundamental approach to the problem. Although sieving is a common procedure both for routine laboratory testing and for pretreatment of samples for sub-sieve size analysis, insufficient care is usually taken in laying down the conditions ; standardisation of sieving procedures for the testing of the common powdered materials of industry is urgently needed. Such standardized procedures should be framed to suit the special particle properties of the materials and cover the details of the method, type of equipment used, and upkeep of sieves.

*Material Research Laboratory, Philip's Lamps, Ltd.

Dr. Heywood said, in reply, that he was glad to find that Dr. Brown supported him on this question of standardization and that he had found the method of correcting sieves of use in his researches. Such a method he thought would also be very useful in many other investigations. Mr. Davies had questioned Mr. Pryor's remark on grinding. He did not know whether Mr. Pryor would care to answer the point for himself, but there was another paper to be taken later in the same meeting in which Rittinger's law would come up for discussion.

He had been very much interested in Dr. Smith's use of the test profiles for training operators. In the paper he had not, of course, given any clue to the names of the observers who had kindly co-operated but he thought he might declare the name of the ' prize winner '. This was Mr. G. L. Fairs, of Imperial Chemical Industries, who had a great deal of experience in the subject, which showed that practice did make for perfection in this particular line of work. He himself was well down on the list. It was very curious how different people reacted to these tests ; if a wrong start were made the individual continued on that scale. Most of the observers overestimated quite considerably, but he might add that Mrs. Heywood, who had made very close estimates, had a considerable knowledge of drawing and judging perspective, as well as experience in the use of the microscope. Skill in drawing might be a useful trait in selecting people for this sort of work. On the whole, he thought that the recording of microscopical measurements should be restricted to a minimum, for it was no job for a human being at all.

Mr. E. J. Pryor said that he thought he would like to study the point raised by Mr. Davies before he replied, but his account had related entirely to closed-circuit work.

On the motion of **the President** a hearty vote of thanks was accorded to the authors of the three papers.

CONTRIBUTED REMARKS.

Dr. Marcus Fraser*: I have read through the three papers with interest and am glad to note the careful work which has been put into them. One of the most disturbing points about the use of sieves is the wide tolerance permitted in the aperture sizes and one feels inclined to wonder whether the acme of perfection has yet been reached in their construction.

*Director, British Pottery Research Association.

The pottery industry is mainly concerned with materials which have been reduced by nature to a fine state of division. The particle shape of these materials seems to depart widely from the shapes found by the author to be characteristic of many artificially-crushed materials. In particular one dimension is usually very much smaller than the other two ; in other words they are very thin plates. For these materials the attempt to deduce equivalent sieve sizes from sedimentation rates may lead to the formation of mental images which are far removed from reality. Apart from this special case, is it really wise to depart from the simplicity and convenience of the geometrical series for sieve sizes, which constitutes the great advantage of the B.S. and similar systems over earlier ones? Even as a mnemonic the suggestion in Table IV of the paper dealing with mill practice may have its pitfalls.

Dr. H. E. Rose* : These papers by Dr. Heywood are a valuable contribution to the literature on the subject of powder size measurement, since accurate experiment is the basis of advancement of all scientific knowledge. The paper on ' A Comparison of Methods of Measuring Microscopical Particles ' is particularly gratifying to the writer, because, as far as he knows, the first suggestion that biased errors occur in the microscopical estimation of powders was made in his M.Sc. Thesis and it was Dr. Heywood who suggested ' Powder Size Measurement ' as the subject for that research.

Two conclusions from the writer's preliminary investigations have been confirmed by Dr. Heywood : (*a*) A definite tendency to overestimate the size of the circle intended to match an irregular area, and (*b*) when transparent circles are used as the ' scale ' there is a definite tendency to increased error with increasing elongation ratio (the present writer did not study the errors involved in the matching of opaque circles).

These earlier experiments showed that the error varies with the triangle shape, decreasing in the order scalene, isosceles, equilateral —that is, with increasing symmetry. For the rectangles, all of which are equally symmetrical, the errors are much less than for the triangles and although there was a tendency for the error to increase with elongation ratio the correlation is not very marked for values of elongation ratio varying between 1 to 1 and 4 to 1. Perhaps Dr. Heywood could give an opinion as to whether the error is mainly dependent on the symmetry of the particle or upon the elongation ratio.

The experiments of Dr. Heywood, and those of the present

*Engineering Dept., King's College, London.

writer, relate to a special, and rather limited, aspect of the extremely-complicated problem of the errors in observations made under unnatural conditions and it must be admitted that the experiments of both workers are open to the criticism that although the results are intended to be applicable to observations made through a microscope one of the most disturbing factors—the hard black ring limiting the field of view of the instrument—has been eliminated from these tests. However, the present experiments are the first step in the more complete investigation and the above objection could be met by experiments using a series of fine powders under the microscope and comparison of visual estimation with the results of measurements of photomicrographs.

The theory of errors arising from disturbances of the observer by the instrument and conditions of observation are discussed by Whitehead* and he quotes an example which will repay repeating here, as it well illustrates the effects of these disturbing conditions. A range-finder was developed comprising a pair of binoculars into which are fitted sliding graticules bearing black triangular indexes. In use the graticules are adjusted until the triangles 'fuse' over the object being ranged and the range is then read from a drum attached to the screw adjusting the parallax of the indexes. The theory is quite sound, but in practice only about 5 per cent of the observers could use the instrument with any success. The reason is probably that a number of unnatural conditions caused any but the strongest mind to revolt and to cease to make an effort to cause the indexes to 'fuse', these disturbing conditions being the rings limiting the field of view: the 'mysterious' property of the indexes, which appeared as 'kites' which were capable of retaining the same apparent size as they approached to 1,000 yards or receded to 20 miles from the observer (this 'property' would require that the 'kites' had the highly unusual capacity of increasing in real size as they receded) and also the property of remaining intensely black as their apparent distance changed without showing the haze usually associated with increasing distance. In view of this example of the effects of conditions under which observation is made it is evident that the greatest care should be taken in comparing the results of tests made under different conditions.

The tests of Dr. Heywood relate to the characteristics of flat surfaces lying parallel to a plane, but the analysis of a real powder is a three-dimensional problem, so that further research into the

* WHITEHEAD, T. N. 'The Design and Use of Instruments and Accurate *Mechanisms*'.

problem of the orientation of irregular particles upon the microscope slide is necessary—for instance, flaky particles will have a high probability of settling flat on the slide, while well-rounded particles will have practically a random orientation. This problem would appear to be further complicated by the fact that the orientation will be related to the technique of mounting the powder.

The comparison of the errors involved in the assessment of the statistical diameters of Feret and Martin is extremely interesting, especially as, at first sight, the two methods of measurement would appear to be almost identical, and these results emphasize the need for mathematical analysis of any problem wherever possible.

DISCUSSION

ON

Crushing and Grinding Efficiencies.

By T. K. PRENTICE, *Member.*

The President said that the next paper, by Mr. Prentice, was being presented for simultaneous discussion by the Chemical, Metallurgical, and Mining Society of South Africa and the Institution. It was a most important paper, on a subject of great interest to all engineers, and he was sure that it would lead to a very fruitful discussion. The author, who was in South Africa, was unable to present his paper in person. In his absence Professor Truscott had kindly consented to introduce it.

Professor S. J. Truscott, in introducing the paper, said they would all regret that the author was not there himself to present it; he was on duty at Johannesburg, yet they must feel complimented that he should desire to have their opinion and their contribution in this important matter.

For his own part the pleasant task of opening the discussion gave him some concern on two grounds. He was concerned that he might fail to do justice to the author because his own activity in the subject belonged rather to the past, whereas the author was both active and eminent in the present. Then, years ago now,* he had committed himself to a view contrary to that the author was sustaining. Subsequently, it was true, a research student, under his supervision and at his suggestion, had carried out some systematic research on this subject, coming to the conclusion that Rittinger's law, which the author favoured, was more closely allied to the facts. He had communicated that conclusion to the author after a preliminary reading of the paper as it gave him, the speaker, the cover for any necessary retreat from his previous position.

In the paper now presented the author started with a Summary; it was a little unusual to start with a summary, but he believed it was a recommendation of the Chemical, Metallurgical, and

* 'Text-Book of Ore Dressing', 1923, pp. 182–194.

Mining Society of South Africa. In that summary the author remarked that evidence would be *submitted* supporting the increasingly growing belief that the energy absorbed in crushing and grinding was directly proportional to the new surface area produced in the material being comminuted. He stressed the word 'submitted' because it revealed the author's modest attitude throughout. Recognizing, however, that the fact that deformation of rock under compression up to the point of fracture was approximately proportional to the volume of the rock did not easily conform with the finding that the energy absorbed in achieving comminution was directly proportional to the new surface produced, all the data in connection with the experimental tests were submitted in some detail; the author stated, moreover, that he might have erred in some of his deductions. In advancing deductions of opposite tendency he, the speaker, desired to do so in the same tentative manner.

In the Introduction the author made it clear that the matter of the paper was that of mechanical efficiency, not the metallurgical. There was no other way of ascertaining the mechanical efficiency than by taking the power input and the mechanical value of the product. Estimates of this mechanical efficiency had ranged widely. Figures were given as low as less than 1 per cent and others up to 45 per cent. That disagreement was a challenge. First of all it was necessary to ascertain particle size. In Table I the author gave the theoretical surface area of a given weight of pulp, calculated from its screening analysis. He centered his considerations around the cubic particle, but the same results would have been obtained had he used the spherical particle. Then he went on to enumerate ways by which the surface area might be obtained directly—namely, by the dissolution method, the permeability method, and by others.

Table II gave a very useful comparison between the theoretical surface area and the surface area obtained in different ways directly. This comparison showed that the dissolution method using hydrofluoric acid gave the highest multiple of the theoretical surface. He then described the particular methods adopted in the Rand Mines laboratory, and their range of application.

Coming to the Theory of Crushing and Grinding, the author started with five ways in which rock might be broken by the application of force. The first of these was impact. He, the speaker, took it that impact was the most important application of force in comminution. Next came shear, which also was important. The third was bending. Bending, the speaker thought, was really not applicable to material, it was applicable rather to a structural member. What was important in bending was that tension came in. In the case of rock, the tensile strength being very low, the material gave way to tension. The next application, namely, compression, meant here the relatively slow application of pressure.

Then the author went on to speak about the useful work in crushing and he attributed the differences in efficiency estimates largely to the fact that those who had credited crushing with the lowest efficiency had counted the surface energy in the crushed mass alone as representing the work done. The author rightly insisted that not only did this surface energy count, but also the much larger amount of energy represented by deformation prior to fracture. Fracture followed only if the material had been strained to and beyond the limit, and the energy which went into that strain must be counted as part of the useful work done. Fracture was the end result.

A useful summary of Rittinger's law was then given—namely, that the energy required for crushing was proportional to the new surface formed; and of Kick's postulate that the energy required for producing analogous changes of configuration of geometrically similar bodies of equal technological state varied as the volumes or weights of those bodies.' Kick, so far as he, the speaker, knew, did not go as far as fracture. He kept within the elastic limit throughout. The first man to introduce the idea that Kick's law extended to fracture was apparently Stadler, in Johannesburg in 1908 or thereabouts. With such a material as rock, there being no yield point as there was with metal, elastic strain passed without interval to fracture. The author was careful to say, 'From this it is deduced', not that Kick deduced it.

It was, nevertheless, quite a reasonable deduction, and 'Kick *versus* Rittinger' was a title of controversy. Accepting Kick, the work done in crushing a given mass is unity at each stage in reduction, then, since the resulting total surface increased geometrically; the energy required per unit of surface would be greatest at the first step and least at the last, ranging all the way from some capital X ft.-lb. to some small x ft.-lb. This was his, the speaker's view. It was not constant throughout as Rittinger's law would assume. More simply, the greater the average size of piece the greater expenditure of energy necessary to produce unit new surface. If Rittinger's law were accepted the energy to produce new surface would be the same with large size as with small; capital X would in fact equal small x right down through every stage of reduction.

The author then went on to cite previous experimental determinations and particularly those by Gross, who, using a drop-ball on graded sizes of quartz varying from 4-mesh to 100-mesh, found that it required 3.82 ft.-lb. to produce 1 sq. ft. of new surface.

Following this, he gave his own impact tests on carefully-selected Witwatersrand quartzitic rock, when, instead of dropping a weight on the rock, he dropped the rock on to a steel plate. The results were given in Table VIII and the speaker had condensed the results given in that Table and in subsequent Tables into a tabulation (Table XXIV). It was the author's opinion that the impact tests in Table VIII supported Rittinger, but it seemed to him, the speaker, that they supported Kick rather than Rittinger, in that with larger size of test piece the energy required to effect unit fracture-surface was greater.

In Table IX he gave the relevant data relating to shear tests. Cross-section was probably the important dimension in shear, because shear had to cut across the test piece; and here with the larger pieces the ft.-lb. required per unit surface was greater than with the smaller. With the bending tests, Tables XI and XII, of course, tension came in, but here again the pieces of 1-in. core and 0.854-in.^{2} cross-section required more energy per unit fracture-surface than the pieces of $\frac{13}{16}$-in. core and 0.528-in.^{2} cross-section, these results being summarized in the text on page 12.

The next results, given in Table XV, related to compression. Here apparently they had the reverse relation—that is to say, with the larger piece a less amount of energy per unit of surface was required. But if they turned to the photographs of the smaller test piece shown in Plates III and IV it would be seen, in Fig. 4, for example, that very little fine material came with fracture, and it was with fine material that big areas of surface came and consequently less energy per unit of surface. Test pieces 1 and 2, the larger pieces which yielded much more fine material, were from higher up in the same hole as test pieces 3–5; test pieces 6–8 were from another drill-core. Fig. 4 was particularly interesting because it showed the shear planes along which fracture developed. *Finally, with* the tension tests given in Table XVI, the interrupted relation of larger size of test piece with larger energy requirements per unit of fracture surface was pronouncedly resumed.

This relation could readily be pictured: one could imagine having to break a sphere of quartzite the size of a pea and alongside of it a piece the size of a golf ball. One could smash the pea-size quite readily with a carpenter's hammer and get a large resultant total surface, because in the smash the particles would be small. But the same blow given to the piece the size of a golf ball would effect nothing. One would require a good blow with an engineer's hammer or even a miner's hammer before any impression was made, and that impression would be a more or less clean fracture right through, and comparatively little new surface.

It seemed patent to him, the speaker, that the energy required to produce a unit of surface on the bigger piece was greater than that required to produce a unit of surface on the smaller. All the figures, with the exception of those of compression, appeared to bear out Kick's law rather than Rittinger's law. The author, however, summed up with the statement, 'It is further concluded that, in crushing and grinding Witwatersrand quartzites, it requires approximately 3.5 ft.-lb. to produce 1 sq. ft. of surface area'. Apparently, for reasons given, the results of the impact and the shear tests were not used in coming to this figure, nor perhaps the tension results, but only those of bending and compression. It was obviously a round figure, but it seemed to be confirmed by the work of Gross, previously mentioned, who found that the energy required to produce 1 sq. ft. of new surface was 3.82 ft.-lb. That figure of Gross's was the result of many experiments with results satisfactorily regular throughout; but they were not on single particles, not on test pieces, but on a mass of crushed quartz particles put into a little mortar and levelled off, a steel ball being then dropped upon a closely fitting anvil inserted to rest upon the particle bed, the resulting new surface being measured. When a stamp was dropped upon a mass of crushed particles the first effect was a cushioning, the particles had to be rearranged, and there was loss of energy by friction before effective crushing could take place. The results of Gross accordingly appeared to the speaker to be of doubtful validity. If it was desired to know the resistance offered by strength of the material it must be arrived at by means of individual test pieces, the way the author had proceeded. It would be asked how that was to be done with small particles.

Well, it must be done somehow or other. One could imagine small particles all of a size arranged in a single layer on a die and a weight allowed to fall on them ; or one could project the particles at high velocity in an air-borne stream through a jet against a hard surface. The agreement between the author's figure of 3.5 ft.–lb. and Gross's figure of 3.82 was to him not so impressive as might appear.

Then the author passed on to the determination of the efficiencies of industrial machines, and he considered here again that, the feed changing from coarse to fine and the setting of the gyratory crusher from ⅜ in. to 1¼ in., the results supported Rittinger. But the efficiencies given in Table XVIII, based on available power, of 32.6, 31.5, and 34.3 per cent all had reference to the constant figure of 3.5 ft.–lb. to produce 1 sq. ft. of surface area. From this he went on to stamp-milling, using different screen-apertures, finding there, once more assuming the 100 per cent efficiency of the figure of 3.5 ft.–lb., that the stamp with the smallest aperture of the three was the most efficient. Generally, but perhaps not necessarily here with these large apertures, the finer the aperture in relation to the material crushed the less efficient the stamp (Table XIX).

With regard to tube milling, Table XXI ; here again everything was referred to the 100 per cent. efficiency of the 3.5 ft.–lb. He, the speaker, would say that getting down to the finest material the crushing measured per unit of fracture surface was much easier, and if that were so the efficiencies given were that much too high.

To sum up, it must be said that the author in his paper had given them abundant information and a large number of data—upon some of which he had not had time to touch—from which they could make their own deductions and draw their own con-conclusions, as he himself had taken the liberty of doing. The author made acknowledgments at the end to various colleagues who had assisted him, and in particular to the staff of the Rand Mines Mechanical Laboratory. The great preoccupation on the Rand at the present moment was descent into greater depth. Concerted research there was focused on the question of how much deeper they could go and on the associated questions of ventilation and rock pressure. They were all the more indebted, therefore, to him for initiating and carrying through at the present juncture this investigation into crushing and grinding efficiencies. He finally appended a list of authorities. The speaker was sure that Mr. Prentice's own name and the paper he had had the privilege to introduce that evening would figure prominently in any future bibliography.

Table XXIV.

Tabulation of Experimental Averages,
showing the conformity between size of test piece and energy required to produce unit new fracture-surface.

VIII *Impact*			IX *Shear*			XI *and* XII *Bending*		
Group	*Test Piece Weight*	*ft.-lb. per ft.*^{2}	*Group*	*Test Piece Cross Section*	*ft.-lb. per ft.*^{2}	*Group*	*Test Piece Diam.*	*ft.-lb. per ft.*^{2}
	lb.			in.^{2}				
A	2·253	+59·6	1— 9	0·854	23·1	*Centrally Loaded**		
B	1·803	43·3	10—15	0·528	12·5	1— 2	1 in.	4·24
C	1·057	41·4				3—11	$\frac{11}{16}$ in.	2·82
D	0·498	+33·3				*End Loaded†*		
						1—12	1 in.	4·09
						13—15	$\frac{11}{16}$ in.	3·70

XV *Compression*			XVI *Tension*		
Group	*Test Piece Weight*	*ft.-lb. per ft.*^{2}	*Group*	*Test Piece Weight*	*ft.-lb. per ft.*^{2}
	lb.			lb.	
1—2	0·403	2·95	1— 4	0·814	13·30
3—8	0·174	3·77	5— 8	0·474	6·66
			9—12	0·273	2·88

*See summary of these tests on top of p. 18.
†See summary of these tests lower down on p. 18.

Dr. Heywood said that he had prepared a written contribution to discussion on this paper and in view of the lateness of the hour he would send it in to be appended to the discussion. He added that although he had cast doubt on the validity of using a compression test as a basis for estimating the relative efficiency of a fine-grinding machine, yet he would agree with Mr. Prentice that the method of evaluating the performance of industrial mills by means of a ratio of energy to surface was exceedingly useful and one that should be adopted as far as possible. He had plotted the ratio of energy to surface against the output of a coal pulverizer. If they could devise an ideal comparison process then the value of energy to surface for the ideal process divided by the value determined on the pulverizing mill would be the relative grinding efficiency of the mill for that particular output. The problem was to devise that ideal crushing process. Progress in elucidating the fundamental principles of crushing and grinding had been slow, but there was an accumulating mass of data and Mr. Prentice's paper constituted a big step forward.

Mr. R. T. Hancock desired to offer some comment on the Kozeny-Carman equation, which was the basis of the surface area determinations (Method III) on *minus* 20-mesh material, as carried out at the Rand Mines Laboratory.

He had recently put forward equations of one general type for the upward flow of fluids through suspensions and for downward flow through beds of granular material,* and where Darcy's law applied his equation and the Kozeny-Carman could be put in the comparable forms :

$$\textit{Kozeny:}\quad \frac{d^2(V)^3}{2k.\ v(1-V)^2} \times \frac{H.\rho}{h(1-V)} \times \frac{4g}{3\mu} = 24$$

$$\textit{Hancock:}\quad \frac{d^2(V)^{4.5}}{v} \times \frac{H.\rho}{h\ (1-V)} \times \frac{4g}{3\mu} = 24$$

In the Kozeny equation the constant k was assigned the value 5 and Carman had been at pains to establish this figure from the available experimental data. If it were actually a constant, then equality between the two equations could only be obtained by giving d slightly different values in each. The value of d entered the equation for the specific surface of a powder, as—

$$S = \frac{6\ (1-V)}{d}$$

so that if d was not the same for both equations, then the specific surface, S, would be different according to which was employed.

The specific surface of a powder was a constant and should not vary according to the equation used, but if the function of the voidage, V, which appeared in an equation was not correct, then the surface figure would be found to vary according to the voidage at which the powder was tested. It was, therefore, significant that where the Kozeny equation was the basis of the determination the result was not the same on the whole sample as on the fractions

*'Symposium of Papers on Flow of Fluids'. *Proc.* Inst.Mech.E., *Vol. 153,* War Emergency Issue No. 5. 1945.

into which it could be separated by screening, as the three voidages would all differ.

If, on the other hand, the proposed equation was correct, then, subject only to experimental error in determining the voidage V and the velocity of flow v, either method should give the same figure.

To convert specific surfaces in sq. cm. per g. obtained with the Kozeny equation to those which would be given by the equation proposed, the former should be divided by the square root of $10(V)^{1.5}(1-V)^2$.

This would give specific surfaces about 4.5 per cent greater than the Kozeny figure when the voidage was 48 per cent, and about 5.5 per cent greater when the voidage was either 38 or 48 per cent.

It would be of extreme interest if the author would apply this test to any figures he had for determinations on the whole sample and on the fractions obtained by screening and publish the results.

Mr. E. A. Knapp said that the author uttered a truism when he stated that the object of crushing and grinding was to make particles smaller and smaller, but the speaker believed that this was the wrong conception. He believed that the real reason for crushing and grinding was to produce free mineral particles and if the crushing or grinding, either of valuable material or of waste, took place any further than was necessary for this purpose it was a waste of energy.

The author had dwelt upon the efficiencies of the actual crushing and grinding machines of different types, but with a ball-mill they had always recognized that the mechanical grinding efficiency was low. It was for this reason that ball-mill grinding using a classifier and ball-mill in closed circuit was developed. It was a grinding unit and, therefore, they could not divorce the classifier from the ball-mill as the author desired. Even the installation of a jig or a unit flotation cell to remove a coarse valuable mineral played some part in improving grinding efficiency and not enough attention was paid to the removal of coarse absolute waste particles —say by jigging the classifier sands—which might be possible in the grinding of certain ores. If by such elimination a 5 ft. diameter mill was used when previously a 6 ft. mill was required then a definite achievement had been made.

The next thing that struck him was the efficiency of the crusher, which, by the author's figures, was not much greater than that of the ball-mill. They had always been under the impression that the crusher had a much greater efficiency. They had even gone so far as to make finer and finer products for feeding to the ball-mill, thinking it was more efficient. He had some idea that the author's figures were wrong at some point; he rather thought at the *minus* 20-mesh fraction. In the feed to the crusher there was 87.1 per cent of the total surface area in that fraction. It could not be expected that the crusher would do any work on that fine material. He thought, therefore, that the test would have been more correct if the *minus* 20-mesh fraction was eliminated by washing thoroughly, before starting the test. The speaker remembered that in the old days, when he used to do coefficient of grinding resistance tests against Portland ore, he always eliminated the *minus* 200-mesh fraction from the feed sample. He would like to see some further tests go forward with the elimination of the *minus* 20-mesh fraction, or coarser grading—even so far as discarding all products finer than the crusher discharge opening.

The question of sampling had been brought up in the discussion of the previous papers. It must be very difficult to take a sample that had the identical sizing analysis as the bulk. The larger quantity of sample of a flowing pulp was also difficult to cut down to practical quantity. The best the speaker knew was to pass the product of the automatic sampler over two sets of Jones rifflers, which would cut the bulk sample by four. This smaller quantity could then be handled by a laboratory pressure filter. To sample the filter cake was then an easy matter. Mr. Prentice did not give them any details as to how the samples were taken; if they were not taken accurately then the whole of his figures would be wrong.

Mr. E. J. Pryor said that the main point of his criticism of an excellent paper was that he felt that Mr. Prentice was up against a very real difficulty in the selection of the material on which he worked. There was a world of difference between the breaking behaviour of an ore which contained 1 per cent of a sulphide in a quartz matrix and 5 per cent of sulphide. There was a world of difference between two samples, one of which was a random piece with random contours dropped on to a plate and allowed to smash according to whether a small portion of the ore impinged thereon and the other a perfectly flat surface dropping on to a perfectly flat piece.

Mr. P. Rabone thought that the moral of the paper was that ball- or tube-milling was fundamentally wrong as a method of fine grinding and that no amount of research would increase the efficiency of the process more than a few per cent. Ball-milling was analogous to trying to break a rock face underground by battering it with a power hammer or wearing it away with a grinding wheel. He did not suggest, however, that the work of Mr. Prentice and his associates was wasted. On the contrary it was exactly the type of investigation which an industrial research organization should make. But he considered that any fundamental research in size reduction in the fine range should be directed towards finding a method of disrupting the particle from the inside, not towards cracking it more efficiently from the outside.

The President expressed the thanks of the Institution to the author for his interesting paper and to Professor Truscott for introducing it.

CONTRIBUTED REMARKS.

Dr. Harold Heywood: The controversy of Rittinger's *versus* Kick's laws for crushing still persists, nearly 100 years after the enunciation of the former. On the one hand many experimenters have shown that in fine grinding the surface increase is proportional to the energy applied, yet it is also established that the strain energy stored in an elastic and homogeneous material at the instant of fracture is proportional to the volume of the test piece, provided that the energy is uniformly distributed. Perhaps the reason for the lack of agreement between the two laws is that in interpreting Kick's law it is postulated that a cube can be neatly broken into eight, or some other whole number, smaller cubes. This does not occur in practice, for a cube when shattered by compression will form into several large pieces and a great number of smaller particles ranging down to the finest dust. Hence it is impossible in practice to fulfill Kick's original postulate. It should also be noted that Rittinger's law can only apply if the physical constants for the material do not alter with varying particle size.

Estimates of the grinding efficiency of mills made by various research workers fall into two groups—namely, values of the order of a fraction of one per cent, and values in the region of 20 to 60 per cent. The former estimates are based on the surface energy of the crystal lattice and the latter are relative efficiencies of the mill compared with an ' ideal ' process of crushing—*e.g.*, by compression test or impact under controlled conditions. The latter is a more practical criterion of the performance of an industrial mill, provided that a comparable ' ideal ' process can be devised.

I am particularly interested in the method of estimating grinding efficiency used by Mr. Prentice, as I used a similar method to estimate the efficiency of coal pulverizers. These researches were not published in full, for a reason that will be explained later, but some of the experimental data may be found in the Reports of the Fuel Research Board and in the *Journal* of the Institute of Fuel.* To illustrate the point at issue, I have plotted (Fig. 5) the energy applied against the surface increase for compression tests on coal cubes, tests with experimental crushing rolls and with a small ball-mill, both these machines being arranged so that the energy applied could be measured with a high degree of accuracy. The curves have been plotted to logarithmic and to linear scales, the former being considered first. The average of the ratio energy/surface increase was 0.68 ft. lb. per sq. ft. for coal cubes tested in

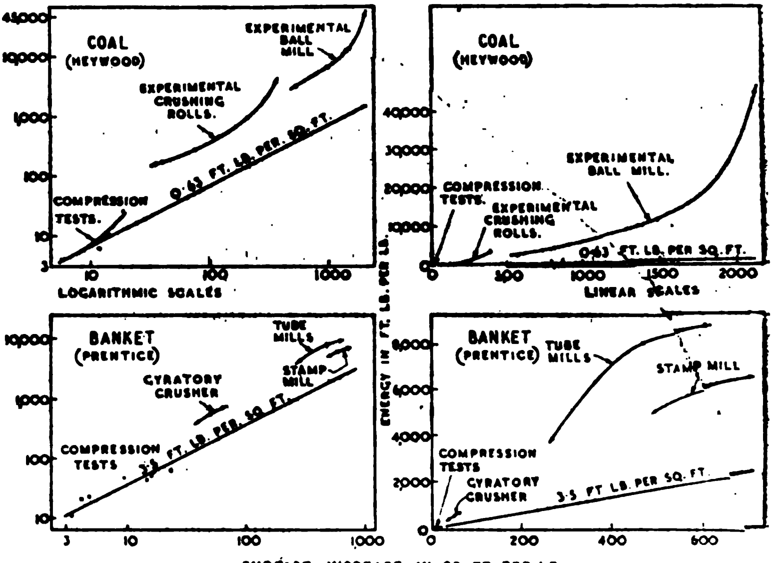


Fig. 5.

compression, and this line has been extrapolated so that the relative efficiencies of the machines may be determined as described in the paper by Mr. Prentice. The lower set of curves are plotted for data on the crushing and grinding of banket taken from the paper. When plotted to logarithmic scales these curves appear very convincing, but the illusion becomes apparent when the co-ordinates are linear scales. The small range of values of energy and surface covered by the compression tests is then realized and it is clear that the validity of extrapolating the compression test

*Annual Reports of the Fuel Research Board; 1936, p. 49; 1937, p. 42; 1938, p. 71.

Journ. Inst. Fuel; 1935, vol. 9, p. 94.

results over a 100-fold range of energy and surface values is very debatable. It would seem to be much more probable that the physical properties of a material change with size and that the ratio of energy/surface increases with decreasing size of particle. This would make the relative efficiencies of the industrial mills, after allowance for mechanical friction losses, even greater than as estimated in the paper, but there is no reason to believe that such an effect is impossible. The postulated increase in the energy/surface ratio might be due to the fissured structure of materials, which is very pronounced with coals and has been shown by recent researches to exist even in hard crystalline minerals.

The range of output with the experimental coal-grinding mills is much greater than can usually be obtained on industrial mills because of interference with plant operation, and this is one of the disadvantages of research on industrial units. The experimental mill results show a minimum value for the energy/surface ratio, with higher ratios at low outputs when the elastic energy of compression forms an appreciable proportion of the energy applied, and also at high outputs when the mill is overloaded and choking occurs. I have previously demonstrated that the efficiency of crushing is a maximum when the energy applied is only just sufficient to fracture the particles completely.

Determination of the mechanical energy losses of an industrial mill is not an easy matter; for instance, in the case of a tube- or ball-mill, it is not sufficient to run the mill empty. For an accurate determination of the frictional losses, the mill should be filled with a balanced load of concrete blocks equal in weight to the ball charge, and a measured torque applied by means of a friction brake fitted to the mill casing. By this means the transmission gear is loaded similarly to normal running conditions.

Although I have ventured to cast doubt on the validity of extrapolating the results of compression tests as a basis for estimating the relative grinding efficiency of fine-grinding mills, the method of assessing the output of a mill in terms of energy per unit surface increase described by Mr. Prentice is undoubtedly the most effective way of comparing different crushers and grinding mills. In applying this method to coal pulverizers, I have plotted (Fig. 6) curves relating the energy/surface ratio and the rate of output. The external characteristic refers to the energy applied to the mill shaft, and the internal characteristic to the energy applied to grinding. Hence the ratio of the internal to the external characteristic is the mechanical efficiency of the mill at any output, and the ratio of the 'ideal' energy/surface ratio (if this can be determined) to the internal characteristic would be the relative grinding efficiency of the mill. Progress in elucidating the fundamental principles of crushing and grinding has been slow, but there is an accumulating mass of data, and Mr. Prentice's paper constitutes a big step towards a solution of the problem.

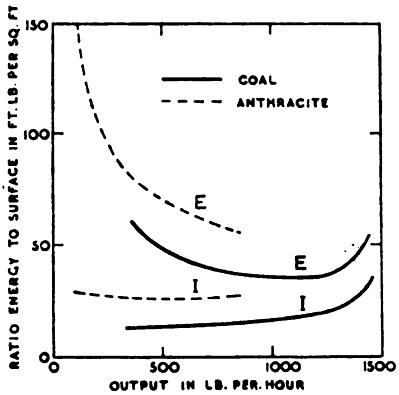


FIG. 6.—Energy-surface characteristics of impact pulverizer.
E.—external characteristic of pulverizer and fan.
I.—internal characteristic of pulverizer.

Mr. E J. Pryor: This paper provides a valuable empirical approach to our knowledge of its subject, particularly since all the Rand banket is of one generic type and has a more consistent crushing reaction than most ores. In respect of the author's opening sentence I prefer to regard the primary function of crushing and grinding as preparation—either to grade material; to liberate ore-constituents in such a manner as to present an optimum feed to a concentrating process ; or to expose optimum surface of the desired mineral to special chemical reagents in the pulp. Further small points of criticism are that most laboratories today use the Tyler $\sqrt{2}$ series of screens and not the Tyler 2, and that the term ' diameter ' in Table I, col. 2, is less lucid than would be ' mean mesh '. Again, screens can become unsatisfactory well on the coarse side of 50 microns, and unless they are carefully calibrated and used in a standardized manner, with control for ' bunching ', dried-on salts, compacted particles and fissuring, the figures for total new surface will be misleading.

Papers presented by Gaudin and others* should be carefully considered by all concerned with fine-grinding technology. They show that when homogeneous rock receives a single shattering blow

*A.I.M.E. Tech. Pubs. 1779 and 1819 (Nov., 1944, and May, 1945).

under controlled conditions, the same amount of new surface is developed in each grade of a $\sqrt{2}$ series of screens, and that rock always breaks to random sizes. Table III, by its theoretical discussion of cubic breakdown, might distract attention from this most important breaking relationship.

The author gives a list of five forces at work in breaking rock. I find it easier to resolve these down to two—compression and tension—although shear, which is a mixture of simultaneous compression and tension, might be considered as a third. Impact has to do with the rate of application of one or both of these forces and can, therefore, be thought of as an element in their operation. If we admit it as a separate force we should also bring in the method of grip of the work-piece, the anvil, the inertia of the compressed system, and similar matters. In bending there is straight compression on one side of a neutral axis and straight tension on the other.

On p. 11 the author says ' additional classification does not to any extent alter the amount of grinding performed by a mill '. This surely needs qualification. Removal of ' finished ' material is not the only objective in classification. The settling rate of a solid through a fluid varies according to the frictional drag, which contains two elements—viscosity and total surface area involved in friction at the solid-liquid interface. Hence the mixture of grain sizes in the crop of the mill determines the sink-float characteristics of the slurry. I think of these as pseudo-viscosity and lubricating effect. By ' pseudo-viscosity ' I mean the retardation of settling due to surface friction (a function of total surface presented per unit volume by the mixture of particles in the slurry), the prefix ' pseudo ' being introduced to make it clear that true viscosity or molecular shearing of the liquid fraction of the pulp is not under consideration. The lubricating effect enters milling work in so many obscure ways that a discussion would need a paper to itself. It affects the transporting power of the mechanical components of the closed circuit ; the keying of liners to balls and balls to each other and to the work-pieces (ore in circuit) : the efficient proportioning of solid to liquid ; the displacement of centre of gravity, which equates the mill's input of useful grinding power ; the buoyancy of the balls and hence their effective kinetic energy and combined crushing mass in the bottom segment of the mill. New surface produced is generally conceded to be proportional to the useful work put into the mill and the classifier, therefore, plays a most important rôle in sorting over the mill-discharge and returning not only insufficiently-ground material, but also the best keying load for using the mechanical power of the mill in such a way as to produce maximum displacement of load centre and hence maximum ammeter reading in a mill in good running condition. I would qualify my phrase 'useful work' by adding that its usefulness depends on the proposed end use of the ground material, because grinding is only a means to that end.

Coming now to the tests, it is necessary to remember that any work on ore composed of several elements must, in our present state of knowledge, be largely empirical and beyond fundamental formulation. Much research work has been done using glass spheres as a source of homogeneous material. As these are produced by shot-tower methods they are imperfectly annealed and are erratic in their crushing response as revealed by modern methods of surface measurement. Glass is a simple super-cooled liquid, elementary in its behaviour when compared with run-of-mine material.

Yet it is run-of-mine material which provides us with our livelihood, not chunks of glass. With mine ore there must be discordant results and in fundamental research we are still seeking an ideal reference material by which our tests can be calibrated. The research ideal is to control all possible variables, and then to change one only, step by step. Here 'practical' tests meet a substantial difficulty. Is it possible to obtain a series of physically similar work-pieces when dealing with a non-homogeneous rock? If there is any change in the percentage of gangue, sulphide, and gold; if there is change, however slight, in grain structure; if variation in mode of severance of the rock from its lode, or of pre-stressing the working places has occurred; the tenacity of association between the bonded phases of the various pieces of ore varies and so will their crushing response. For strict reproducibility all test pieces should have similar assay, distribution of contents, grain structure, geological history, and mode of severance. Obviously this is an impossible ideal and we must therefore rely on the statistical method in empirical research, averaging a sufficient number of test results while controlling as many variables as possible. I venture the opinion that, given the validity of the above argument, more tests are needed before it will be safe to set up the reference figure of 3.5 sq. feet/ft.lb. Individual tests in the Tables lie well off the average finally struck and this indicates the need for a wide base from which to compute that average.

Next, do such things as the size, the shape, and the rate of loading of the test piece matter? In the tests summarized in Table VIII *the rate* of application of impact is widely varied; also the size and hence the inertia and elastic reaction of the test-piece is out of control. Again, the application of the shattering force appears to have been left to a random collision between an unknown rock contour and a flat piece of steel. These tests, therefore, contain several uncontrolled variables and this may have a bearing on the random individual results.

Turning to the shear tests, the mode of preparation of the specimens seems important. Were they taken from the same place and drilled in similar fashion so that no more incipient fissuring would be likely to be produced on one than another specimen ? The author's mention of vibration introduces that complex of phenomena, propagation of fracture by shock-loading, of which we have had so many strange examples during the bombing of Britain. In Table IX the velocity with which the blow was administered varies considerably and the core diameter somewhat, suggesting dissimilar drilling conditions, though the results are surprisingly consistent if No. 1 is omitted.

In the ball-mill tests part of any new surface produced comes from wear of liners and balls. Unfortunately the mill does not discriminate between what we wish to grind and what we would like to preserve, so the Brinell figures of the grinding media and the corrosive activity of the ore (in a chemical sense) become factors in assessing grinding efficiency, though this latter effect is, I believe, negligible in Rand practice. In Table XXI (Plate 2) screen analysis stops at 200-mesh. A surprising amount of the total new surface occurs in the *minus* 200-mesh fraction and some elutriation or infra-sizing work would be of value in bringing this into the reckoning.

Some writers rate the efficiency of the ball-mill as a fraction of 1 per cent, while others set it high. To me it seems that the right empirical approach in our present state of knowledge is not some theoretical abstraction, but the practical question 'How efficient is my grinding circuit in this mill with this ore, as compared with 100 per cent efficiency for this class of machine ?' A question of this sort is helped tremendously by papers such as the one under discussion. If we can find a generally acceptable reference ore and if we can agree on a reasonably standardized handling technique as being in practice 100 per cent efficient, we shall have a standard which, though empirical, will make possible direct comparisons. I can think of no mine field so well provided by nature with the standard ore as the Rand, or by the far-sightedness of those controlling South Africa's gold-mining industry with the technical means to work out a standard procedure for such comparative work.

The fundamental approach to crushing aud wet-grinding problems is so beset with problems of control that much patient research will be needed before reliable formulation as a guide to basic design can compete with the empirical approach. Pure research must at present continue cautiously, using homogeneous material, but applied technology cannot wait. It is becoming increasingly clear that good grinding is the key to good extraction. The whole subject is of major importance, technically and economically, and far more research is needed, and needed now. Whether the ball-mill is a machine of low or high efficiency, it is the best tool we have at present, and must be used understandingly. This paper will help toward that end, and I thank the author for his valuable addition to our literature.

FURTHER CONTRIBUTED REMARKS

ON

Tunnelling in Gibraltar during the 1939-1945 War.*

By W. H. WILSON, *Member.*

Mr. R. S. Botsford: This paper is particularly interesting, both as a description of what went on at Gibraltar during the recent war and of the tunnelling and dimensioned blasting of large excavations, where the uniformity and reasonable smoothness of the roof, walls, and floor merit special consideration, to accommodate the population of the Rock in time of emergency. Had they not been so pressed for time the walls of the chambers might have been sawn with wire, as at Carrara for cutting marble, with less fracturing of the walls; as it is they seem to have made a very good job. The excavations will remain there for all time, doubtless with additions as required through future centuries. Already they must be considering double doors to all openings to avoid the incendiary effect of atom bombs.

Plate No. 1 shows the lay-out for extracting the fumes and smoke from blasting, but the fresh air supplied to the face was contaminated with dust, as the hand-drills were not adapted for the use of water for laying the dust. The amount of limestone dust seemed harmless, but must have been uncomfortable to the operators. One firm mentioned makes jack-hammers for water which with a slight alteration can be used dry when desired, as may happen at the surface, especially if there is a wind to blow the dust away. In the Medellin single-track railway tunnel in Colombia, four kilometres long and worked from both ends only, the Canadian contractors (Fraser Brace, Ltd., of Montreal) exhausted the fumes from the blasting of the 42 holes using first, second, third, and fourth delays as did the author and after 12 minutes blew air in on to the face; they could comfortably return to the face in 20 minutes from the blast, always supposing that the two men listening agreed that all had gone off. The floor was blasted out to one foot below the specified depth so as to have no delays in laying the track for the loader, for even a lump of unbroken rock six inches high in the way of the rail will cause considerable annoyance and delay while it is being removed.

*Bull. 475, November, 1945.

It is most discouraging for any driller to start the shift with a drill less effective than that of his companions and so, on this Colombia job, the drills of each shift were daily taken apart, blown back with compressed air, washed if necessary, air-sieve cleaned, and lubrication inspected, any worn parts replaced, etc., so that each driller started work with what was to all intents and purposes a new drill. The screen in the compressed-air connexion may get clogged to a certain extent, thus reducing the air pressure at that drill and, therefore, the work done during that shift. Iron rust comes from the pipes and finer particles may pass the screen and cause wear in the drill. Attention to these and other similar considerations contributes to faster and, therefore, cheaper penetration of headings, etc. In cold countries, where the air, hot when compressed, cools off in transit to the work place, it is worth while to have a heater to heat the compressed air just before use—that is, as near the face as ventilation will permit; some 25 or 30 per cent efficiency is gained that way.

Capt. G. A. Schnellmann: It is abundantly evident that every aspect of this project was carefully planned, the points made below being requests for enlightenment rather than criticisms. What, for instance, was the reason for adopting the pyramid cut in an 8-ft. by 8-ft. heading ? The pyramid is much more difficult to drill than the wedge, particularly with hand-held machines and it is doubtful whether any limestone requires it. This doubt is supported by reference to Fig. 5, from which it can be deduced that the easers were left with a burden of nearly three feet, yet they broke. Moreover, only 20 holes were needed for the whole round. What, again, was the reason for adopting so short a round as 5-ft. in a 15-ft. wide heading ? Possibly the answer to this is the overriding desirability of completing the entire cycle in a shift, since it is evident from the remarks on p. 4 that the machines were quite capable of drilling 8-ft. or even 10-ft. to 12-ft. holes.

The use of the diamond-drill for drilling blast-holes is still sufficiently novel to excite special interest and from other publications on the subject it is fair to assume that more difficulties were met with (especially in connection with the bits) than Major Wilson's brief remarks on this aspect would lead one to suppose. Were any designs other than the solid concave tried before this was adopted ? Has the author any criticisms to make of this design, or any suggestions for alternatives ? It has been noted as one of the disadvantages of the solid bit that the great difference in linear speed between the central and peripheral stones causes very *uneven* wear. Evidently this difficulty was encountered on The

Rock, as it is stated that '. . . in course of the work some of the diamonds needed re-setting'. Which stones most commonly needed re-setting? One mining company experimented with a removable central insert to minimize this problem. The author refers to 'diamonds'. Were any comparative tests between carbons and bortz possible, and, if so, with what results? Finally, it would be interesting to have more details of the Speed rods and of the special feed screw, as well as of the feed ratio—revs. per inch of forward travel.

Mr. I. W. Morley: The application of modern underground mining practice to military engineering, as illustrated in Major Wilson's informative paper, is of interest to all mining men and, particularly, should provide a valuable addition to the growing literature on blast-hole drilling. Probably many of the points raised in the present contribution will have been brought forward in the discussion; however, at the expense of possible reiteration, it is desired to mention some of the features that are most striking to a civilian, whose main activities are concerned with safety and, as an observer, the development of blast-hole drilling at Mount Isa (Queensland) during the war period.

As an advocate of 'safe practice' it is disappointing to learn that the Army has not yet discarded 'dry' drilling. Whatever may have been the advantages of simplicity in retaining 'dry' jack-hammers, the production of rock dust by drilling without any suppression devices is basically wrong and it is surprising to learn that the mining engineers in the Royal Engineers did not insist on the use of wet machines. Though no injurious effects from the limestone dust were disclosed in the X-ray chest examinations it cannot be denied that all rock dust should be suppressed, if only in the interests of good practice. This is particularly desirable, since the 1939–45 Gibraltar experience will no doubt guide future military engineering practice and military engineers without mining training may, from lack of knowledge, use 'dry' machines in siliceous ground. The problem of using salt water for dust suppression on The Rock is noted, but notwithstanding such a local difficulty it is to be hoped in this respect that military practice will shortly be raised to civilian standards. It would also be of interest to learn whether the broken rock was 'wetted down' after firing and during shovelling. In this connection it should be mentioned that the conversion of a S.L.9 'dry' Handril to the 'wet' model is a relatively simple and inexpensive job and few extra spare parts need be carried. In view of the adoption of Holman scrapers, Eimco-Finlay loaders, and Boyles diamond drills, and the necessary training of operators for such modern plant, it is not thought that a change from 'dry' to 'wet' machines is a serious problem.

It is noted that drifters are preferred to jack-hammers for driving and, notwithstanding the excellent performance of the Handrils and teamwork of the crews, it is thought that automatic feed drifters of the Gardner-Denver or Ingersoll-Rand types might have eliminated one pair of drillers in the 8-ft. by 8-ft. drives. At Wiluna (Western Australia) in 1938 the writer observed a regular advance of 5 ft. per 7-hr. 12-min. shift in an 8-ft. by 8-ft. drive in schist with a crew of three men with two automatic drifters, ordinary drill steel, and an Eimco-Finlay loader. The face was bored out with from 16 to 20 holes and some 200 ft. of hand tramming of the mullock was included in the contract. Only those who have operated automatic feed machines can appreciate the great saving of physical exertion; with increased mechanization and the necessity for labour-saving devices their use must become more widespread.

The illustration of the Eimco-Finlay loader at The Rock shows that it was operating under the most suitable conditions, with relatively finely broken rock. With similar-sized material at Wiluna, Kalgoorlie, and Mount Isa they give excellent results. However, at Mount Isa, where the ore often breaks in slabs a square foot or more in section, it has been found that in such ground neither they nor the somewhat similar Gardner-Denver loader will stand up to the heavy work involved in 'biting' into the broken rock. Consequently, at that mine 32-in. hoe-type scrapers with 15-h.p. electric motors, or 54-in. scrapers with 35-h.p. motors are used in drives with scraper loaders resembling the Holman type used on The Rock. Such scrapers, without loaders, are used at Mount Isa in sub-levels, centre headings, and in flat (30°) stopes and have proved most valuable in scraping the ore into ore passes and chutes. At the Phoenix mine of the Central Norseman Gold Corporation, N.L., in Western Australia, where the ore-body dips at 45° and less, similar scrapers and loaders were extensively used in both development and stoping in 1939. However, where space and accessibility permit its operation, the Ruston-Bucyrus type of shovel illustrated in Fig. 23 is to be preferred.

Referring to the drilling round used: it is thought that even better results would have been obtained by using a 'burn' cut, instead of the pyramidal cut. All the holes being horizontal the use of hole directors would have been unnecessary, while the 'hard homogeneous limestone' should prove ideal for the use of such a cut.

It is noted from the plates that hard hats were not worn underground, which is again surprising in view of the Canadians employed and the availability of the Army 'tin hat'. In this connection also one might ask whether protective equipment—such as safety hats, gloves, and goggles—was worn by the miners. The use of rope (presumably hemp) ladders underground, although they are probably standard military equipment, does not appear a desirable practice. For access to rises wooden ladders with steel rungs, or chain ladders, are much preferable and almost universally prescribed by Mines Regulation Acts. Underground examination of ropes tends to be perfunctory, and deterioration is often undetected.

The use of diesel lorries, shovels, bulldozers, and locomotives underground is a forward move and in this respect the R.E. is in the vanguard of development. It indicates the realization that mining is essentially an 'earth-moving' job and that earth-moving equipment and methods can be applied to it. Although the use of diesel locomotives underground has been common practice, particularly in coal mines, for a number of years, it is only recently that modern earth-moving equipment has been used underground in other types of mining. A paper entitled 'Mining a Deep Limestone Deposit in Ohio' appearing in *Mining Technology* for September, 1943, describes the use of diesel lorries and large shovels at a depth of 2,000 ft. in Ohio. In the *Queensland Government Mining Journal* for July, 1945, H. J. Redmond in an article entitled 'Diesel Lorry Transport Underground' describes the use of such trucks at Mount Morgan (Queensland), detailing the care necessary to minimize the risks from carbon monoxide and other gases. Regarding such risks, it is noted that although excellent natural ventilation conditions obtained on The Rock apparently apart from scrubbers on the locomotives no special devices were installed to prevent the exhaust gases of the locomotives, lorries, shovels, and bulldozers from contaminating the air. Such precautions appear desirable, particularly in view of the possibilities of similar work under much less favourable ventilating conditions. Analyses of the air, its quantity, and notes of recorded cases of objectionable fumes (if any) would be useful, if they are available. The production of objectionable fumes, mainly carbon monoxide, due to incomplete combustion of the wrappers of the long rigid cartridges used in the blast-hole drilling is also noted. It would be interesting to learn whether these fumes necessitated any special safety precautions in the clearing of the spoil after the blasts. It is not unusual to find objectionable fumes trapped in large heaps of such spoil for many days after the firing. The most effective remedy would be to remove the cause of the incomplete combustion, by the use of a length of instantaneous fuse, parallel with the charge of gelignite in the blast hole.

Considering the ventilation problem of the development ends and the use of rubberized ducting (commonly known in Australia and the United States as 'Ventube') and Meco fans, it would have been useful if the capacity, water gauge, and horse power of the fans had been detailed. The excellent performance of this equipment at Gibraltar is not decried, but comparable results have not been observed in Australia. Long lengths of Ventube have been found unsuitable as semi-permanent installations, owing to misuse, both wilful and accidental, by cutting and tearing in handling, etc. However, possibly military discipline imposed a higher standard of care of equipment on the miners on The Rock than can be obtained in metal mines. At Mount Isa in development ends the use of 16-in. diameter galvanized tubing has been found more satisfactory, and in the long run more economical, than Ventube, save for the last 100 to 150 ft. up to the face, even though natural ventilation was subsequently established and the line of galvanized tubing dismantled. At The Rock the Meco fans doubtless sufficed to deliver at the face the required volume of air through an 1,800-ft. length of ducting, in view of the equitable temperatures obtaining underground. Wherever air-conditioning equipment has been installed to reduce underground temperatures in metal mines it has been found that reduction in both temperature and humidity—however important—is secondary to maintaining a large volume of uncontaminated air at the face. At Mount Isa, with rock temperatures exceeding 90° F. on the lower levels, it is found necessary regularly to instal booster fans in series at 300 to 600 ft. intervals along the line in order to ensure sufficient 'coolth' at the face of long development drives. Such booster fans are of 4,000 c.f.m. capacity at 6-in. w.g. 2,850 r.p.m., and are driven by a 5-h.p. motor.

Amplifying the desirability of safe practice in military engineering it is understood that the United States Corps of Engineers has a safety branch. Safety is a special study requiring the direction of full-time enthusiastic officers and the co-operation of all concerned. Good safety practice gives results by increasing production, whether it be of ore or storage chambers. If such a branch of the Royal Engineers is not already operating, it is suggested that the Institution might care to make suitable representations to the War Office for its establishment.

Probably the most important technical advance detailed in the paper is the use of blast-hole drilling to control the safety and shape of large excavations and in this respect the R.E. with their peculiar problem of removing valueless rock to make a valuable storage space are dealing with the opposite set of conditions to those of the mining engineer, who wants to remove valuable rock at the minimum of cost and is relatively unconcerned with the final shape or safety of the excavation left, provided wall dilution or collapse are avoided. However, the success of the R.E. in this job will inspire the profession to tackle similar jobs in pump chambers, underground magazines, etc.

Although blast-hole drilling with diamond drills is now the accepted practice in large metal mines with suitable ore-bodies, the published literature, apart from the Canadian, Australian, and Rhodesian references given, has been unfortunately scanty. The brochure 'Blast Hole Diamond Drilling', by Olaf V. Lindqvist and published by J. K. Smit & Sons Inc. in 1944, gives a good summary of Canadian and American practice to that date, whilst the October, 1945, issue of the *Engineering and Mining Journal* has a most interesting tabulation of the present practice at many mines. Recently the manuscript of a report has been seen, which it is hoped will be published shortly in Australia; this report has been written by an Australian mining engineer who spent three months last year in North America making a special study of blast-hole drilling. In addition the writer has been privileged to watch the progress of this method at Mount Isa, where pillar robbing and stoping by diamond drills was introduced in 1989 and where, since 1942, blast-hole drilling has been used for all ore broken at this major producer, save terminal and flanking projections of the ore-bodies, which are benched with jack-hammers.

Examining Major Wilson's experience it is noted that 1½-in. holes were drilled; these are larger than the average, and since only 1⅛-in. and 1¼-in. diameter explosive in rigid containers was used the necessity for large holes is not apparent. Metal-mining practice has established (especially at Mount Isa) that it is desirable to concentrate the charges in blast-hole drilling in order to obtain good fragmentation. The use of rigid containers much smaller than the hole prevents proper tamping and concentration of the charge. If rigid containers were necessary (Mount Isa in holes more than 100-ft. long has not used them) it would appear that they should have been of 1⅜-in. or 1 7/16-in. diameter in 1½-in. holes. The larger diameter holes, admittedly, have less tendency to deviate from their course, but, irrespective of hole size, metal-mining practice favours the use of full-size cartridges of explosives, and in some cases the use of compressible cartridges, such as 'Tamptite', in blast holes, so as to fill the hole. At The Rock, where distribution of the breaking effect along the length of the hole was important, the same result might have been obtained with compressible explosive and wooden spacers, if the total charge of a hole, full of explosive, was too great. Mount Isa has found most valuable the use of a length of instantaneous fuse ('Primacord' or 'Cordtex') along the side of the hole with the explosive tamped tight beside it. In the siliceous dolomite copper ore-bodies of that mine explosive consumption in secondary blasting has been heavy and the use of instantaneous fuse has notably increased the fragmentation. In addition the question of placing the detonator at the top or bottom of the hole does not arise and only one detonator is used. The use of this fuse gives a very high detonation wave and shattering effect. It is thought that on The Rock such a fuse and compressible cartridges would have reduced the explosive consumption or alternatively permitted a wider spacing of the holes. Major Wilson has emphasized the desirability of instantaneous blasting and it is remarkable that the Royal Engineers did not try instantaneous detonating fuse for interconnecting and firing the charges since 'Fuse, instantaneous' is a standard Army issue. Such fuse may be fired by either ordinary detonators and safety fuse or by electric detonators and exploders.

In view of the almost universal use of pre-cast diamond drill bits in blast-hole drilling in metal mining it is surprising to learn that the bits on The Rock were handset. Was it not practicable to fly out pre-cast bits from England? The use of stellite for hard-facing calyx or other bits in coal-core drilling has been the practice in Queensland and elsewhere for more than 25 years, while during the past ten years cemented carbides of tungsten have been used for the same purpose. However, in the writer's opinion, such tungsten alloys and carbides can never compete with diamonds in blast-hole drilling in the hard rocks of metal mines.

The difficulties experienced in keeping the long diamond drill holes true in direction and grade are appreciated. As a matter of information it is of interest to note that at Mount Isa, in core drilling long holes (300 ft. to 1,100 ft.) from the hanging-wall across steeply-dipping beds of generally uniform composition, it was found that the holes tended to droop or rise, as the case may be, so as to become normal to the dip of the beds. This rule, which of course does not hold if the holes pass into a different class of rock, may have some bearing on the dip of the holes at Gibraltar, if they were drilled normal to the strike of the limestone beds. No comparable information is available concerning directional deviation from line in blast holes, but their ready deviation, one way or another, unless a coring bit is used, is the principal reason for the recent trend to limit their length to 60 ft. or thereabouts in metal mining. Incidentally it may be mentioned that the 140-ft. blast holes drilled at The Rock are among the longest of such holes so far used. Both Mount Isa and Aldermac (Canada) in recent years have successfully blasted holes from 100 ft. to 130 ft. in length, but this was done in pillar robbing. Even in pillar robbing shorter holes are now preferred wherever the overall cost permits.

It would be interesting to learn whether the R.E. attempted to cut the slots at the ends of the chambers with blast holes. From experience at Mount Isa it is believed that the slots on The Rock could have been cut in this manner, with a fan of holes from a central set-up, with a great saving of labour. In the same way once the central drive and cross-cuts at either end of the chambers illustrated in Figs. 11 and 12 had been driven the undercutting could have been done by blast holes, successively firing off the 'slashes'. Undercuts at Mount Isa have been similarly excavated. In the same manner the 2nd Operation in Fig. 17 could have been slashed out by blast holes.

Major Wilson's optimism regarding future developments in diamond drilling for general underground use is well justified and the work of the Royal Engineers at Gibraltar will give great impetus to this practice. In the same *Bulletin* as his paper, Messrs. Botsford, Hatton, and Alec Jones, in their several contributions to the discussion on Mr. Irving's paper on 'Some Aspects of Rock-Drilling Practice (The Witwatersrand Goldfield)' all emphasize the same point. If a small robust high-speed electric diamond-drill can be developed—and with the wealth of wartime engineering experience in special alloys and electric motors such should not be difficult—the writer is of the opinion that within the next 20 years hammer drills will be relegated to a minor rôle, even in development headings where short holes are the rule. Such a drill would also reduce the danger from dust in drilling. Notwithstanding its many advantageous, compressed air is a very expensive medium of power.

The writer desires to acknowledge the courtesy of Mount Isa Mines, Ltd., in permitting the publication of so many details relating to their underground practice which have not yet appeared elsewhere in print.

AUTHOR'S FURTHER REPLY TO DISCUSSION

Major W. H. Wilson: The contributions to the discussion of my paper by Mr. R. S. Botsford, Capt. G. A. Schnellmann, and Mr. I. W. Morley having just come to hand I am obliged to reply in the form of an addendum. Part of Mr. Morley's excellent contribution is additional information and does not call for further comment, while some of his points have already been dealt with elsewhere. It is evident, however, that he shares with many other members a misconception or an under-estimation of the supply and other difficulties with which we were constantly faced, and attributes to us, isolated on The Rock, all the resources of an established mining field in peacetime. Before the war there were no mining units or mining engineers in the Royal Engineers, so that when war commenced there were no mining resources. Whatever was achieved was the work of civilian engineers who had volunteered for that purpose, and I trust that some attempt will be made to carry forward the organization thus laboriously created.

Capt. Schnellmann and Mr. Morley comment upon the cuts used in the rounds. The pyramid cut, used in the 8-ft. by 8-ft. round, was really a relic from the time when fuse blasting was used, this form of cut ensuring simultaneous detonation of all the cut holes. It is easy to drill, especially with the help of the hole-director. When we turned over to electric blasting no attempt was made to devise a new round, as this small cross-section of tunnel was then falling out of use. In re-standardizing these rounds the 'burn' cut, which is now receiving widespread attention,* would obviously merit consideration. When I left The Rock we were experimenting with a diamond-drilled 'burn' cut to connect a series of ventilation tunnels with a system of big chambers. The connections came through the backs of these very high chambers and were, therefore, an awkward proposition. The plan was to blow through the cut, a distance of some 25 ft., in one blast. We did not succeed very well, the reason being that the holes were spread too far. It is to be regretted that this plan was abandoned after the initial failure. Mining engineers will be fully aware of the possibilities of the burn cut and will welcome all reliable information about it.

*Davis, A. Savile. 'Notes on the Development of the Blyvooruitzicht Gold Mining Co., Ltd., South Africa.' *Bull.* Inst. Min. Met., Nov., 1945.

Lorrimer, J. 'Some Uses of Explosives in Civil Engineering'. *J. Inst. Civil Engineers*, March, 1946.

I never doubted the desirability of the use of drifters—and Mr. Morley mentions the latest automatic feed machines—but we had to make the best use of the machines provided. The wet version of the hand-held machine is a most uncomfortable machine to use at shoulder height, especially to miners unprovided with the proper waterproof clothing. The same supply problem was at the back of the lack of miners' helmets, commented upon by Mr. Morley; these were not received until early 1945, and were at once put into use. Tin hats are heavy and uncomfortable and their use underground was optional.

Capt. Schnellmann and Mr. Morley comment upon the diamond-drilling bits. The diamonds used were bortz, and no tests were made upon the shapes of bits or the various sorts of diamonds. That wear was erratic was no doubt due, in part, to the fact that the bits were handset. The design I submitted for the tungsten carbide bit had a hemispherical end, with five inset teeth, but the bit eventually supplied contained, as described in the paper, splinters of carbide set in an alloy. The toothed bit would presumably need a special sharpening machine, with a diamondiferous wheel. A wide future may lie ahead of bits of this, and allied, types.

I hardly care to comment upon Mr. Botsford's suggested use of the wire-saw for cutting the walls of excavations, as, although I am conversant with its use in quarries,* I find it difficult to assess its possibilities underground. Although, no doubt, the walls of very big chambers could be cut with this machine prior to blasting, the back would still remain as the dominant factor, for which diamond-drill blasting seems the obvious answer.

I am quite alive to the uses of detonating fuse, yet unconverted by Mr. Morley's arguments in its favour. I favour electric detonation for diamond-drill blasting of the sort developed on The Rock, and venture to doubt his assertion that the use of detonating fuse down the sides of the holes would obviate the fumes arising from blasting. Since, as far as I know, we alone used these rigid cartridges, ordered specifically for the purpose at my request, it seems doubtful if Mr. Morley's comments are based on observation. The great practical advantage in the use of rigid cartridges, outweighing factors such as filling the holes completely with explosive, is the great ease of loading them in conditions where a stuck cartridge, with subsequent incomplete loading of a hole, would have prejudiced the entire blast, as disaster would certainly have followed the failure of a single hole.

*See especially '*Calyx Core Drilling*'—an Ingersoll-Rand publication.

Turning to the tests on the Meco fans and ducting, it is pointed out that this type of rubberized ducting has, at each end, a ring of stiff rubber material, and junction is effected merely by inserting one ring within another of the adjacent length, air pressure then expanding the inner ring to make an absolutely tight joint, of a streamlined nature, with no re-entrant angles. The practical significance of this ingenious device is considerable.

AUTHOR'S REPLY TO DISCUSSION*

ON

A Projected Central Mill for the Durham Fluorspar Industry.

By ANDREW PEARSON, *Member*.

Mr. Andrew Pearson: I have to record my thanks to Mr. W. C. C. Rose for his kindly act in introducing this paper on my behalf. In his position as Assistant Controller he had an intimate contact with all the work done by the Control and was therefore able to speak with some authority on the fluorspar position.

Mr. Pryor's remarks on the advantages of desliming before flotation would have had more value if the mill had been designed for dump material treatment. In this case, however, mine ore was to be the feed and the fines all carried high values which had to be recovered. As Mr. Wood pointed out in his reply, with which I am in full accord, the presence of some secondary slimes can sometimes prove to be an advantage in flotation. Mr. Wood also dealt with the query as to the relative advantages of rod- *versus* ball-mills and Denver *versus* other cells in a manner on which I cannot improve. In small projects of this type economic rather than academic considerations influence the engineer in his decisions. I would, however, quite agree with Mr. Pryor that in large-scale operations the relative characteristics of the various cell types should be carefully balanced before a decision is taken.

I have to thank Mr. Wood for his generous appreciation in his written contributed remarks, more especially since I have drawn on his own work for so much that is germane to the project. With most of the data he quotes I am, of course, acquainted, but I should like to endorse his statement that in this, as indeed in all other ore treatment schemes, generalization is dangerous and every problem should be investigated separately. The interpolation of data obtained from the treatment of one type of ore into a project dealing with another, outwardly similar but inherently different, type, may, and often does, prove fatal to the prospects of an over-ambitious or too-hasty promoter, and the knowledge that

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he has in consequence obtained a clearer appreciation of the meaning of the adage 'penny wise pound foolish' is little compensation.

Mr. Gill's remarks are very interesting and raise an important point when he suggests the fluorine content of imported phosphate rock as a potential source of fluorine for domestic use. I had not realized that the tonnage of imported rock and, consequently, of contained fluorine, was so large, and on looking into the matter was amazed to see that in the United States of America the fluorine content of their phosphatic ore reserves reached the figure of *200,000,000 tons*, or 75 times that of the fluorine content of the known fluorspar reserves of the Illinois and Kentucky deposits, which carry 85 per cent or more of calcium fluoride.* This, however, in no way detracts from Mr. Rose's statement that fluorspar is our only substantial source of fluorine in this country and as such should be treated with respect.

In those phosphatic rocks the average F/P_2O_5 ratio is around 0.11 ; in the Algerian and North African phosphates the ratio may be somewhat higher—0.12 to 0.14, for example.

Mr. Gill has indicated the condition under which this fluorine exists in the phosphate rock and its behaviour in the course of the conversion of the rock into superphosphate when much of it is driven off as gaseous hydrofluoric acid, silicon tetrafluoride, hydrofluosilicic acid, etc., in the presence of steam. As mentioned elsewhere the effect of the presence of fluorine is to reduce the citric acid solubility of the P_2O_5 content of the fertilizer and the elimination of the fluorine is therefore to the advantage of the producer of superphosphate. I know that calcination of the rock at fairly high temperatures in the presence of steam has been advocated and tried, but I have no experience of the methods of recovery of the fluorine so driven off. As the main impurity associated with any fluorine so recovered would probably be silica I would suggest to Mr. Gill that provided the silica content was below 3 per cent there would be a definite place for such recovered fluorine in the non-ferrous metals, glass, and enamel industries.

Whether the cost of such recovery could permit of direct competition with the products of the fluorspar industry is another matter and one which I am not in a position to answer. I would, however, consider that the fluorine so driven off would be in a condition to be more readily and profitably converted into fluorine salts, artificial cryolite, etc., than to be used to compete in open market with graded spar for fluxing purposes. This, however, is

**U.S. Dept. Agriculture Tech. Bull. 364.*

only an expression of opinion, with no actual data to back it, but I do agree with Mr. Gill that the recovery of such fluorine would undoubtedly be in the national interest of conservation.

Mr. F. Bice Michell's remarks show an appreciation of the problems encountered. The decision to use an impact crusher was based on the desire to obtain as much jig-size material as possible from the ore, coincident with the smallest production of fines. In other projects I had been favourably impressed with the performance of the Lightning machine and for this reason called for its incorporation in the flow sheet.

A certain loss was anticipated in discarding jig tails, but the need to recover a high proportion of the fluorspar for metallurgical purposes at jig size made it necessary to incorporate jigs in the circuit. Obviously, however, such a loss is another argument for expediting the introduction of the flotation concentrate into the metallurgical-grade field. These particular jigs were to be of the hutch-discharge type, some of which were already available. I agree with Mr. Michell on the advantage sometimes obtained by 'over sieve' jigging—*but not on friable material.*

Mr. Michell's reference to the use of 'table flotation' for fluorspar treatment is of interest, especially as he recognizes the difficulty in dealing with the 'fines' by such means. The fines, of course, are the snag and that is why I feel that flotation must constitute at least a stage in any modern fluorspar recovery plant.

In his contributed remarks Mr. Wood deals with the reagents employed in his test work.

The object of the thickener between the grinding and flotation units was to ensure a proper solids/water ratio in the flotation feed—a necessary precaution when dealing with such a high concentrate-forming ratio.

Mr. Norman Wynne's remarks are also of interest, especially in his comments on the physical condition of the spar in the Stanhopeburn lodes and the tendency to 'fines' production, which is so very marked and gives rise to such trouble. To attempt gravity treatment on an ore which runs 50 per cent *minus* 60-mesh before it sees a crushing plant is asking too much and yet only gravity methods have been employed in this district. I quite agree with Mr. Wynne that jigging of the Stanhopeburn ore was not attractive, but the mill was designed for ore from other sources also.

With regard to the use and treatment of flotation concentrates, the possibility of employing ground fluorspar, even when briquetted, in a blast furnace is something outside my experience; I think

Mr. Wynne means an open-hearth or electric steel furnace, in which the major part of the metallurgical-grade spar is consumed. Whether the flotation concentrates are briquetted using any of the various means of bonding known to those experienced in this type of work or whether they are sintered and agglomerated—a method of treatment long in existence and which is employed daily to treat thousands of tons of ore or concentrates on the Dwight-Lloyd and other sintering furnaces throughout the world—is simply a matter of individual preference and local facilities.

Mr. Dawson mentions the advantages to the locality in giving employment if such a scheme could be put through, and his reference to the ancilliary problem of barytes recovery by flotation is opportune. Mr. Dawson has recently carried out tests on barytes ores from various localities and has been able to obtain by flotation quite useful results, both with respect to grade and recovery.

I agree with Mr. Hohnen that it is a pity that this scheme remains a 'paper project'. Had it gone through its value might not have been measured only by its own economic balance sheet, but, as Mr. Hohnen rightly says, in serving as a blue print it might have had a considerable influence on the reawakening and development of the home non-ferrous mining industry.

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JULY 11TH, 1946.

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[NOTE.—*The Report of the Proceedings at the Annual General Meeting held on Thursday, May 16th, 1946, including the Presidential Address by Mr. G. F. Laycock, the Report of Discussion at the Seventh General Meeting of the last Session, Contributed Remarks and Authors' Replies to Papers previously submitted, are also attached hereto, together with the following new Paper :*

Anglo-American Magnesium Production.
By P. L. TEED, *Member.*]

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VISIT TO BILLINGHAM.

By the courtesy of Imperial Chemical Industries, Limited, Billingham Division, and Mr. G. Eland Stewart, Assoc. Inst. M.M., Agent and Manager of the Billingham Mine, permission has been given for a party of members of the Institution to visit the anhydrite mine, sulphuric acid plant, and sulphate plant at Billingham, Co. Durham, on Wednesday, 25th September, 1946.

Owing to limited hotel accommodation and the difficulty of conducting a larger party underground, the number of the party is restricted to 25.

Arrangements are being made as follows :—

Tuesday, September 24th : Party arrives at Redcar; dinner at Coatham Hotel, followed by a short talk on the mine and chemical works by Mr. Eland Stewart, Dr. Dunn, and Mr. Child, of I.C.I., Ltd.

Wednesday, September 25th : 9 a.m., leave Redcar by private bus; 10 a.m. to 12.30 p.m., visit to mine (scraper loading, face drilling and blasting, scaffold repairs on high roof, stoper driller); 12.30 p.m., visit to sulphuric acid plant. Lunch at Billingham; 3 p.m., visit to sulphate plant. Tea at Billingham, after which questions on mine and sulphuric acid and sulphate plants will be answered by the Section Managers; 6 p.m., return to Redcar for dinner.

Thursday, September 26th : Breakfast at Redcar; party then disperses.

The approximate cost of accommodation and meals at Redcar and transport from Redcar to Billingham will be £2 per person. In order to keep the party compact and enable the full programme to be carried out, the party will stay at the hotels at Redcar where accommodation has been reserved. Members who wish to join the party are requested to write to the Secretary of the Institution as soon as possible.

A short account of the Billingham mine, by Mr. Eland Stewart, will be published in the September issue of the *Bulletin*.

MEMBER OF COUNCIL FOR MALAYA.

Under the provisions of Section IV, Clause 7, of the By-Laws, the Council at their Meeting on June 13th, 1946, elected Mr. J. D. Mead as Member of Council for Malaya to fill the vacancy caused by the resignation from the Council of Mr. J. H. Rich.

Mr. Rich, Director and General Manager of Tronoh Mines, Ltd., and Ayer Hitam Tin Dredging, Ltd., has been Member of Council for Malaya since 1943, but has resigned from the Council in order that their representative might be someone actually resident in Malaya. Mr. Mead is senior partner in the firm of Messrs. Osborne & Chappel, Ipoh, Perak.

BALLOTING LIST FOR ELECTION OF MEMBERS OF COUNCIL.

The Council wish to draw the attention of Members and Associates to Section IV, clause 6, of the By-Laws, which contains the following provision : ' The Council shall receive the name of any Member or Members, submitted in writing by any Member or Associate previous to November 1st in any year, and shall decide by ballot upon the inclusion thereof, or otherwise, in the balloting list '.

Any Member or Associate who wishes to suggest a name for inclusion in the balloting list for the election of Members of Council for the Session 1947–48 is requested to notify the Secretary of the Institution before November 1st, 1946.

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 May 9th, 1946 :—

To MEMBERSHIP—

Hutchin, Frank (*Camborne, Cornwall*).
Macpherson, Eric Ogilvy (*Wellington, New Zealand*).

To ASSOCIATESHIP—

Bath, Silas James (*Camborne, Cornwall*).
Buchanan, Douglas (*Dunedin, New Zealand*).
Cullen, Matthew Graeme (*Randfontein, Transvaal*).
Delmé-Radcliffe, Peter Audley (*Aldbourne, Wiltshire*).
Haddon, Michael de Fenton (*Johannesburg, Transvaal*).
Kay, Francis Robert (*Chalfont St. Peter, Buckinghamshire*)
Monro, Donald Francis (*London*).
Nicholls, William (*Accra, Gold Coast*).
Pullar, William Alexander (*Dunedin, New Zealand*).
Wilson, Charles James White (*Gatooma, Southern Rhodesia*).

The following have applied for admission into the Institution since May 9th, 1946 :—

To MEMBERSHIP—

Neustatter, Alfred (*London*).

To ASSOCIATESHIP—

Harvey, George (*Jedda, Saudi Arabia*).
Morgan, Thomas Oliver (*Mwanza, Tanganyika Territory*).
O'Connor, Eugene Roderic Carrington (*Maraisburg, Transvaal*).
Pearson, George Matthew (*Nova Lima, Brazil*).
Rowe, James Henry (*Oruro, Bolivia*).
Sheridan, Gerald Derek (*Avoca, Eire*).
Williams, Samuel Duncan (*West Wittering, Sussex*).

TRANSFERS AND ELECTIONS.

The following have been transferred (subject to confirmation in accordance with the conditions of the By-Laws) since May 9th, 1946 :—

To MEMBERSHIP—

Farrington, John Leonard (*Royal Engineers*).
Park, James Williamson (*Tarkwa, Gold Coast*).
Tasker, Richard Beaumont (*Heidelburg, Transvaal*).

To ASSOCIATESHIP—

Cowlin, William Ronald (*Royal Engineers*).
Eisner, Kurt (*Wembley, Western Australia*).
Job, Arthur Leslie (*Camborne, Cornwall*).
Nicholson, George Lionel (*Royal Engineers*).
Pratt, Norman (*Thames, New Zealand*).
Wright, John Richard (*Randfontein, Transvaal*).

TRANSFERS AND ELECTIONS—*continued.*

The following have been elected (subject to confirmation in accordance with the conditions of the By-Laws) since May 9th, 1946 :—

To Membership—

Naylor, Theodore Rufus (*Royal Air Force*).
Poussin, Jean M. J. de La Vallée (*Mpanda, Tanganyika Territory*).
Talbot, Harold Leroy (*Kitwe, Northern Rhodesia*).

To Associateship—

Allen, Lawrence Wilcock (*Chingola, Northern Rhodesia*).
Bolton, Cedric Michael Grey (*Maidstone, Kent*).
Christie, John (*Kitwe, Northern Rhodesia*).
Foster, Douglas Frank (*Doncaster, Yorkshire*).
Lamba, Bhag Singh (*Khewra, India*).
Paver, Gordon Lyall (*Johannesburg, Transvaal*).

To Studentship—

Daniel, Kenneth Edward (*Birmingham, Warwickshire*).
Earl, Stephen James (*London*).
Edwards, Richard Charles John (*Bushey, Hertfordshire*).
Follows, Edward Arthur (*Birmingham, Warwickshire*).
Goodwin, James (*Derby*).
Quarm, Thomas Alfred Arthur (*London*).
Rao, C. E. Narayana (*Basavangudi, South India*).

NEWS OF MEMBERS.

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

Mr. J. B. Alexander, *Associate*, has left England on his return to the Geological Survey, Malaya.

Mr. H. E. Allen, *Member*, has left the Gold Coast for South Africa on leave, and expects to return to London in the autumn.

Mr. G. P. Anderson, *Associate*, has been demobilized and has returned to his former appointment at Barberton, Transvaal.

Mr. D. G. Armstrong, *Associate*, has returned to England from the Gold Coast.

Mr. G. C. Barnard, *Member*, has returned to Nairobi via Southern Rhodesia and the Northern Rhodesian copper belt after a five months' visit to Johannesburg.

Mr. R. D. F. Barry, *Associate*, has been transferred from A. O. (Nigeria), Ltd., to A. O. (Malaya), Ltd., and has arrived at Selangor.

Captain P. C. M. Bathurst, R.E., *Student*, has returned from Burma, and expects to be demobilized in the autumn.

Mr. R. G. Bergman, *Associate*, formerly a prisoner of war in Manila, has returned to New York City.

Mr. J. G. Berry, *Associate*, has joined the staff of the Indian Copper Corporation, Ltd.

Mr. H. O. Berryman, *Associate*, has left England to join the Mines Department, Tanganyika Territory.

Mr. J. M. Birkbeck, *Student*, has been demobilized and has returned to England.

NEWS OF MEMBERS—*continued.*

Mr. C. W. F. Bond, *Associate*, has left England for the Gold Coast.

Mr. R. K. Boright, *Associate*, has been released from the South African Forces and has returned to Springs, Transvaal.

Mr. A. V. Bradshaw, *Student*, is leaving England to join the staff of Messrs. Mason & Barry in Portugal.

Mr. R. T. Brandt, *Associate*, is now at Springs Mines, Transvaal.

Mr. Leonard G. Brown, *Associate*, has left the Directorate of Opencast Coal Production and has joined the staff of African Manganese Co., Ltd., Gold Coast.

Mr. I. F. Cairns, *Associate*, has been demobilized and has returned to Johannesburg.

Mr. R. L. Carlton, *Student*, has been demobilized on his return from Burma.

Mr. J. Chynoweth, Junior, *Student*, has left England for Malaya to rejoin the staff of Messrs. Osborne & Chappel.

Mr. H. A. Cochran, *Member*, has returned to Scotland on leave from Nigeria.

Mr. W. H. Collins, *Associate*, has returned to England from Nigeria.

Mr. F. F. Considine, *Associate*, has joined the staff of Messrs. Mason & Barry at Mina de San Domingos in Portugal.

Mr. W. S. Coombes, *Associate*, has left England to return to Malaya.

Major W. R. Cowlin, R.E., *Associate*, has returned to India for demobilization, after which he will resume his appointment as underground agent with Champion Reef Gold Mines of India, Ltd.

Dr. D. A. Bryn Davies, *Associate*, has arrived in Jamaica from England.

Mr. A. W. H. Dean, C.I.E., M.C., *Associate*, Chairman of Delhi Improvement Trust, has been made a Knight Bachelor in the King's Birthday Honours.

Mr. C. C. Dell, *Student*, has left England to join the staff of Mufulira Copper Mines, Ltd.

Mr. J. B. Dennison, *Member*, is returning to England from frica.

Mr. J. Descraques, *Associate*, has left Malaya on his retu ance.

Dr. J. V. N. Dorr, *Member*, has been in Germany ting developments in hydro-metallurgy, and expects to return d States via London about the middle of July.

Mr. A. S. Dunstan, *Student*, has left England for Nigeria to ta post with the Kaduna Syndicate, Ltd.

Mr. W. J. Dyack, *Associate*, has taken up the appointment of Engineer to the Nairobi District Council.

Mr. S. J. Earl, *Student*, has left England to join the staff of Frontino Gold Mines, Ltd.

Mr. D. Eastmond, *Associate*, has left England for the Transvaal.

Major J. C. Erskine, R.E., *Associate*, has returned to England from Palestine for demobilization.

Major J. S. Everitt, *Student*, has received his majority in Queen Victoria's Own Sappers and Miners.

Mr. W. E. Everitt, *Associate*, has left England to return to Inspector of Mines, Malaya.

NEWS OF MEMBERS—*continued.*

Mr. J. L. Farrington, *Member*, has been released from the Army and will shortly leave England for Nigeria, where he expects to remain for some weeks.

Captain F. A. Feilden, R.E., *Student*, has joined the R.E. Branch, Headquarters, Rhine Army.

Mr. D. A. Gee, *Associate*, formerly an internee in Malaya, has arrived in England, and expects to rejoin the staff of Gopeng Consolidated, Ltd., later in the year.

Mr. S. Gray, *Student*, has joined the staff of the Caribbean Petroleum Co., Venezuela.

Mr. F. R. H. Green, *Associate*, has arrived in England from British Guiana on leave, prior to taking up his new appointment as Chief Inspector of Mines, Sierra Leone.

Mr. D. W. J. Grey, *Member*, has arrived in England from Brazil.

Mr. T. Haden, *Associate*, who was interned by the Japanese, is now recuperating in Batavia. He hopes to return to his former post when political conditions allow.

Mr. G. F. Hatch, *Associate*, has joined the staff of Roan Antelope Copper Mines, Ltd. at their Johannesburg office.

Mr. S. Haymes, *Associate*, has left India for England on his appointment as manager of British Guiana Consolidated Goldfields, Ltd.

Mr. H. C. Herbert, *Associate*, has arrived in England on furlough from India.

Mr. F. B. Higgins, *Associate*, Chief Inspector of Mines, Gold Coast, was awarded the O.B.E. in the King's Birthday Honours.

Mr. N. A. B. Hill, *Associate*, has left England to join the staff of the Indian Copper Corporation, Ltd.

Mr. W. E. Hosking, *Member*, has arrived in England on his return from India.

Mr. L. G. Hutchison, M.C., *Member*, has left London for the Gold Coast, and expects to be away for a year.

Dr. F. T. Ingham, *Associate*, has left England on his return to Malaya.

Mr. G. R. Jones, *Associate*, has joined the staff of the General Sandur Mining Co., Ltd., South India.

Mr. J. A. Kennedy, *Associate*, has left England on his appointment to the Metallurgical Branch, Economic Division of the Control Commission for Germany.

Mr. P. F. F. Lancaster-Jones, *Student*, has been released by the Army in order to continue his studies at the Royal School of Mines.

The late Flight-Lieutenant R. A. M. Lemmon, R.A.F.V.R., *Student*, is reported to have been mentioned in despatches for distinguished service.

Mr. R. I. Lewis, *Associate*, has arrived in England from Portugal.

Mr. Ben Lightfoot, *Associate*, Director of the Geological Survey, Southern Rhodesia, has been awarded an O.B.E. in the King's Birthday Honours.

Mr. C. F. Lloyd-Jones, *Associate*, expects to be in England in the autumn on leave from Saudi-Arabia.

Mr. G. B. Mackenzie, *Associate*, has left England on his appointment as general manager of the Marampa mines of the Sierra Leone Development Co., Ltd.

NEWS OF MEMBERS—*continued.*

Mr. J. B. Mackie, *Associate*, has returned to Malaya from New Zealand, to resume his work in the Mines Department.

Mr. R. G. W. Mackilligin, M.C., *Student*, has left England to join the staff of United British Oilfields of Trinidad.

Mr. P. B. Marriott, *Associate*, has left England to rejoin the staff of Messrs. Osborne & Chappel at Ipoh, Malaya.

Mr. G. Musgrave, *Member*, consulting engineer to the Rhodesia Chrome Mines, Ltd., is also Chairman of the National Industrial Council of the Mining Industry of Southern Rhodesia, and Chairman of the Industrial Development Commission.

Mr. W. Nicholls, *Student*, has returned to England on leave from the Gold Coast.

Mr. G. L. Nicholson, *Associate*, has arrived in England from Italy on demobilization leave.

Mr. C. Maxwell Norman, *Member*, has been appointed general manager of New Goldfields of Venezuela, Ltd., and has recently paid a short visit to England.

Mr. H. J. D. Penhale, *Member*, is now in England on leave from Fernando Po.

Dr. T. Pickering, *Associate*, has now been released from the Army on his return from India.

Mr. R. F. Powell, *Associate*, having terminated his engagement as consulting metallurgist to Eastern Transvaal Consolidated Mines, Ltd., is now at Gwelo, Southern Rhodesia, as technical adviser to National Chemical Products (Germiston), Ltd.

Mr. W. Prachner, *Student*, has arrived in Trinidad to join the staff of the United British Oilfields of Trinidad, Ltd.

Mr. H. H. Robottom, *Associate*, has arrived home in England from India.

Mr. H. C. Robson, *Member*, has retired from India and is at present in England.

Mr. T. A. Rogers, *Student*, is at present acting mine superintendent, Emperor Gold Mining Co., Ltd., Fiji.

Mr. J. A. Royce-Evans, *Student*, has returned to Gold Coast Main Reef, Ltd., from England.

Mr. J. Sanderson, *Member*, has left England to resume his duties with the Malayan Mines Department.

Mr. F. G. Sharp, *Associate*, is returning to England from Mysore on furlough.

Mr. W. E. Sinclair, *Associate*, has been appointed general manager to the Cape Asbestos Co., Ltd.,

Mr. C. N. Smith, *Associate*, has accepted an appointment as field manager to Messrs. Le Grand, Sutcliff and Gell, Ltd.

Mr. A. J. G. Smout, J.P., *Member*, has been made a Knight Bachelor in the King's Birthday Honours.

Mr. Jack Spalding, *Associate*, expects to return in mid-July from a short visit to the Kolar Gold Field.

Mr. E. Stuart, *Associate*, has left England for Malaya.

Mr. E. H. Sworder, *Associate*, has been appointed district metallurgist of the Mines Department in the Bulawayo Area, Southern Rhodesia.

NEWS OF MEMBERS—*continued.*

Mr. R. M. Thomas, *Student*, has returned to England from Nigeria.

Mr. J. W. Thorburn, *Associate*, has been appointed mines superintendent, Mount Isa Mines, Ltd., Queensland.

Mr. F. E. Thurston, *Associate*, has left the Gold Coast for Chile.

Mr. J. J. Timmins, *Associate*, has left England to take up a position in Kenya.

Mr. C. W. Walker, *Associate*, has left England to join the staff of the Consolidated African Selection Trust, Ltd., Gold Coast.

Mr. E. A. Walker, *Member*, expects to arrive in England shortly on leave from India.

Mr. A. J. Walton, *Member*, will leave South Africa in July for a visit to England.

Mr. E. W. Warner, *Associate*, has arrived in England on leave from the Gold Coast.

Mr. H. J. R. Way, *Associate*, has returned to England on leave from the Swaziland Geological Survey.

Mr. W. Broadhead Williams, *Associate*, has left the staff of Geita Gold Mining Co., Ltd., to take up an appointment under the Government of Tanganyika Territory.

Mr. A. S. W. Wood, M.C., *Associate*, has left England for Sierra Leone.

Mr. J. R. Wright, *Associate*, has been released from the South African Engineer Corps and has returned to Randfontein Estates Gold Mining Co., Ltd.

ADDRESSES WANTED.

E. A. Banning.	A. W. Eyre.	James Russell.
J. S. Burns.	A. McCall.	A. E. Upfold.
L. W. Elsum.	E. I. Robinson.	

OBITUARY.

Lewis Percy Cazalet died on April 25th, 1946, at Warmbaths, Transvaal, at the age of 73. He received his mining education at the Camborne School of Mines from 1889 to 1893, where he graduated and gained two silver medals. For the following two years he was employed underground and on the reduction works at Knight's Tribute and Van Ryn Gold Mines Estate, Ltd., Witwatersrand, becoming cyanide manager in 1895. A year later he left to do private reporting work in the Witwatersrand, and in 1897 was appointed surveyor to the Durban-Roodepoort Gold Mining Co., Ltd., and Vogelstruis Estates and Gold Mines, Ltd. He transferred in 1898 to a similar position with Nourse Deep, Ltd., and after a year was made acting manager and, later, manager. On the amalgamation of Nourse Deep with Henry Nourse Gold Mining Co., Ltd., and South Nourse, Ltd., he retained the managership, remaining there until 1910. During the South African War he served for two years as captain with the 1st Batt. Railway Pioneer Regt., and until 1910 was a captain in the Rand Rifles. From 1904, Mr. Cazalet had been a member of the Commission of Examiners for Mine Overseers and Mine Managers' Certificates in South Africa. He was appointed consulting engineer at Johannesburg to the Central Mining-

OBITUARY—*continued.*

Rand Mines group in 1910, a position which he held for ten years, and in 1920 began independent consulting work in the Transvaal. From 1934 to 1938 he was engaged by Johannesburg Gold Mining Corporation, Ltd., as consulting engineer, and then took up the appointment of manager to Village Main Reef Gold Mining Co. (1934), Ltd. He had retired from professional work at the time of his death, having started farming at White River. He was the first president of the Association of Mine Managers of the Transvaal, and of the Associated Scientific and Technical Societies of South Africa.

Mr. Cazalet was elected a Member of the Institution in 1914.

John Henry Cordner-James died after a short illness at his home at Aldeburgh, Suffolk, on April 20th, 1946, at the age of 88. He was trained from 1874 under his father, Mr. A. T. James, of Redruth, Cornwall, manager of Wheal Basset and South Wheal Francis mines, and left England in 1877 on his appointment as assistant manager and surveyor to New Quebrada Land and Copper Co., Ltd., and in 1880 was engaged in examining the mines and forests of the Gustav and Carlberg Copper Co., Sweden. From 1881 to 1883 he held the appointment of general manager of the Indian Trevelyan Gold Mining Co., Ltd., and in 1884 returned to the New Quebrada Co. as professional adviser. For six months in the following year Mr. Cordner-James held the position of general manager of the New Chile Gold Mining Co., Ltd., in Venezuela, but returned to England and set up in practice in London as consulting engineer. In 1888 he went to the Transvaal as general manager of the United Pioneer Gold Mining Company at Barberton, and from 1890 to 1891 was engaged as consulting mining and metallurgical engineer in Johannesburg. On his return to England in 1891 he resumed practice in London as senior partner in the firm of James Brothers.

Mr. Cordner-James was elected to Membership of the Institution in 1899, and served on the Council almost continuously from 1901 to 1920.

Colonel Arthur Smith Dwight, D.S.O., died at Hobe Sound, Florida, on April 1st, 1946, at the age of 82. He was born at Taunton, Massachusetts, and graduated in 1882 from the Polytechnic Institute of Brooklyn, New York, and three years later obtained the degree of Engineer of Mining at Columbia University. In 1914 he received the honorary degree of M.Sc. at Columbia, and, in 1929, was made an honorary D.Sc. He was a trustee of that University and of the Brooklyn Polytechnic Institute.

On leaving Columbia in 1885 he went to Pueblo, Colorado, and for thirteen years was associated with the Colorado Smelting Co., beginning as assayer and rising to the position of superintendent. He then assumed charge of smelting operations at Pueblo and Leadville, Colorado, at El Pasco, Texas, and at Argentine, Kansas, and from 1900 to 1906 was at San Luis Potosi and Cananea, Mexico. In the autumn of 1906 he returned to New York and in partnership with Mr. R. L. Lloyd began the consulting practice of Dwight & Lloyd. He and Mr. Lloyd developed and patented the well-known process for the roasting and agglomeration of fine ores. He was president of Dwight & Lloyd Sintering Co. and of Dwight & Lloyd Metallurgical Co., and was secretary of Hobe Sound Co.

OBITUARY—*continued.*

During the 1914–1918 war he was a member of the engineering committee which co-operated with the U.S. War Department in organizing the Engineer Officers Reserve Corps, and was one of the first civilians to be commissioned in the Reserve and to enter active service. As major in the 1st Reserve Engineer Regiment from January, 1917, he played an important part in recruiting and training that body, which sailed for France in July, 1917, as the 11th Engineers, the first unit of the A.E.F. in action in France. Major Dwight served in France for over 22 months, on the British front for nine months, commanding the 1st Batt., 11th Engineers. He later was given special duties as metallurgical consultant to French companies and was made engineer salvage officer, A.E.F. He left the Army in May, 1919, with the rank of lieutenant-colonel, and with the awards of D.S.O. (Britain), citation by General Pershing of the U.S. Army, and the Order of the Purple Heart, and was made Chevalier de la Légion d'Honneur (France). He was colonel in the U.S. Reserve at the time of his death, and had been chairman of the minerals advisory committee to the War Department.

Col. Dwight was elected to Membership of the Institution in 1909, but in 1921 was made an Honorary Member on the occasion of the visit to England in June of that year of the Delegation from American Engineering Societies of which he was vice-chairman, as an expression of enduring friendship and of appreciation of the great achievements of American engineers; and in recognition of Col. Dwight's personal services in the advancement of mining and metallurgical practice. He was a past-president and 'James Douglas' medallist of the American Institute of Mining and Metallurgical Engineers and a past-president of the Society of American Military Engineers.

Albert Lucas Entwistle died suddenly on April 29th, 1945, at the age of 67. He was born at Bury, Lancashire, and from 1893 to 1895 was an articled pupil to Mr. T. J. Hutchinson, of Manchester, public analyst for Bury, and for the three following years was a student of metallurgy and chemistry at Owens College, Victoria University, Manchester. In 1899 he was appointed metallurgical chemist and assayer at Sulphides Reduction (New Process), Ltd., Llanelly, and when the works closed down in 1901 took up the appointment of chemist for the North of Ireland Paper Co., Ltd., a position he held until his return in 1903 to employment with Sulphides Reduction, Ltd., at their new works at Landore, Swansea. During 1904 he accompanied Mr. G. T. Holloway on a short visit to Angoulême, France, reporting on a process for treating refractory lead and zinc sulphide ores. He joined the staff of Venesta, Ltd., of Poplar, London, in 1905, also as metallurgical chemist, later taking a temporary post as assistant assayer at the Royal Mint, London. Shortly afterwards, in 1907, he went to Canada on his appointment as first assistant assayer at the Royal Mint, Ottawa, and in 1912 was put in charge of the refinery; after a period in an acting capacity he was, in 1922, appointed chief chemist and assayer, the position he still held at the time of his death, 23 years later.

Mr. Entwistle was elected to Studentship of the Institution in 1903, and was transferred to Associateship in 1907 and to Membership in 1917. He was also a Member of the Canadian Institute of Mining and Metallurgy.

OBITUARY—*continued*.

George Vincent Everitt died suddenly at Ootacamund, India, on February 26th, 1946, at the age of 60. He took a course in metallurgy at the Camborne School of Mines in 1906, and in January of the following year left for India to join the staff of Champion Reef Gold Mining Co., Ltd., for Messrs. John Taylor & Sons, by whom he was employed during the whole of his career.

For the first three years he was assistant in the mill, and from 1910 to 1917 was assistant cyanide officer, and then held the position of assistant reduction officer at Ooregum gold mine for two years. In November, 1920, he joined the Mysore Gold Mining Co., Ltd., as cyanide officer and analytical chemist, and in May, 1926, was appointed chief assistant cyanide chemist, and chief cyanide officer and analytical chemist in March, 1932. He left in 1934 on his promotion to the position of chief metallurgist at Champion Reef gold mine, in which capacity he served until his retirement in February, 1945, owing to ill health. He went with his wife to the Nilgiri Hills in the hope that his health would benefit but it did not improve.

Mr. Everitt was elected an Associate of the Institution in 1922.

Frederick William Jackson is reported to have died of cholera in Burma in 1942 at the age of 57. From 1910 to 1912 he was smelter shift boss for Burma Corporation, Ltd., at Namtu, Northern Shan States, and was then employed by Burma Oil Co. for two years on oil pipe-line construction. He had been sergeant in the East Surrey Regt. from 1904 to 1910, and during the 1914-1918 war served with the Northampton Regt. as sergeant in Mesopotamia and the N.W. Frontier. He returned to Burma on demobilization in 1919 and until 1925 held the position of mine foreman at the Hermyingyi mine, Tavoy, Lower Burma, for Burma Finance and Mining Co., becoming superintendent of the mine under Consolidated Tin Mines of Burma, Ltd., in 1925. In February, 1937, he took over their Bwabin property, also as mine superintendent, and in 1941 was engaged by Kanbauk Mines, Ltd., Tavoy. Mr. Jackson was in Burma at the time of the Japanese invasion, and no news of him had been received since.

He was elected an Associate of the Institution in 1938.

William Alexander MacLeod died on May 7th, 1946, at Harrow-on-the-Hill, Middlesex, at the age of 71. He was born in New Zealand, and studied at the University of Otago, obtaining the B.A. and B.Sc. degrees and the Associateship in both mining and metallurgy of the Otago School of Mines in 1897. In the following two years he gained general experience in coal and metal mining, milling and cyaniding, and for eighteen months of that time held the position of assistant director of the Thames School of Mines in New Zealand, where he was in charge of the ore-testing plant. From 1899 to 1901 he was in charge of the provisional mining course at Hobart University, Tasmania, and then became director of Charters Towers School of Mines, North Queensland, for three years. He left the School in 1903 to become mine manager of Brilliant Extended Gold Mining Co., Ltd., and a year later was made general manager. In 1912 Mr. MacLeod was engaged by Messrs. Bewick, Moreing & Co. for work in Kalgoorlie, Western Australia, and in 1917 he came to the head office of the company in London, where he worked until his death.

OBITUARY—*continued*.

Mr. MacLeod wrote a number of articles for the Australian technical press, and contributed two papers to the *Transactions* of the Institution—'The surface condenser in mining power plant' (vol. 19, 1919–20) and 'The internal combustion engine: some modern types and their application to mining' (vol. 36, 1926–7). He was elected a Member of the Institution in 1909.

Herbert Brantwood Maufe died suddenly in London on May 8th, 1946, at the age of 66. He began his professional training as an extra-mural student in geology at Yorkshire College, Leeds, in 1896 and 1897, and went to Cambridge University from 1898 to 1901 as an exhibitioner and scholar of Christ's College. He obtained his B.A. degree in 1901 and in September of that year joined the Geological Survey of Scotland as geologist. In that capacity he was engaged in the Geological Survey of Ireland in 1903 and 1904, and worked for the Colonial Office in British East Africa from 1905 to 1906. In September, 1910, he was appointed Director of the Geological Survey of Southern Rhodesia, and held this position until 1935.

Mr. Maufe was awarded the Lyell Fund in 1909 by the Geological Society of London and the Lyell Medal in 1930, and in 1934 was awarded the Draper Memorial Medal for distinguished work in South Africa by the Geological Society of South Africa, of which body he had been a member of Council, and president for the session 1918–19. He was elected to Membership of the Institution in 1931.

Horace George Nichols died suddenly on July 30th, 1945, in British Columbia. He had for some time suffered the partial paralysis of one side. He was a student at the Royal School of Mines from 1886 to 1899, and graduated with an Associateship in Metallurgy. He became assayer to the New Morgan gold mine in North Wales in 1890, then worked in South Africa from 1891 to 1894 as millman to Sheba Gold Mining Co., Ltd., with minor engagements in Swaziland. He first went to British Columbia in 1897 on his appointment as assistant superintendent to Hall Mines, Ltd., Nelson, and in 1899 became mill foreman to Ymir Gold Mines, Ltd. From 1901 to 1902 he held the position of assistant superintendent and acting superintendent to Standard Consolidated Mining Co., at Bodie, California. He then joined Republican M. & D. Co., California, as superintendent, and from 1904 was for two years manager of Aramecina Gold and Silver Mining Co., Ltd., and Transito Gold Mine, Ltd., in Honduras and London. Mr. Nichols returned to British Columbia in 1906 and took over the managership of Ymir gold mines until they were closed down in 1910, when he left for South Africa, later becoming resident engineer for Anglo-Siberian Co., Ltd., and Russo-Asiatic Corporation in Russia and Siberia. In 1913 he was partner in the firm Bainbridge, Seymour & Co., mine managers, of London, but during the 1914–1918 war served with the B.E.F. in France as captain in the Royal Engineers. On demobilization in 1919 he went to Mexico as general mines superintendent to Mazapil Copper Co., Ltd., and in 1921 settled in British Columbia, first in practice as consulting engineer in Vancouver, and in 1925 as government resident mining engineer in Central District.

OBITUARY—*continued.*

For the last twenty-two years of his life Mr. Nichols had been associate editor of *Canadian Mining Journal*, and two of his many previous technical writings were published by the Institution—' A method of settling slimes as applied to their separation from solution in cyanide treatment ' and ' The treatment of tin ores in Cornwall : a description of the Geevor mine ' (*Trans.*, vols. 17, 1907–8, and 23, 1913–14, respectively).

He was elected an Associate of the Institution in 1903 and transferred to Membership in 1907. He had also been a member of council and vice-president of the Canadian Institute of Mining and Metallurgy.

Harold Whittingham died suddenly at his home in Toronto on October 23rd, 1945, at the age of 62. After working as assistant estate agent in Sussex for two years, he began his mining career in Canada in May, 1907, as miner, timekeeper and surveyor's assistant at the Nipissing mine, Cobalt, Ontario, and a year later became underground surveyor in full charge. He left Cobalt in September, 1909, to attend the Michigan College of Mines, graduating in 1912 as B.Sc. and Engineer of Mines. He then joined Mr. Geo. H. Garrey, consulting geologist, Philadelphia, as assistant on geological examinations in Mexico for the American Smelting and Refining Co., and in November, 1913, went to Sardinia as manager of the Gennamari mine. On the outbreak of war in 1914 Mr. Whittingham joined the Royal Garrison Artillery, rising to the rank of major in 1916, when he was in command of the 71st Heavy Battery. He served continuously with the B.E.F. during the first three years of war, in France and Flanders, gaining a ' mention ', and at the end of 1917 was wounded and taken prisoner.

Mr. Whittingham returned to his position at Gennamari mines in March, 1919, but left a year later and worked privately on the South Wales coal-field, in connection with coal tar derivatives. From April to November, 1921, he was engaged on civil and mechanical engineering work in Leicestershire, but left England again early in 1922 on obtaining the position of manager of the Keeley silver mines, Silver Centre, Ontario, and two years later became manager of the Huronian Belt Co. Since that time he had been associated with Ventures, Ltd., from its inception, as secretary-treasurer, and was president of Falconbridge Nickel Mines, Ltd., Sherritt-Gordon Mines, Ltd., Michipicoten Iron Mines, Ltd., and Opemiska Copper Mines, Ltd. ; he was vice-president and director of many other mining companies. He was elected to Associateship of the Institution in 1919, and was transferred to Membership in 1924, and was also a member of the Canadian Institute of Mining and Metallurgy.

Cecil William Wicks has been officially recorded as having died on or about February 13th, 1942, as a result of the sinking of the S.S. *Giang Bee*. He received his mining training at Sydney University from 1920 to 1923, graduating with the degree of B.Eng. (Min.). He began his career at the North mine (British Section), Broken Hill, N.S.W., where he was assistant surveyor from January to October, 1924, and from then on was employed by Austral-Malay Tin, Ltd., and associated companies in Malaya. He first spent three years on the prospecting staff and was appointed assistant manager to Kampong Kamunting Tin Dredging, Ltd., in April,

OBITUARY—*continued.*

1928. He subsequently held the position of manager of Ulu Yam Tin, Ltd., for six months, and of Thabawleik Tin Dredging, Ltd., for the following six months, and was manager of Pungah Tin Dredging, Ltd., Siam, from 1930 to 1934. He was with Austral Malay Tin, Ltd., at Taiping, Perak, from 1934 until the Japanese occupation of Malaya.

Mr. Wicks was elected to Associateship of the Institution in 1931.

The Council regret to report the death of **George Ernest Collins**, *Member*, on May 4th, 1946. An obituary notice will appear in a later issue of the *Bulletin*.

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

The address of the Library is now 424 Salisbury House, London, E.C. 2, and books (excluding periodicals) may be borrowed by members on personal application or by post.

American Institute of Electrical Engineers: Year Book. New York: The Institute. 1946. 333 p.

American Institute of Mining and Metallurgical Engineers: Transactions. Vol. 161. Institute of metals division, 1945. 646 p. Vol. 162. Iron and steel division, 1945. New York: The Institute. 1945. 748 p.

American Society of Civil Engineers: Proceedings. Vol. 72, No. 2, Pt. 2, Transactions number 110, 1945. Lancaster, Pa: The Society. 1946. 1952 p.

Barium Minerals in England and Wales, by K. C. Dunham, and H. G. Dines, with contributions by T. Eastwood, J. V. Stephens, S. E. Hollingworth, W. Anderson, and J. R. Earp. London: Geological Survey. 1945. 149 p. (Gt. Britain. Geological Survey. Wartime pamphlet No. 46.)

British Standards:
No. 1085. 1946: Lead pipes, silver-copper-lead alloy. 11 p. 2s.
No. 61, Pt. 2, 1946: Screw threads for copper tubes. 14 p. 2s.
No. 882, 1944: Concrete aggregates and building sands from natural sources. 54 p. 5s.
London: British Standards Institution.

California. Division of Mines. Bulletin No. 132. California mineral production and directory of mineral producers for 1944. San Francisco: Division of Mines. Oct., 1945. 224 p.

Ceylon. Department of Mineralogy: Professional paper No. 2, 1944. Colombo: Govt. Printer. 1945. 44 p.

China. Geological Survey: Bulletin No. 35, April, 1942. Pehpei, Chungking: Geological Survey. 1942. 150 p. (In Chinese with English Summary.) (*Presented by the China Section of the Science Dept., British Council.*)

Colliery Manager's Examinations New Guide, 2nd ed. rev., by H. C. Harris. London: Griffin. 1945. 398 p. 15s.

Economic Mineral Deposits, by Alan M. Bateman. New York: Wiley. 1942. 898 p. 39s.

Environmental Warmth and its Measurement, by T. Bedford. London: H.M.S.O. 1946. 40 p. 10 charts. (Medical Research Council War Memorandum No. 17.) 2s. 3d.

LIST OF ADDITIONS TO THE JOINT LIBRARY—*continued.*

GEOLOGY OF CENTRAL AND SOUTHERN KANSU, by L. T. Yeh and S. C. Kwan. Pehpei, Chungkung: Geological Survey of China. Dec., 1944. 84 p. maps. (China. Geological Memoirs, Series A, No. 19. In Chinese with English Summary.) (*Presented by the China Section of the Science Dept., British Council.*)

GT. BRITAIN. MINISTRY OF FUEL AND POWER: FUEL EFFICIENCY BULLETIN NO. 43, THE FURNACEMAN'S MANUAL. London: The Ministry. 1946. 30 p.

INDUSTRIAL HEALTH RESEARCH BOARD: REPORT NO. 89. ARTIFICIAL SUNLIGHT TREATMENT IN INDUSTRY, by Dora Colebrook. London: H.M.S.O. 1946. 64 p. 1s.

INSTITUTION OF MECHANICAL ENGINEERS: PROCEEDINGS JAN.–DEC., 1945. VOL. 152. London. The Institution. 1945. 468 p.

INSTITUTION OF MINING ENGINEERS: TRANSACTIONS. VOL. 54. 1944–45. London: The Institution. 1945. 737 p.

MECHANICAL LOADING OF COAL UNDERGROUND, by Ivan A. Given. New York: McGraw-Hill. 1943. 397 p. 24s.

MICA POWDERS FOR USE IN PAINT, by A. A. C. Dickson. Derby: Wall Paper Manufacturers Ltd. 1945. 9 p. (*Presented by the author.*)

MINISTRY OF FUEL AND POWER: TESTING MEMORANDUM NO. 4. TEST AND CERTIFICATION OF THE FLAMEPROOF ENCLOSURE OF ELECTRICAL APPARATUS. London: H.M.S.O. 1946. 28 p. 6d.

NICKEL BULLETIN, VOL. 17. 1944. London: Mond Nickel Co. 1945. 208 p. (*Presented by the Mond Nickel Co.*)

NOTES ON MALARIA AND ITS CONTROL FOR PLANTERS AND MINERS, by G. Macdonald. London: Ross Institute. 1946. 61 p.

NOVA SCOTIA. DEPARTMENT OF MINES: ANNUAL REPORT ON MINES FOR 1945. Halifax: King's Printer. 1946. 320 p.

ONTARIO. DEPARTMENT OF MINES: ANNUAL REPORTS.
51st annual report being Vol. 51, Pt. 1, 1942. 282 p.
53rd annual report being Vol. 53, Pt. 4, 1944. 32 p.
53rd annual report being Vol. 53, Pt. 6, 1944. 55 p.
Toronto: Bowman. 1946.

PALAEOZOIC GEOLOGY OF THE WINDSOR–SARNIA AREA, ONTARIO, by J. F. Caley. Ottawa: Govt. Printer. 1945. 227 p. maps. (Canada. Bureau of Geology and Topography: Geological Survey Memoir No. 240.) 50 cents.

LIST OF ADDITIONS TO THE JOINT LIBRARY—*continued.*

REPORTS OF THE BRITISH AND COMBINED INTELLIGENCE OBJECTIVES SUB-COMMITTEES (B.I.O.S. AND C.I.O.S.) AND FIELD INFORMATION AGENCY, TECHNICAL (F.I.A.T.). Various reports have been presented by the Department of Scientific and Industrial Research, and are listed under the Index of Recent Articles.

SEVERN TUNNEL, ITS CONSTRUCTION AND DIFFICULTIES, 1872–1887, by Thomas A. Walker. London: Bentley. 1888. 188 p. diags. (*Presented by Lt.-Col. J. V. Ramsden.*)

TANGANYIKA. GEOLOGICAL DIVISION. Preliminary report on the mining geology of the Iramba-Sekenke Goldfield, by R. B. McConnell. Tanganyika: Geological Division. 1945. 22 p. (with geological map, scale 1: 10,000.) 2s.

TECHNICAL REPORT ON THE RUHR COALFIELD, VOL. 1, by a Mission from the Mechanisation Advisory Committee of the Ministry of Fuel and Power. London: H.M.S.O. 1946. 61 p. 3s.

UPSALA UNIVERSITY. GEOLOGICAL INSTITUTION: BULLETIN. Vol. 31. Upsala: The University. 1946. 404 p.

MAPS.

DYSON CREEK, WEST OF FIFTH MERIDIAN, ALBERTA. Geological Survey Map 827A. Scale: 1 in. = 1 ml. Ottawa: Bureau of Geology and Topography. 1945.

GREGG LAKE, WEST OF FIFTH MERIDIAN, ALBERTA. Geological Survey Preliminary Map 46–4A, and Paper 46–4. Scale: 2 in. = 1 ml. Ottawa: Bureau of Geology and Topography. 1946.

HONEYDALE, CHARLOTTE COUNTY, NEW BRUNSWICK. Geological Survey Preliminary Map 46–3. Scale: 1 in. = 1 ml. Ottawa: Bureau of Geology and Topography. 1946.

MCCONNELL CREEK, CASSIAR DISTRICT, BRITISH COLUMBIA. Geological Survey Preliminary Map 46–6A, and Paper 46–6. Scale: 1 in. = 2 ml. Ottawa: Bureau of Geology and Topography. 1946.

MIKANAGAN LAKE, WEST OF PRINCIPAL MERIDIAN, MANITOBA. Geological Survey Map 832A. Scale: 1 in. = 1 ml. Ottawa: Bureau of Geology and Topography. 1946.

ST. STEPHEN, CHARLOTTE COUNTY, NEW BRUNSWICK. Geological Survey Preliminary Map 46–2. Scale: 1 in. = 1 ml. Ottawa: Bureau of Geology and Topography. 1946.

TOWNSHIP OF TECK, DISTRICT OF TIMISKAMING, ONTARIO. Map No. 1945–1. Scale: 1 in. = 1,000 ft. Ontario: Dept. of Mines. 1945.

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.

INDEX OF RECENT ARTICLES—*continued.*

INDEX OF RECENT ARTICLES—*continued.*

COAL—*continued.*

MACHINE MINING.—Modern machine mining. G. S. Hyson.—*Min. Elect. Mech. Engr.*, Manchr., Vol. 26, May, 1946, pp. 412–16. 2s. 6d.

MINING METHODS — OPENCAST. — Opencast coal production in wartime. Kenelm Charles Appleyard and Gerald Curry.—*J. Instn. Civ. Engrs.*, Lond., Vol. 26, May, 1946, pp. 331–76. 2s. 6d.

MINING METHODS—OPENCAST.—Opencast coal; some aspects of production. P. H. McDouall.—*Mine & Quarry Engng.*, Maidenhead, Vol. 11, June, 1946, pp. 133–38. 1s.

MINING METHODS — OPENCAST — ILLINOIS.—Planned stripping helps modern units produce efficiently.—*Coal Age*, Albany, N.Y., Vol. 51, April, 1946, pp. 81–85. 35 cents.

PREPARATION — CLEANING — NEW SOUTH WALES.—New coal cleaning plant at Newcastle steel works. W. H. Wainwright and J. S. Ratcliffe.—*Chem. Engng. Min. Rev.*, Melb., Vol. 38, March 11, 1946, pp. 189–93. 1s.

PREPARATION — WASHING — U.S.A. — Modern washeries culminate T.C.I. preparation evolution. W. S. Springer and Wallace J. McRoy.—*Coal Age*, Albany, N.Y., Vol. 51, April, 1946, pp. 99–104. 35 cents.

RESEARCH — GERMANY. — Interrogation of Dr. Gunter Spengler, Munich, formerly of Institute for Coal Research German Technical High School, Prague.—B.O.I.S. Final Report, No. 368, 1945, 16 p.

SAMPLING — ELECTRON MICROSCOPIC METHODS.—Determination of the size distribution of fine coal particles by the electron microscope. J. T. McCartney.—*U.S. Bur. Min., Rep. Invest.* No. 3827, Sept., 1945. 11 p.

SCREENING — CUTTINGS. — The fineness of cuttings in underground workings. R. L. Brown and B. R. Lawton.—*B.C.O.R.A. Quart. Bull.*, Lond., Sep., 1945, pp. 9–21.

SUPPORT.—Rock pressure and roof support. Pt. 4. Z. S. Beryl.—*Colliery Engng.*, Lond., Vol. 23, June, 1946, pp. 149–51. 1s.

SUPPORT — FILLING — PNEUMATIC. — Pneumatic stowing at Bullcroft main colliery. J. Hunter.—*Trans. Instn. Min. Engrs.*, Lond., Vol. 105, Pt. 3, Dec., 1945. 6s.

SYSTEMS—BORD AND PILLAR.—Bord and pillar mining (a historical survey of the development of this system of coal mining). Pts. 5 and 6. A. Roberts.— *Colliery Engng.*, Lond., Vol. 23, 1946; May, pp. 124–26; June, pp. 153–59. 1s. ea.

SYSTEMS—HORIZON MINING.—Horizon mining with special reference to methods applied in the Netherlands. L. L. Van Praag. —*Colliery Guard.*, Lond., Vol. 172, June 14, 1946, pp. 799–805. 10d.

COAL—*continued.*

TESTING — BURNING CHARACTERISTICS.—Burning bitumous and sub-bitumous coals on an anthracite, ash-removal-type domestic stoker. H. F. Yancey and K. A. Johnson.— *U.S. Bur. Min., Rep. Invest.* No. 3849, Dec., 1945, 14 p.

TRANSPORT — SPIRAL CHUTES. — Spiral chutes for vertical coal transport. M. G. Driessen.—*Trans. Instn. Min. Engrs.*, Lond., Vol. 105, Pt. 4, Jan., 1946, pp. 156–66. 6s.

U.S.A. — American commentary. A. G. Douthwaite, *comp.*—*Trans. Instn. Min. Engrs.*, Lond., Vol. 105, Pt. 1, Oct. 1945, pp. 25–47. 6s.

WESTERN AUSTRALIA—COLLIE.—The Collie coalfield, its problems and its economic importance. R. C. Wilson.—*W. Aust. Dep. Min. Rep.* 1944, Perth, 1945, pp. 33–50. (Maps, diags.)

YORKSHIRE — BARNSLEY. — New workings at Bentley Colliery.—*Colliery Engng.*, Lond., Vol. 23, 1946; May, pp. 114–20; June, pp. 141–48. 1s. ea.

COPPER.

METALLURGY — GERMANY. — Copper smelting and refining, particularly from alloy scrap and residues at Zinnwerke Wilhelmsburg G.m.b.H. — B.I.O.S. Final Report No. 25, Item No. 21, 1945, 24 p.

METALLURGY — GERMANY. — Copper smelting and refining, together with related metallurgical activities at Nord Deutsche Affinerie, Hamburg.—B.I.O.S. Final Report No. 26, Item No. 21, 1945, 21 p.

MEXICO.—Copper from Mexico. Roger V. Price.—*Compr. Air. Mag.*, N.Y., Vol. 51, May, 1946, pp. 127–29. 35 cents.

ONTARIO — COCHRANE. — Some copper properties in Robb, Jamieson and Godfrey Townships. S. A. Ferguson.—*Ont. Dep. Min. 53rd Ann. Rep.*, Toronto, Vol. 53, Pt. 4, 1944 (pub. 1946), pp. 17–29.

ORE DRESSING—CONCENTRATION—MEXICO.—New flotation tehnique in copper lead separation. G. O. Deshler and T. E. Herndon.—*Engng. Min. J.*, N.Y., Vol. 147, May, 1946, pp. 67–69. 50 cents.

ORE DRESSING—GENERAL.—The dressing of copper ores. F. B. Michell. (The author gives examples from practice involving all types of flow sheets.)—*Mine & Quarry Engng.*, Maidenhead, Vol. 11, 1946; April, pp. 107–12; June, pp. 145–49. 1s. ea.

QUEENSLAND — DUCHESS DISTRICT.—Some mines on the Cloncurry field. S. R. Leonard Shepherd.—*Qd. Govt. Min. J.*, Vol. 47, Feb., 1946, pp. 45–6. 6d.

INDEX OF RECENT ARTICLES—*continued.*

PER—*continued.*

ENSLAND — MOUNT ISA. — The al composition of the Black Star copper ody, Mount Isa, Queensland. F. L. ell and A. B. Edwards.—*Aust. Inst. Min. l, Proc.*, Melb., Vol. 139, Sep., 1945, pp. 9.

NING—ELECTROLYTIC METHODS. adian copper refineries, Montreal East J. H. Schloen and E. M. Elkin.— . *Canad. Inst. Min. Metall.*, Vol. 49, 1946, 43–90. (*Canad. Min. Metall. Bull.*, eal, No. 408, April, 1946, $1.)

NOMICS.

UNTS—STOCK RECORDS.—How on Bay runs its mine warehouse. Part. '. G. Bishop.—*Engng. Min. J.*, N.Y., 47, April, 1946, pp. 97–99. 50 cents.

—IRON MINING—MINNESOTA.— ground Lake iron ore costs must come J. Fred. Wolff.—*Engng. Min. J.*, Vol. 147, April, 1946, pp. 70–74. 50

CATION — MINING ENGINEERS. — ar education for mining engineers. Read.—*Min. Metall.*, N.Y., Vol. 27, ., 1946, pp. 173–74. 50 cents.

CATION — MINING ENGINEERS. — cal training, being the third report on cations and training for mining ers. Prepared by the examinations (formerly Committee on Qualifications raining for Mining Engineers).—*Trans. Min. Engrs.*, Lond., Vol. 105, Pt. I, 1945, pp. 1–7. 6s.

LTH OF MINERS—CHANGE HOUSES. omatic hot air clothes drier for mine e houses. M. D. Midson.—*Chem. . Min. Rev.*, Melb., Vol. 38, April, 1946, 39–40. 1s.

LTH OF MINERS — SILICOSIS— VENTION.—Aluminium dust as a y against silicosis.—*Min. J.*, Lond., 226, May 25, 1946, pp. 407–08. 8d.

LTH OF MINERS—VIBRATION.— injuries through vibratory tools. chweisheimer.—*Min. Mag.*, Lond., Vol. pril, 1946, pp. 222–4. 1s.

LTH OF MINERS — WORKING DITIONS.—Assainissement du travail es mines de charbons. Robert Dessard. . *univ. Min.*, Liége, 9th series, Vol. 2, 1946, pp. 69–97. 150 frs.

LTH OF MINERS — WORKING DITIONS.—L'effort fait pour améliorer nfort et la securité des entreprises rielles. M. Guerin.—*Rev. univ. Min.*, 9th series, Vol. 2, No. 2, 1946, pp. 1. 150 frs.

AGEMENT — COLLIERIES. — iistration of collieries—industrial ad- ration as a social function.—Pt. 8. *ssion*). F. Colin Swallow.—*Colliery* ., Lond., Vol. 23, May, 1946, pp. 130–32.

ECONOMICS—*continued.*

MINE MANAGEMENT—EFFICIENCY.— Mining efficiency. S. V. Griffith.—*Min. Mag.*, Lond., Vol. 74, April, 1946, pp. 215–6. 1s.

MINING LAW—IRAN.—Iran and its mineral deposits. N. W. Wilson.—*Min. Mag.*, Lond., Vol. 74, May, 1946, pp. 277–89. 1s.

MINING LAW—U.S.A.—COAL MINES.— Federal coal mine inspection. D. Harrington. —*Min. Metall.*, N.Y., Vol. 27, May, 1946, pp. 255–59. 50 cents.

PRODUCTION — AUSTRALIA. —— Australia's mineral industries; their development and expansion.—*S. Afr. Min. Engng. J.*, J'burg, Vol. 57, Pt. 1, April 13, 1946, pp. 157–59. 6d.

PRODUCTION — BASE METALS. — Post-war outlook for base metals: world conditions reviewed. Alan E. Bateman.—*S. Afr. Min. Engng. J.*, J'burg, Vol. 57, Pt. 1, April 20, 1946, pp. 169–71. 6d.

PRODUCTION — BRITISH COLUMBIA.— British Columbia mineral production in 1945. H. Sargent.—*W. Miner*, Vancouver, B.C., Vol. 19, April, 1946, pp. 58–59. 25 cents.

PRODUCTION — COAL — ALBERTA.— A review of Alberta coal mining during 1945. John Crawford.—*W. Miner*, Vancouver, B.C., Vol. 19, April. 1946, pp. 86–88. 25 cents.

PRODUCTION — COAL — BRITISH COLUMBIA.—British Columbia coal production. James Dickson.—*W. Miner*, Vancouver, B.C., Vol. 19, April, 1946, p. 92. 25 cents.

PRODUCTION — COAL — SASKATCHEWAN.—Coal mining in Saskatchewan in 1945. W. H. Hastings.—*W. Miner*, Vancouver, B.C., Vol. 19, April, 1946, pp. 90. 25 cents.

PRODUCTION — DIAMONDS — SOUTH AFRICA. — The Union's diamond industry: remarkable economic recovery.—*S. Afr. Min. Engng. J.*, J'burg, Vol. 57, Pt. 1, April 13, 1946, pp. 153–55. 6d.

PRODUCTION — EGYPT. — Egypt's minerals; post-war development plans. Martin Herlihy.—*Min. J.*, Lond., Vol. 226, May 4, 1946, pp. 346–7. 8d.

PRODUCTION — GOLD — AUSTRALIA.— Australian gold, the rise and fall of production. —*S. Afr. Min. Engng. J.*, J'burg, Vol. 57, Pt. 1, April 20, 1946, pp. 167–68. 6d.

PRODUCTION — GOLD — TRANSVAAL. —Transvaal gold industry; present position reviewed. C. S. McLean.—*S. Afr. Min. Engng. J.*, J'burg, Vol. 57, Pt. 1, March 30, 1946, pp. 95–99. 6d.

PRODUCTION — GOLD — U.S.A. — Gold mining in the United States; the slow recovery of the industry.—*S. Afr. Min. Engng. J.*, J'burg, Vol. 57, Pt. 1, May 11, 1946, pp. 265–66. 6d.

INDEX OF RECENT ARTICLES—*continued.*

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LIGHT METALS—*continued.*

BERYLLIUM — METALLURGY. — Converting beryl ore: the production of beryllium oxide and beryllium copper.—*Metal Ind.*, Lond., Vol. 68, May 17, 1946, pp. 385–88. 6d.

MAGNESIUM — METALLURGY — CARBOTHERMIC REDUCTION.—Construction and operation of the Dearborn magnesium pilot plant. C. E. Wood and others.—*U.S. Bur. Min., Rep. Invest.* No. 3823, Aug., 1945, 26 p.

MAGNESIUM — METALLURGY —CHLORINATION OF MAGNESIA.—Chlorination of magnesia. H. A. Doerner and W. F. Holbrook.—*U.S. Bur. Min., Rep. Invest.* No. 3833, Dec., 1945, 25 p.

MAGNESIUM — METALLURGY — GERMANY.—Aluminium and magnesium production and fabrication at I.G. Farbenindustrie A.G. Works at Bitterfeld, Aken, and Stassfurt, Saxony, Germany.—C.I.O.S. Item No. 21, File XXXII-59, 1946, 96 p., 68 plans.

MAGNESIUM — METALLURGY — MELTING.—Magnesium foundry practice; a review of melting methods. L. M. Nash.—*Metal Ind.*, Lond., Vol. 68, May 17, 1946, pp. 393–94. 6d.

SCRAP RECOVERY.—Recovery of metals from scrapped airplanes—a review of German practice at the end of the war. J. T. Kemp, H. H. Hall, and A. J Sidery.—C.I.O.S. Item No. 21, File No. XXII-55, 1945, 28 p.

METALLURGY AND CYANIDING.

CYANIDING — QUEBEC. — Cyaniding at Noranda. C. G. McLachlan, H. L. Ames, and R. J. Morton.—*Trans., Canad. Inst. Min. Metall.*, Vol. 49, 1946, pp. 91–122. (*Canad. Min. Metall. Bull.*, Montreal, No. 407, March, 1946. $1.)

CYANIDING — MANITOBA — FLIN FLON —Cyanide and regeneration plant and practice at Flin Flon. Del Davis, D. J. Robertson, W. Kirkwood, and J. Goodman.—*Trans. Canad. Inst. Min. Metall.*, Vol. 49, 1946, pp. 130–42. (*Canad. Min. Metall. Bull.*, Montreal, No. 408, April, 1946. $1.)

METAL WORKING — WELDING. — Arc welding at the Newcastle steel works. Alex Young.—*Industr. Aust. Min. Stand.*, Melb., Vol. 101, March 15, 1946, pp. 682–84. 1s.

METAL WORKING — WELDING. — Welding inspection. A. Warren-Smith.—*Welder*, Waltham Cross, Vol. 15, Jan.-March, 1946, pp. 18–21.

METALLURGY — GERMANY. — Mansfeldscher Kupperschieferbergbau A.G., Eisleben, Germany. (A source of non-ferrous metals in West-Central Germany).—C.I.O.S. Item No. 21. File XXXI-55, 1945, 11 p.

METALLURGY — NON FERROUS METALS — HISTORICAL. — Chemistry, metallurgy and Empire. S. Robson.—*Chem. Ind.*, Lond., 1946; April 20, pp. 162–65; April 27, 170–73. 1s. 9d. ea.

METALLURGY AND CYANIDING—*continued.*

POWDER METALLURGY—GT. BRITAIN.—British powder metallurgy; its history, development and commercial future. W. D. Jones.—*Metal Ind.*, Lond., Vol. 68, May 31, 1946, pp. 431–34. 6d.

SMELTING — BLAST FURNACES. — The blast furnace process and means of control. T. L. Joseph.—(A.I.M.E. Tech. Pub. No. 2021.) *Min. Tech.*, N.Y., Vol. 13, April, 1946, 22 p. $1.25.

SMELTING — BLAST FURNACES — SLAGS.—Effect of some added materials on dicalcium silicate. Edwin S. Newman and Lansing S. Wells.—(U.S. Nat. Bur. Stand. Research Paper 1696.)—*J. Res. Nat. Bur. Stand.*, Wash., D.C., Vol. 36, Feb., 1946, pp. 137–58. 30 cents.

MILLING AND CONCENTRATION.

CONCENTRATION — PLANT — BUCKMAN CONCENTRATOR. — Modernized device makes tin plant possible. John B. Huttl.—*Engng. Min. J.*, N.Y., Vol. 147, May, 1946, pp. 67–69. 50 cents.

CONCENTRATION — TABLES —LABORATORY. — Improved laboratory concentration table. Will H. Coghill, G. T. Adams, and H. S. Hardman.—*U.S. Bur. Min. Rep. Invest.* No. 3831, Oct., 1945. 5 p.

CONCENTRATION—TABLES — SIZING ACTION.—The sizing action of concentrating tables. R. T. Hancock.—*Min. Mag.*, Lond., Vol. 74, April, 1946, pp. 220–222. 1s.

CRUSHING.—Crushing and grinding efficiencies. T. K. Prentice.—*J. Chem. Metall. Min. Soc. S. Afr.*, J'burg, Vol. 46, Jan.-Feb., 1946, pp. 189–229. 3s. 6d.

FLOTATION — LABORATORY TESTING —ZINC.—Etude de la flottation des minerais calaminaires aux laboratoires de l'Institut de Métallurgie. F. Gaty and H. de Rycker.—*Rev. univ. Min.*, Liége, 9th series, Vol. 2, No. 4, 1946, pp. 155–66. 60 frs.

FLOTATION — SULPHIDES. — On the influence of oxidation processes on the flotability of sulphide minerals. I. N. Plaksin.—*C.R. (Dokl.) Acad. Sci. U.R.S.S.*, Moscow, Vol. 47, No. 9, 1945, pp. 632–34.

ORE HANDLING — NORTHERN RHODESIA.—Ore handling at Roan Antelope. M.R. Goldick.—*Min. Mag.*, Lond., Vol. 74, April, 1946, pp. 209–14. 1s.

PLANT — TASMANIA — KING ISLAND. —Laboratory investigations leading to the design of the King Island scheelite company's new concentrator. J. G. Hart.—*Aust. Inst. Min. Metall. Proc.*, Melb., Vol. 136, Dec., 1944, pp. 1–81.

SIZING — CLASSIFICATION — HYDRAULIC CLASSIFIERS.—Performance of a hydraulic classifier designed to incorporate four hitherto neglected principles. W. H. Coghill and others.—*U.S. Bur. Min. Rep. Invest.* No. 3844, Dec., 1945, 13 p.

INDEX OF RECENT ARTICLES—*continued.*

MINING (GENERAL).

ACCIDENTS — EXPLOSIONS — ACETYLENE MIXTURES.—Effect of pressure on the explosibility of acetylene-water vapour, acetylene-air, and acetylene-hydrocarbon mixtures. G. W. Jones and others.—*U.S. Bur. Min., Rep. Invest.* No. 3826, Sep., 1945, 17 p.

ACCIDENTS — FIRES — U.S.A. — Coal mine explosions and coal and metal mine fires in the U.S. during the fiscal year ended June 30, 1944. D. Harrington and W. J. Fene. — *U.S. Bur. Min. Inform. Circ.* No. 7330, Aug., 1946, 35 p.

ACCIDENTS—HAULAGE.—Safety aspects of modern mines transport systems. James Frame.—*Min. Elect. Mech. Engr.*, Manchester, Vol. 26, April, 1946, pp. 364–74. 2s. 6d.

ACCIDENTS — PREVENTION — U.S.A. — Federal coal mine inspection. D. Harrington. —*Min. Metall.*, N.Y., Vol. 27, May, 1946, pp. 255–59. 50 cents.

ACCIDENTS — SAFETY MEASURES — TROLLEY WIRES.—Guarding trolley wires in mines. E. J. Gleim.—*U.S. Bur. Min., Inform. Circ.* No. 7332, Sep., 1945, 19 p.

ACCIDENTS — TRAVELWAYS. — Safety and welfare; keeping travelways safe. H. Brodie Hicks.—*Canad. Min. J.*, Gardenvale, P.Q., Vol. 67, April, 1946, pp. 259–62. 25 cents.

ALASKA.—A summary of Alaskan mining. H. G. Wilcox.—*W. Miner*, Vancouver, B.C., Vol. 19, April, 1946, pp. 55–57. 25 cents.

BLASTING — EXPLOSIVES — CHLORATES.—Hazards from chlorates and perchlorates in mixtures with reducing materials. Irving Kabik.—*U.S. Bur. Min., Inform. Circ.* No. 7340, Dec., 1945, 6 p.

BLASTING — EXPLOSIVES — DISPOSAL. —Destruction of damaged, deteriorated, or unwanted commercial explosives. R. D. Leitch and P. R. Moyer.—*U.S. Bur. Min., Inform. Circ.* No. 7335, Aug., 1945, 6 p.

BLASTING — SHAPED CHARGES. — The shaped charge for cheaper mine blasting. John B. Huttl.—*Engng. Min. J.*, N.Y., Vol. 147, May, 1946, pp. 58–63. 50 cents.

BLASTING—STEMMING.—The resistance of stemming: dynamical investigations. James Taylor.—*Trans. Instn. Min. Engrs.*, Lond., Vol., 105, Pt. 2, Nov., 1945, pp. 51–74. 6s.

BORING — DIAMOND DRILLS — AUTOMATIC FEED.—Diamond drill with automatic feed. C. J. Esseling.—*Min. Mag.*, Lond., Vol. 74, April, 1946, pp. 217–19. 1s.

BORING—BOREHOLES—SURVEYING. — A method of platting the projections of crooked boreholes. Lysle E. Shaffer.—*Bull. Mo. Sch. Min. tec. ser.*, Rolla, Vol. 16, No. 4, Dec., 1945, 14 p.

MINING (General)—*continued.*

BRITISH COLUMBIA—BRIDGE RIVER AREA.—The Bridge River district. Franc R. Joubin.—*W. Miner*, Vancouver, B.C., Vol. 19, April, 1946, pp. 71–73. 25 cents.

BRITISH COLUMBIA—CARIBOO AREA. —The Cariboo district. Donald D. Fraser.—*W. Miner*, Vancouver, B.C., Vol. 19, April, 1946, p. 84. 25 cents.

BRITISH COLUMBIA — KOOTENAY AREA.—Mining in the Kootenays. Angus W. Davis.—*W. Miner*, Vancouver, B.C., Vol. 19, April, 1946, pp. 69–70. 25 cents.

DRAINAGE — PUMPS. — Some pumps for mine service (Boyle, Mono and Pneu pump). S. V. Griffith.—*Min. Mag.*, Lond., Vol. 74, June, 1946, pp. 356–62. 1s.

DRAINAGE — PUMPS — OPERATION. — Care and operation of pumps.—*Rhod. Min. J.*, J'burg, Vol. 18, May, 1946, pp. 111–15. 6d.

DRAINAGE — PUMPS — SUBMERSIBLE MOTORS. — Investigation of deep well turbine submersible pump motors.—B.I.O.S. Final Report No. 476, 1945, 15 p.

DRILLS—BITS.—Some new ideas in rock drill bit design. T. M. Waterland and G. E. Aproberts.—*Trans. Canad. Inst. Min. Metall.*, (Vol. 49, 1946, pp. 123–29. (*Canad. Min. Metall. Bull.*, Montreal, No. 408, April, 1946. $1.)

DRILLS—DRILL STEELS.—Progress in mining drill steels. L. Sanderson.—*Mine & Quarry Engng.*, Maidenhead, Vol. 11, June, 1946, pp. 139–40. 1s.

HAULAGE — CONVEYORS — CHAIN. — Conveyors and scrapers raise efficiency in Mammoth mining. Elkins Read and A. R. R. Richart.—*Coal Age*, Albany, N.Y., Vol. 51, April, 1946, pp. 93–98. 35 cents.

HAULAGE — CONVEYORS — CONTROL. —Automatic controls essential for conveyor operation today. C. W. Thompson.—*Coal Age*, Albany, N.Y., Vol. 51, April, 1946, pp. 86–92. 35 cents.

HAULAGE — LOCOMOTIVES. — New machinery—the Planet locomotive for gold and other non-gaseous mines.—*S. Afr. Engng.*, Vol. 57, May, 1946, p. 111. 1s.

HAULAGE — LOCOMOTIVES — DIESEL. —A new type of diesel locomotive for use in the Dutch State coal mines. M. G. Driessen.—*Trans. Instn. Min. Engrs.*, Lond., Vol. 105, Pt. 4, Jan., 1946, pp. 169–88. 6s.

HAULAGE — LOCOMOTIVES — DIESEL. —The diesel locomotive underground: Pts. 3 and 4. Brian Reed.—*Colliery Engng.*, Lond., Vol. 23, 1946; May, pp. 127–29; June, pp. 152–54. 1s. ea.

HISTORY — SPAIN.—Early mining in Spain. Douglas Rennie Hudson.—*Metallurgia*, Manchr., Vol. 33, April, 1946, pp. 293–300. 2s. (*Reprint. Presented to the I.M.E. by the author.*)

INDEX OF RECENT ARTICLES—*continued.*

MINING (General)—*continued.*

HOISTING—CAGE GUIDES.—Cage guides and winding ropes. R. W. Bell.—*Min. Elect. Mech. Engr.*, Manchester, Vol. 26, April, 1946, pp. 356–64.

HOISTING—CONTROL.—The control of mine winders. Rolt Hammond.—*Mine & Quarry Engng.*, Maidenhead, Vol. 11, April, 1946, pp. 117–22. 1s.

HOISTING — IDAHO — KELLOGG. — Sunshine's modernized hoist uses rotating control. R. F. Mahoney and H. B. Hodgins.—*Engng. Min. J.*, N.Y., Vol. 147, April, 1946. pp. 90–92. 50 cents.

HOISTING — SKIPS — MONTANA. — A bottom dump skip at Butte. W. R. Catrow.—*Min. Congr. J.*, Wash., D.C., Vol. 32, March, 1946, pp. 24–6. 20 cents.

ILLUMINATION — FLUORESCENT LIGHTING.—Fluorescent lighting in mines. —*Min. Elect. Mech. Engr.*, Manchr., Vol. 26, May, 1946, pp. 430–31. 2s. 6d.

ILLUMINATION—MAINS LIGHTING.—Mains lighting underground. C. S. Chubb.—*Colliery Engng.*, Lond., Vol. 23, May, 1946, pp. 121–24. 1s.

KOREA.—Mining in Korea.—*Min. J.*, Lond., Vol. 226, April 20, 1946, pp. 213–14. 8d.

LANCASHIRE.—Forgotten mines in Lancashire. Charles W. Loch.—*Min. Mag.*, Lond., Vol. 74, May, 1946, pp. 290–97. 1s.

MADAGASCAR.—Mining in Madagascar.—*Rhod. Min. J.*, J'burg, Vol. 18, April, 1946, pp. 93–7. 6d.

MANITOBA.—Looking back on Manitoba's mining in 1945. Geo. E. Cole.—*W. Miner*, Vancouver, B.C., Vol. 19, April, 1946, pp. 68–70. 25 cents.

METALLURGICAL DEVELOPMENTS.—Metallurgical news for the mining engineer. L. Sanderson. (A review of recent trends in metallurgical research and their application to the mining industry.)—*Mine & Quarry Engng.*, Maidenhead, Vol. 11, April, 1946, pp. 113–14. 1s.

MINING METHODS — BLOCK CAVING. —Lowering mining costs with block caving. Philip B. Bucky.—*Engng. Min. J.*, N.Y., Vol. 147, April, 1946, pp. 75–79. 50 cents.

PROSPECTING — GEOPHYSICAL PROSPECTING—BIBLIOGRAPHY.—Geophysical abstracts 121, April-June, 1945. W. Ayvazoglou and V. Skitsky.—*U.S. Bur. Min., Inform. Circ.* No. 7388, Nov., 1945, 48 p.

PROSPECTING — GEOPHYSICAL PROSPECTING — SOUTH AUSTRALIA.—Geophysical prospecting at Moonta and Kadina. E. Broadhurst.—*Chem. Engng. Min. Rev.*, Melb., Vol. 38, March 11, 1946, pp. 197–204. 1s.

MINING (General)—*continued.*

ROCK DRILLS—ROOF DRILLING.—A new idea in drill boosters. Julian A. Fuller.—*Min. Congr. J.*, Wash., D.C., Vol. 32, March, 1946, pp. 30–3. 20 cents.

SHAFTS — SINKING — ROTARY DRILLING.—Shaft sinking by rotary drilling. D. H. Platt.—*U.S. Bur. Min., Inform. Circ.* No. 7336, Sep., 1946, 14 p., biblio.

SOUTH AFRICA — NAMAQUALAND. — Mining progress in Namaqualand : wartime achievements outlined.—*S. Afr. Min. Engng. J.*, J'burg, Vol. 57, Pt. 1, April 13, 1946, pp. 145–47. 6d.

SUPPORT — STOPE FILLING — SAND FILLING.—Sandfilling at Mufulira. A. C. Turton.—*Bull. Instn. Min. Metall.*, Lond., No. 478, May, 1946, 24 p. 1s. 6d.

SUPPORT — TIMBERING. — Re-timbering No. 6 shaft, McIntyre Porcupine Mines. Keith O'Donnell.—*Canad. Min. J.*, Gardenvale, P.Q., Vol. 67, April, 1946, pp. 245–48. 25 cents.

SUPPORT — TIMBERING — TIMBER PRESERVATION. — Wartime operation of pressure wood-preserving plants in Germany. —F.I.A.T. Final Report No. 496, 1945, 38 p.

TRANSPORT—AERIAL ROPEWAYS.—A bi-cable aerial ropeway. A. E. Higgs.—*J. Instn. Mech. Engrs.*, Lond., Vol. 154, June, 1946, pp. 15–21.

VENTILATION — AIR CONDITIONING. —Mining and equipment problems.—*Gold Min. Rec.*, Lond., Vol. 15, 1946 ; April, pp. 82–3 ; May, pp. 108–09 ; June, pp. 144–45. 1s. ea.

VENTILATION — DUST — AIR SAMPLING — GRAVIMETRIC METHODS. —Gravimetric sampling technique, and investigations of results from different methods. A. G. Withers.—*Colliery Guard.*, Lond., Vol. 172, 1946 ; May 10, pp. 627–36 ; May 17, pp. 661–64. 10d. ea.

VENTILATION — DUST — PARTICLE SIZE.—A rapid method for the determination of the specific surface and mean particle size of black powders. F. Frith and R. A. Mott.—*J. Soc. Chem. Ind.*, Lond., Vol. 65, March, 1946, pp. 81–87. 3s. 6d.

VENTILATION — DUST — PARTICLE SIZE.—Meaning of particle size. Pt. 2. Calculation of nominal or equivalent average diameters of particles. H. H. Dunkin.—*Chem. Engng. Min. Rev.*, Melb., Vol. 38, Feb. 11, 1946, pp. 169–72. 1s.

VENTILATION — DUST — PARTICLE SIZE.—On the measurement of the size characteristics of powders by photo-extinction methods.—No. 2. The experimental investigation of the photo-extinction laws. H. E. Rose and H. B. Lloyd.—*J. Soc. Chem. Ind.*, Lond., Vol. 65, March, 1946, pp. 66–74. 3s. 6d.

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MINOR METALS—*continued.*

URANIUM—CZECHOSLOVAKIA—ERZGEBIRGE.—Europe's only uranium field; the Joachimsthal deposits.—*S. Afr. Min. Engng. J.*, J'burg, Vol. 57, Pt. 1, March 30, 1946, pp. 109. 6d.

VANADIUM—IDAHO—ROCKY MOUNTAINS.—Recuperation du vanadium dans les phosphates des Montagnes Rocheuses. Ph. Schereschewsky.—*Ann. Min. Carburants, Mem.*, Paris, Feb., 1946, pp. 123–28.

ZIRCONIUM.—Zirconium metal—manufacture, fabrication, and properties. D. B. Alnutt and C. L. Scheer.—*Metal Ind.*, Lond., Vol. 68, May 3, 1946, pp. 343–45. 6d.

NON-METALLIC MINERALS.

ABRASIVES — ELECTROCORUNDUM. — On some high alumina minerals in a block of electrocorundum. N. E. Filonenko.—*C.R. (Dokl.) Acad. Sci. U.R.S.S.*, Moscow, Vol. 48, No. 6, 1945, pp. 430–33.

ASBESTOS—SOUTH AFRICA.—Commercial asbestos with special reference to chrysotile. Alan Stuart.—*S. Afr. Min. Engng. J.*, J'burg, Vol. 57, Pt. 1, April 13, 1946, pp. 141–43. 6d.

BORON — RUSSIA — URALSK. — On the questions related to the origin of Inder borates. J. J. Yarzhemsky.—*C.R. (Dokl.) Acad. Sci. U.R.S.S.*, Moscow, Vol. 47, No. 7, 1945, pp. 642–45.

CEMENT—ANALYSIS.—Methods of mechanical analysis of portland cement. H. W. Rose.—*Engineering*, Lond., Vol. 161, 1946; May 17, pp. 457–59; May 21, 483–84. 1s. ea.

CEMENT—PORTLAND.—Effect of some added materials on diacalcium silicate. Edwin S. Newman and Lansing S. Wells. (U.S. Nat. Bur. Stand. Research Paper 1696.)—*J. Res. Nat. Bur. Stand.*, Wash., D.C., Vol. 36, Feb., 1946, pp. 137–58. 30 cents.

CLAY—STRUCTURE.—The structure of clay in relation to road construction. J. F. T. Blott.—*J. Soc. Chem. Ind.*, Lond., Vol. 65, April, 1946, pp. 113–18. 3s. 6d.

CONCRETE—AGGREGATES.—The durability of concrete in aggregates. F. H. Jackson.—*Cement, Lime & Grav.*, Lond., Vol. 20, April, 1946, pp. 351–63. 1s.

GRAPHITE—WESTERN AUSTRALIA.—Report on the Munglinup graphite deposits. H. A. Ellis.—*W. Aust. Dep. Min. Rep.* 1944, Perth, 1945, pp. 70–84.

PHOSPHATES—OCEANIA.—Phosphates in Oceania.—*Min. J.*, Lond., Vol. 226, April 6, 1946, pp. 266. 8d.

POTASSIUM — POTASH — CONCENTRATION.—Beneficiation of Bonneville potash deposits.—*Deco Trefoil*, Denver, Vol. 10, March, 1946, pp. 5–12.

NON-METALLIC MINERALS—*continued.*

QUARTZ—PIEZO ELECTRIC—TANGANYIKA.—Piezo-electric quartz in Tanganyika. B. N. Temperley.—Reprinted from *Tanganyika Notes & Records*, No. 19, June, 1945, 14 p.

SALT—CHINA—SZECHWAN.—Salt wells in Szechwan.—*Min. J.*, Lond., Vol. 226, April 27, 1946, pp. 325–7. 8d.

SALT—GEOLOGY.—On the theory of salt tectonics. N. M. Strakhov.—*C.R. (Dokl.) Acad. Sci. U.R.S.S.*, Moscow, Vol. 47, No. 6, 1945, pp. 421–24.

SLATE—U.S.A.—The slate industry in U.S.A.—*Quarry Mgrs.' J.*, Lond., Vol. 29, May, 1946, pp. 589–93. 1s.

SLATE—WALES.—A plan for the slate industry. D. Dylan Pritchard.—*Quarry Mgrs.' J.*, Lond., Vol. 29, May, 1946, pp. 559–67. 1s.

SULPHUR—ITALY.—Italy's sulphur industry.—*Min. J.*, Lond., Vol. 226, April 20, 1946, pp. 306–07. 8d.

ZIRCON.—Variation in the structure of zircon. Vaughan H. Stott and Alfred Hilliard.—*Miner. Mag.*, Lond., Vol. 27, June, 1946, pp. 198–203. 6s.

PETROLEUM.

ASPHALT—IVORY COAST—EBOINDA.—Etude du bitume d'Eboinda. Monsieur Louis.—*Ann. Min. Carburants, Mem.*, Paris, Feb., 1946, pp. 156–58.

BURMA.—The oilfields of India and Burma. Percy Evans.—*J.R. Soc. Arts.*, Lond., Vol. 94, May 10, 1946, pp. 369–79. 2s.

GERMANY.—German petroleum industry, Hamburg district.—C.I.O.S. Item No. 30, File XXXII-94, 1945, 141 p.

INDIA.—The oilfields of India and Burma. Percy Evans.—*J.R. Soc. Arts.*, Lond., Vol. 94, May 10, 1946, pp. 369–79. 2s.

NATURAL GAS—U.S.A.—Le gaz naturel aux Etats-Unis. Ph. Schereschewsky.—*Ann. Min. Carburants, Mem.*, Paris, Jan., 1946, pp. 24–33.

REFINING—GERMANY.—Band oil filter oil refinery at Porta (near Minden).—B.I.O.S. Final Report No. 324, Item No. 30, 1945, 9 p.

VICTORIA — LAKE ENTRANCE. — The search for oil at Lake Entrance, Victoria.—*Chem. Engng. Min. Rev.*, Melb., Vol. 38, March 11, 1946, pp. 195–96. 1s.

PLANT AND POWER.

ELECTRICITY—GOLD MINING.—Hydro-electric power and Ontario gold mining. Ralph E. Foster.—*Canad. Min. J.*, Gardenvale, P.Q., Vol. 67, April, 1946, pp. 253–58. 25 cents.

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THE INSTITUTION OF MINING AND METALLURGY

FIFTY-FIFTH ANNUAL GENERAL MEETING.

Held in the Apartments of the Geological Society, Burlington House, Piccadilly, W.1,

ON

Thursday, May 16th, 1946.

Mr. G. F. Laycock, *President*, in the Chair.

The Minutes of the previous Annual General Meeting, which had already been circulated, were taken as read, and confirmed and signed.

Appointment of Scrutineers

Major W. H. Wilson proposed, **Major J. L. Farrington** seconded and it was agreed that Messrs. P. T. Halsey, C. H. W. Martyn, and H. C. Webb be appointed to act as Scrutineers to examine the balloting papers for the election or re-election of Members of Council.

The Benevolent Fund

Mr. E. D. McDermott, in making a brief report on the Benevolent Fund, said that the Twenty-First Annual Meeting of this Fund had just been held and copies of the Report of the Committee of Management and the Accounts would be circulated to the members, together with a list of subscribers for the current year. From the Report it would be seen that the expenditure, as compared with the previous year, had increased by about £185 and exceeded the revenue by just over £100. On the other side of the account the capital fund had benefited by legacies to the extent of £1,600 and there had again been a slight increase in the amount recovered from the Inland Revenue. He appealed to all subscribers liable for British Income Tax to make use of the form of covenant attached to the Report and Accounts. If they could get a greater number of subscribers under covenant it would counterbalance the loss which would arise from the slight decrease in the rate of Income Tax.

He was grateful for this opportunity of thanking their subscribers and assuring them that their generosity had enabled the Committee of Management to alleviate to some extent the hardship of members of their profession and their dependents who had fallen upon hard times, and also to help in the education of the younger generation. At the same time and at the risk of appearing ungrateful, he would point out again that he thought the proportion of members and associates who subscribed to the fund was too low and he earnestly appealed to all who had not done so to send a donation, however small, preferably under covenant and by means of a banker's order.

He was glad to say that Dr. S. W. Smith had kindly consented to act as Honorary Secretary for the coming year and their thanks were due to him and to all the other members of the Committee who had given their time and services in the efficient management of the Fund.

Report of Council and Statement of Accounts

The President, in moving the adoption of the Annual Report of the Council for the session 1945–46 and the Statement of Accounts for the year ended December 31st, 1945, said the accounts continued to show a satisfactory position and would be dealt with by the Honorary Treasurer, who would be seconding this motion. As regards the Annual Report, there were one or two items to which he would like to make brief reference.

As members knew the Institution had suffered a severe loss by the death of their late President, Colonel Edgar Pam, during his second year of office. They would all miss his great practical knowledge and ability as an outstanding mining engineer of broad vision who had the best interests of the Institution always at heart.

Dr. Lawn having expressed his desire to retire from the office of Honorary Treasurer, the Council very regretfully acceded to his request and he (the President) would be referring to this again later on in the course of the meeting. Members would be pleased to hear that Mr. Robert Annan had agreed to accept this very important office. (Applause.) Mr. Annan had for some time been Chairman of the Finance Committee, and he was sure no more suitable successor to Dr. Lawn could have been appointed.

Members would have learnt with great pleasure of the generous action of the Directors of the Mond Nickel Company in making available the sum of £50,000 for the establishment of Post-Graduate Fellowships in Metallurgy.

It was announced in the Annual Report that arrangements were being discussed for a joint meeting with the Institution of Mining Engineers on the subject, of silicosis and dust control in mines. The Council had recently decided to establish a lecture in memory of the late Sir Julius Wernher, a great figure in the history of South African mining and a benefactor of the Institution. They were pleased to announce that Dr. A. J. Orenstein, the eminent

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South African authority on silicosis, had consented to deliver the first Julius Wernher lecture, which would be held in London in April or May of next year in connection with the joint meeting already mentioned. Dr. Orenstein had also kindly consented to have prepared a comprehensive paper on South African methods of combating silicosis, which the Council felt would be one of the most valuable contributions which the metalliferous mining profession could make to their colleagues in the coal-mining industry at the present time. The value of this paper would be greatly augmented by the opportunity which would be afforded of personal discussion with the author.

Dr. James G. Lawn, in seconding the adoption of the Report and Accounts, said that that was, so to speak, his swan-song, as he was ceasing to be Honorary Treasurer. He was very thankful, however, to know that he would be succeeded by a better man. With regard to the finances, they were in a sound position. As would be seen, they had four blocks of investments in their balance-sheet and in every case the present-day value was substantially greater than the cost price of the stocks held. Altogether the appreciation amounted to about £12,000, while the total amount in the invested fund was about £122,000.

They had got through the war without any undue expenditure. They undertook a little risk in order to save the library. They moved it to Derbyshire and bought a house there to hold it. They gave £1,000 for the building and wrote it down to half that amount. The other day they sold it by auction and got £2,000 for it, so they had come out of that deal quite well. His only regret was that the Institution was not more adequately housed at the present time. They had, as the accounts showed, a certain amount of money available and no effort was being spared to find more suitable accommodation, but there was considerable difficulty at the present time owing to the conditions produced by the war.

The President invited questions on the Report and Accounts, but none was forthcoming and the motion for the adoption of the Report and Accounts was put to the meeting and carried unanimously.

RE-APPOINTMENT OF AUDITORS

On the motion of **Mr. H. M. Ball,** seconded by **Colonel J. B. Simpson,** Messrs. Woodthorpe, Bevan & Co., were re-appointed Auditors of the Institution for the current year.

VOTE OF THANKS TO THE GEOLOGICAL SOCIETY

The President then 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. This was seconded by **Dr. J. G. Lawn** and carried by acclamation.

VOTE OF THANKS TO THE COUNCIL, OFFICERS AND STAFF

Colonel L. C. Hill, in proposing a vote of thanks to the Council for their work during the past year, said he supposed he represented the majority of the members of the Institution, who spent most of their time abroad and who therefore had little opportunity of taking an active part in the work of the Institution and who very rarely had a chance of meeting members of the Council. He thought there was considerable danger of their taking the work of the Council very much for granted, but if they stopped to think a little it must be realized that a tremendous amount of really hard work had to be done in order that the work of the Institution should be carried on satisfactorily. This was principally done by busy men in their spare time and must entail a considerable sacrifice of leisure on their part. If these sacrifices were not made freely and generously the standing of the Institution could not possibly be as high as it undoubtedly was, neither could the Institution be of such use to its members or the public as was certainly the case. Members were deeply grateful to the Council for their unfailing care of their interests and for the jealous maintenance of those high standards which had come to be regarded as normal for the profession.

Lieut.-Colonel A. R. O. Williams, in seconding the resolution, said he felt sure that all classes of members would heartily endorse Colonel Hill's remarks. He quite agreed that many of them took for granted the work done by the Council and the staff. A study of the Annual Reports, which contained much useful and interesting information, made one realise the manifold activities of the Institution. He wished to add a special word of thanks to the permanent staff headed by their genial secretary Mr. Felton. Any member visiting the headquarters of the Institution could be sure of a very real and delightful welcome.

The resolution was also carried by acclamation.

The President, in returning thanks, said it was a fact that a great deal of work had to be done, but it was always most cheerfully undertaken by members of the Council, many of whom devoted a considerable amount of time to the affairs of the Institution. As to the Secretary and staff, there was nothing much which one could add to what had been said on previous occasions about their loyalty and efficiency, except perhaps that during the last twelve months their task had been more onerous than usual. This was due to shortage of personnel and the extra work involved in the return of the library from Derbyshire to the much more convenient premises at Salisbury House.

Mr. Robert Annan thanked the President and Dr. Lawn for the kind remarks they had made about him and the members of the Institution for the confidence which they had shown in his election to Honorary Treasurership. He was afraid he had made a bad start, as only that morning he had received a reminder from the Secretary that he had omitted to pay his subscription, but he would endeavour to do better in the future. (Laughter.)

The meeting then adjourned for tea, resuming at 5.15.

Presentation of Awards

The President said he had not realized that such pleasant duties as those he now had to perform would fall to his lot. As members would have seen from the Council's Report, the Gold Medal of the Institution had been awarded to Mr. Carl Davis in recognition of his services to the gold-mining industry of South Africa, to mining in Northern Rhodesia, and to mining education as the representative of the Institution for eight years on the Governing Body of the Imperial College of Science and Technology. Nothing could give him greater pleasure than to have the delightful task of presenting the highest award in the gift of the Institution to a man whom he had always admired as such an outstanding example of the mining engineering profession. The citation on the award spoke for itself and showed what a wonderful career Mr. Carl Davis had enjoyed.

In addition he would like to refer to another of his many activities and that was the valuable work he did during the past world war for the Tunnelling Companies of the Royal Engineers as Chairman of the Tunnellers' Comforts Committee. This Committee raised a considerable sum of money almost entirely, he might say, as a result of Mr. Davis's exertions, and did a great deal of most useful work in looking after the welfare and comfort of Tunnellers both overseas and at home.

His one regret on this important occasion was that, owing to war-time restrictions which still prevailed, they were only able to present Mr. Davis with an " austerity " edition of a Gold Medal. To have to present to a man who had been responsible for the production of so much gold in this world nothing better than a 9-carat model was, to his mind, rather absurd. However, it was the best they were allowed to do, but he understood that the balance of the metal used in the medal was mostly copper, which perhaps was rather appropriate in view of Mr. Davis's close connection with the copper-mining industry in Northern Rhodesia.

Mr. Carl Davis, in returning thanks, said he found this rather a novel and embarrassing situation. Nevertheless, he was grateful to be the principal actor in it. He felt that as now he had reached a stage in life when he might take, and would be taking, a less active part in the mining industry, this wonderful award, which he thought was priceless to any mining engineer, came at a very opportune time, because it marked the culmination of his more active participation in the industry. He had not prepared any formal speech for the occasion. He felt he would much prefer, if they would bear with his disjointed remarks, to talk in a manner which was significant of the very intimate and close relations he had had for many years with so many of the members present on this occasion. Also, in thinking of this medal which they had awarded him, although it came to him personally, mentally at least he would share it with many good friends and colleagues who had co-operated with him in so many enterprises over many years. None of these operations was due entirely to individual effort and in regard to his own case he felt that whatever modest success he might have achieved, he owed much to his many friends.

There was also a point in connection with the award which in itself gave one the greatest possible pleasure and that was, it would be a constant reminder of the many years he had had the honour to be associated with this Institution. The Institution had done a great work in the past and he saw no reason why it should not do an even greater work in the future. Conditions changed but he hoped that while there would be no lessening in the service the Institution would be able to render in the spread of technical information for the benefit of the industry, it would continue to take an ever-increasing part in the broader aspects of the industry—such as health, social conditions of the workman, taxation, etc. In fact, he might run through a list of the various things which engaged the attention of the President and Council, showing that the Institution was playing an increasingly important part and carrying an increasing weight with the authorities and with appropriate bodies everywhere. That he was perfectly certain of and he could only say it was his belief that in those matters what they would do in the future would redound, as in the past, to the credit of the Institution.

He was personally very proud of being a member of the Institution and very proud of being a mining engineer. If he had to live his life over again he would be a mining engineer. There was an interesting thing which had occurred to him, as it had probably occurred to many of those present, regarding mining engineers. He remembered when he was a very young man receiving his degrees and that sort of thing; in the old days, when applying for a job, that was one of the things they kept in the dark from the

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'practical people' who directed the industry. That had all gone and today they found the highly-trained men occupying a much larger sphere. The old system of putting the engineer in a water-tight compartment and confining him purely to technical matters had in a large measure disappeared. In the principal mining fields of the world the positions of importance—executive and others—were now largely occupied by mining engineers. He thought that was as it should be and it ought to be an encouragement to their younger members.

In conclusion he could only say he was very proud indeed to have been awarded the highest honour the Institution could confer and he thanked them very much. (Applause.)

The President said that next to presenting a Gold Medal to Mr. Davis he did not think anything could give him greater personal satisfaction than the next duty he had to perform, and that was the award of Honorary Membership to Dr. James Lawn. As everybody knew, Dr. Lawn had carried out the important duties of Honorary Treasurer of the Institution for the past eight years, but he doubted whether very many members realized the responsibilities attached to this office. Dr. Lawn had been re-elected Honorary Treasurer every year since he was first appointed in 1938, so in a sense he had been a permanent official of the Institution for that period. Presidents had come and gone, but Dr. Lawn had always been there to watch over the financial affairs of the Institution, with what good effect members could see for themselves in the satisfactory state of the Institution's finances today.

But beside all this Dr. Lawn had always been, if he might say so, their 'sheet-anchor' in the general affairs of the Institution. Many had been the time when, in Council and Committee, members had been considering some particularly knotty problem and had not been able perhaps to make up their minds as to just what was the right course to adopt, when Dr. Lawn in his calm and deliberate way had put the position in a much clearer light and pointed the way out of the difficulties. Fortunately his very able and valuable services would still be available to the Council as a Past-President. If anybody had ever been entitled to be called an 'Elder Statesman' of the Institution, that man was Dr. James G. Lawn, to whom he now had the greatest pleasure in presenting the Honorary Membership of the Institution. (Applause.)

Dr. James G. Lawn said he scarcely knew how to thank them for the great honour they had done him in making him an Honorary Member of the Institution. This honour was usually given to those who had great names and great achievements to their credit in some special direction, but in this case it was, as he understood it, an acknowledgment, so to speak, of just steady, patient work, which was, of course, very necessary. He felt that in the higher plane of mining engineers he could not rank for such an honour, but he took it as an expression of gratitude on a somewhat lower plane. The Institution had always been very near to his heart.

He had done what little he could to help it along and would continue to do so so long as he had health and strength left and could be of any further service.

He felt that it was right that he should go at this time, because it was an era when changes were coming about in many directions. Points of view were changing and the future would not be exactly as the past had been ; the Institution, he was quite certain, would benefit by the change. It was the greatest satisfaction to him and a great comfort to feel that he was being succeeded in this position by such an able man as Mr. Robert Annan. He felt sure that in Mr. Annan's hands the financial future of the Institution would greatly benefit. (Applause.)

The President said that his next duty was to present 'The Consolidated Gold Fields of South Africa, Limited' Gold Medal and Premium of 40 guineas to Mr. Hildick-Smith for his paper on 'Shaft Pillars and Shaft Spaces.' Members would remember what an interesting paper this was and, he was sure, would heartily approve of this award. Mr. Hildick-Smith, who occupied a very important position on the Rand, was unfortunately unable to be present to receive the award in person. He had, however, done the next best thing and sent his son, Dr. G. Hildick-Smith, on his behalf to whom he now had much pleasure in handing the award.

Dr. G. Hildick-Smith said he felt it a very great honour to be allowed to attend this meeting on behalf of his father and a very much greater honour that the Council should have made this award. He knew that nothing would have pleased his father more than to be able to attend this meeting himself and to thank them personally for the award.

Address by the President

The President said that in the ordinary way the next item on the Agenda at an Annual Meeting would be the induction of the new President. Owing to the very unfortunate death of their late President while in office, he had the honour of being appointed to succeed him and was duly inducted into the Chair at the General Meeting held in January last. He was hoping that perhaps this might relieve him of the task of having to deliver a Presidential Address, but apparently he was not to be let off so lightly as all that. He was afraid therefore they were going to have to bear with him for a little while longer while he carried out the time-honoured tradition of the Institution of reading an address, or, at least, as much of it as time would permit.

He then proceeded to deliver his address.

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THE INSTITUTION OF MINING AND METALLURGY.

PRESIDENTIAL ADDRESS.

Mineral Exploration and the Outlook for Metal Supplies.

ALTHOUGH previous occupants of this Chair have in the past often remarked upon the difficulty they have experienced in selecting a suitable subject for a Presidential Address, I must admit that I have not on the present occasion found this to be a problem. This is due, I suppose, to the fact that there is to-day one topic uppermost in the minds of most mining engineers—namely, 'Where are our future supplies of some of the metals to come from?' and if, as would certainly appear to be the case, the answer is not at all clear at the moment, then, 'What do we as mining engineers suggest ought to be done about it?'

I think, therefore, that I cannot do better than devote my remarks this evening to the subject of 'Mineral Exploration'. First, I would like to consider the general position as regards the probable resources of some of the more important metals in the world to-day and what the prospects are for finding further supplies of them, and, secondly, to review briefly the methods employed in the search for new orebodies.

There are many people occupying important positions both here and in America who are inclined to take a rather pessimistic view about the future supply of some of the base metals. On the other hand there are many persons, whose views certainly cannot be ignored, who are just as strongly of the opinion that discoveries of new ore deposits will continue to be made as in the past and that there is no need to become unduly alarmed about the situation. As is so often the case in differences of opinion in matters of this kind, the probability is that neither side is completely right and that the real answer will be found to lie somewhere between these two extremes.

While it is true that there still are very large reserves of many of the important minerals, such as coal, phosphates, the ores of iron, molybdenum, and nickel, it is equally true that the resources of the ores of some other important metals, particularly lead, tin, zinc, antimony, and chromium, are decidedly limited in extent as far as our present knowledge goes. It is not very clear in which category copper should be placed, but this will be referred to again later.

The mining engineer and the mining geologist are faced with two problems of paramount importance :

(1) The location of new orebodies within mineralized areas already known, and

(2) the location of new mineralized districts where the geological conditions are favourable and within whose confines previously unknown and valuable ore deposits may occur.

It is elementary, but nevertheless worthy of repetition over and over again, that some of the most promising places in which to look for new ore deposits are in those areas that are already known to be favourable from a geological point of view for the deposition of mineral. In this connection the closer examination of the immediate neighbourhood of known mineralized areas is probably as good a bet as any and perhaps a good deal better than most. There are doubtless numerous orebodies still lying undiscovered in the vicinity of old mine workings simply for the lack of imagination on the part of the management or the absence of additional lateral exploratory work by cross-cutting or diamond drilling. The importance of cross-cutting cannot be over-emphasized and yet it is surprising how often this fundamental principle of mining practice is neglected.

As has already been remarked the great majority of producing mines were discovered many years ago and of recent years comparatively few new ones have been brought to the production stage. In the case of some of the more important base metals, particularly lead and tin, it is becoming very obvious that the known resources of these metals are none too plentiful. If the supply of these two metals in particular is to be maintained, then time is getting very short in which to make the much-needed discoveries of additional sources of supply.

A tremendous amount of energy and skill have been devoted in the past and are still being employed in improving mining and milling methods and developing new processes for the treatment of ores and minerals. This is all to the good and will no doubt result in certain complex or low-grade ores being brought into the economic category instead of being left unworked as at present. It might perhaps be more logical, however, if a similar amount of technical skill were also to be devoted to improving present-day methods of finding new orebodies.

One often hears the opinion expressed that in the past most of the important orebodies found in the world have been discovered, not by engineers or geologists, but by accident or by those persons who happen to have a flair for such things. This may have been true to some extent, but it is very much open to question whether such an assertion will apply at all in the future. Most of the easily-discoverable mineral occurrences, namely those that outcrop on the surface, have already been found and in future one will need something more than a flair, or a good nose for ore, to find orebodies which do not give any direct surface indication of their presence.

Nobody will deny, I think, that the era of visual prospecting has entered the phase of diminishing returns. Doubtless there are a few parts of the globe which have not yet been thoroughly explored or probably even looked over by the old-fashioned prospector, such as parts of Alaska, some of the North-western districts of Canada and parts of Siberia, and Central Asia. These areas, however, represent only a comparatively small portion of the earth's surface and many of them do not contain favourable rock formations. It is often overlooked that a large part of the earth's surface consists of rocks which are not at all likely to contain metalliferous deposits of economic importance. On the other hand few, if any, mining engineers and geologists will deny that in all probability there are many valuable ore deposits, still undiscovered, which do not outcrop or even approach very close to the surface. The question arises, therefore, how are these deposits going to be found so that they can be developed and brought to the production stage to take the place of present-day deposits, many of which are rapidly becoming worked out.

There is an old Cornish saying that 'Good mines die hard' and this is just as true to-day as ever it was. Ore reserves sometimes have a fortunate habit of persisting much longer than expected, owing to the discovery of new makes of ore underground and the finding of extensions to known orebodies during the extraction of the ore. Whilst this is a very comforting thought, it would be dangerous to place too much reliance upon such persistence, as we all know of cases where things did not work out that way at all.

It would be interesting if one could calculate what the world's total known ore reserves of the more important base metals really are at the present time. Reliable data, however, are unfortunately very hard to obtain, but an investigation into what information is available makes it apparent that in the case of some metals at any rate existing known reserves are becoming uncomfortably low.

Lead is probably as simple a case as any to enquire into, owing to the fact that the bulk of the known resources of this metal are centred either in comparatively few districts or in large deposits which can be more or less gauged as to their probable extent and life. I have therefore endeavoured to make a more detailed study of the position of lead—and this naturally had to include its close associate zinc—but soon found that any such investigation entailed a great number of assumptions. While many of the producing mines publish figures of their ore reserves, some companies appear very reluctant to do so. Having had some experience of the behaviour of lead-zinc orebodies, I have felt emboldened to fill in some of the gaps with estimates of my own which, if anything, have perhaps erred on the optimistic side. Nothing, however, has been allowed for the U.S.S.R. resources and, knowing from personal experience something about the mineral deposits of that country and their possibilities, I fully realize that this is a most important omission. Any estimate for that country, based on our present meagre information, would be nothing more than a wild guess. But whatever those resources may be most of them will probably be required for the future industrial development of that great country itself. For the U.S.A., which still contains a large proportion of the world's reserves of these two metals, I have taken the figures prepared by E. T. McKnight and E. F. FitzHugh, which were quoted by Elmer W. Pehrson last year in his most illuminating address on the mineral position in the United States.

The result of this admittedly rather incomplete study is startling. As far as I can learn from the information available there would appear to be only sufficient known resources of the ores of lead in the world at the present time to last for about another 14 years at the pre-war rate of consumption. In the case of zinc the period is undoubtedly longer—probably about 21 years. That does not imply that all the present mines which are producing lead and zinc will be worked out in that comparatively short time. Some of them no doubt have much longer lives at their current rate of production, but it does certainly look as if the total world supply will in a few years begin to decrease and before long will be insufficient to fill all requirements.

A somewhat similar position would appear to be the case as regards the world supply of tin, but I have been quite unable to find any reliable figures on which to base an estimate, and here the price factor will also probably play a very important rôle. There are undoubtedly large tonnages of low-grade material, both lode and alluvial, available in different parts of the world which cannot be economically operated under present conditions. A substantial increase in the price of the metal or an improvement in the present methods of treatment, whereby better recoveries could be made, would probably bring a large quantity of tin-bearing ground into the economic picture. In this connection it should be borne in mind that we do not have to look very far afield to find such places. According to many competent observers Cornwall is only awaiting an opportunity of demonstrating that it is by no means a completely worked-out area and that it is quite capable of continuing to produce substantial quantities of tin, provided that conditions are made more attractive to capital than at present and that methods can be devised whereby something better than a fifty per cent recovery can be made from some of the ores remaining to be worked.

Copper is much more widely distributed in the various countries of the world than lead, zinc, and tin and in much more generous quantities. In fact it would appear at first glance that there was a plentiful supply of this most essential metal but it must be remembered that the demand is also very great. It would be difficult to estimate how much copper there is in the known deposits of the world but there is apparently no real shortage in sight in the near future.

All this, I think, goes to emphasize the urgent necessity in the immediate future for a global survey of the world's resources of these base metals, as has already been suggested in certain quarters. This would enable a much clearer view of the whole situation to be envisaged. It is often claimed that the price factor will always successfully adjust supply to demand, but there must be definite limitations to the application of this theory to wasting assets. Just how and by whom the survey should be carried out has not, as far as I am aware, been made very clear, but it would seem that this is an obvious task for the United Nations Organization. This is possibly the only body that would have the necessary authority and personnel to obtain the requisite information from all the various countries concerned. And surely if some of the hopes expressed in the Atlantic Charter and subsequent statements are to become translated into actual fulfilment, then a global survey of the mineral resources of the world is, it would seem, one of the primary tasks that this body ought to set itself.

Perhaps one of the best ways to ensure a revival of interest in the search for new mineral deposits would be for governments to agree to allow all expenditures incurred in the search for, and development of, new orebodies to be used as a write-off against future operating profits before taxation is applied. The profit motive is, after all, the principal incentive in mining as in all other industries and as mining is more highly speculative than most it deserves much more sympathetic consideration from governments and taxation departments than it has received in the past. This obvious home truth has been stressed over and over again by speakers both here and in North America. As is well known, the Institution has taken a leading part in bringing the matter to the notice of the Authorities in this country with very encouraging results. Much has been accomplished, but more remains to be done, and we must lose no opportunity of stating our case as often and as loudly as possible.

There is another way in which the profit motive can be used as a further incentive in the search for new ore deposits and that is by giving the prospector or discoverer either a cash bonus or a small percentage interest in anything he finds. This is already the practice in Canada and other parts of North America. It is true that it is liable to lead to a number of 'wild-cats' being brought in for examination at times, but an occasional 'winner' makes up for a lot of disappointments.

Having demonstrated, I hope, the vital need for more intensive mineral exploration in the world to-day, I would now like to review, somewhat briefly, the various prospecting methods which are in use at the present time. These can be classified for the most part under the following headings:

(1) Aerial Reconnaissance.
(2) Surface Prospecting.
(3) Geological Survey and Mapping.
(4) Geophysical and Geochemical Methods.
(5) Diamond Drilling.
(6) Exploration by means of Sub-Surface Workings.

Aerial Reconnaissance

From the point of view of making direct mineral discoveries it is doubtful whether aerial reconnaissance is of very great practical value at present. It can, however, be helpful in a general way, provided too much is not expected from the service. Topographical features can be picked out more quickly perhaps from the air than by any other means and sometimes characteristic changes in the rock formations may also be noted by experienced and competent observers. For instance, it is reported that at Yellowknife in the North-West Territories of Canada the geologists were able to pick out from the air, much more easily in fact than they could on the ground, valuable information as to the extent and boundaries of the favourable shear-zones in which the gold-bearing formations are found. It has also recently been announced that some of the Nigerian tin companies are arranging to make an air survey of their areas in that country for topographical and geological information.

One way in which aircraft can be of the utmost service in prospecting in remote regions is as a means of transportation for both men and materials. Tremendous strides have been made of recent years along these lines, particularly in Northern Canada, where many mining prospects and even some producing mines are almost entirely dependant upon air transport. In this connection the development of the helicopter is being watched with great interest by mining men. This would seem to be the ideal method of supplying and keeping in touch with distant prospecting parties in a country difficult of access by normal means. At present helicopters are stated to be rather expensive, both as regards initial cost and operating charges, and to require a much greater degree of technical skill to operate than ordinary aeroplanes, but these disadvantages will no doubt be overcome in the not too distant future.

Surface Prospecting

There is nothing very new that one can say about this method of locating mineral deposits, which is almost as old as the hills themselves. The real old-fashioned prospector is unfortunately becoming nearly extinct, having been almost frozen out by the lack of opportunity and the fierce competition of the field parties employed by so many of the large mining companies to-day. Also the burden of heavy taxation has practically eliminated the grub-staker. Nevertheless there will probably always be a few adventurous spirits who have the call of the wild in their blood and who will continue to spend most of their time in the 'back of beyond', ever hoping to strike something rich over the next divide. In those countries where fur-bearing animals abound the trapper is often also something of a prospector and many are the samples of mineral brought into camp by these hardy fellows. New discoveries will therefore no doubt continue to be made occasionally by these rough and ready methods, but to a much lesser degree than in the past.

The development of the drag-line scraper and, more recently, that of the bull-dozer and angle-dozer, has placed more efficient weapons in the hands of the engineer for stripping areas around prospects once they have been located. Many of us can no doubt think of occasions in the past where we would have given a great deal to have had an angle-dozer available at a promising surface prospect, instead of having had to rely on pick-and-shovel methods to remove overburden and expose the rock formation.

Geological Survey and Mapping

This is probably the most important first step to take in all mineral exploration programmes after a promising area has once been located. Topographical and geological maps are of the utmost importance and without them one is liable to waste a great deal of time and probably miss important clues. Such an important subject can hardly be enlarged upon here and it must therefore be left with only this brief reference.

Geophysical Methods

To many people geophysical prospecting has been something of a disappointment. In its early days it appeared to have all the possibilities of the revolutionary method for which we had all been waiting for finding concealed orebodies, and some of us thought that here at last was the answer to the mining engineer's prayer. There is no question that very important orebodies have been discovered by these methods, as some of us well know and for which we are everlastingly grateful. As one who has actually seen several million tons of high-grade sulphide ore, which gave no indication of its presence on the surface, discovered entirely by geophysical methods, I feel that there must be a great future for this comparatively new science. But, unfortunately the early successes were rather too good to last and it was soon realized that there were decided limitations to geophysical methods of prospecting.

Although many valuable discoveries have been made by these methods, it is equally true that there have also been failures and disappointments. One of the main defects appears to be that so far it has not been found possible in many instances to distinguish between valuable and worthless mineralization. This is perhaps not to be wondered at, as it is not always easy to do this even by visual methods. Unfortunately it is also often impossible to distinguish between indications caused by mineralization and those resulting from barren structures, such as faults, shears, and brecciation.

It is doubtful whether any of the electric or magnetic methods at present employed, other than in exceptionally favourable circumstances, can be depended upon to locate orebodies at more than a comparatively short distance below the surface. To what depth these geophysical methods are effective is an uncertain factor at present and this no doubt varies with the nature of the formation. It does not follow, therefore, that because an area has been prospected geophysically and drawn blank that there are no mineral deposits at depth in that area. It may, however, be taken for granted that there is no mineralization—at least of a kind which gives electrical anomalies—of major importance lying anywhere close to the surface. Here I must emphasize that these remarks refer only to prospecting for metalliferous deposits and have nothing to do with oil, which is quite a different story.

In a paper recently published by one of our members, Hans Lundberg, a leading authority on these matters, it is definitely stated that

(1) Bodies of sulphide may create less electrical activity than a scant dissemination of the same mineral.

(2) Porosity and fracturing in a rock will increase its conductivity. Thus the usual resistivity methods show no distinction between the mineralized zones and moist porous rock.

The above two quotations alone illustrate only too clearly some of the difficulties to be overcome. However, in these days of marvellous scientific discoveries, of which we have had evidence during the war years in connection with 'Radar' and supersonic devices, perhaps it is not too much to hope that before long some new and revolutionary process may be perfected whereby mineral deposits which are at present completely hidden may be brought into the service of mankind.

There have been numerous rumours lately of wonderful results that are being obtained by Radar methods for the locating of concealed ore deposits, but unfortunately it is impossible to obtain any reliable information about these reports. Meanwhile one hesitates to attach too much importance to them. We also hear of Radar being employed in topographical map-making from aeroplanes and it is well known that experiments are now in progress in the use of geophysical methods from the air where the instruments are carried in a helicopter which can travel at any required speed and height from the ground. By this means it is claimed that a lot of territory can be covered in a comparatively short time, but whether it will be possible to discover orebodies by this method remains to be seen. It may sound rather fantastic, but scoffers should beware, they may get a big surprise one day!

It is of the greatest interest, therefore, to the mining profession in this country to learn that steps have already been taken, or are in the course of being taken, to provide departments of geophysical surveying both at Cambridge University and at the Imperial College in London. There is no doubt a great field for experiment and research in this comparatively new branch of science, and it is to be sincerely hoped that no time will be lost in bringing these departments into full-scale operation. There surely must be a great and promising future for geophysical methods in mineral exploration and some of us are still hopeful that this will eventually prove to be the long looked for solution of our problems connected with the finding of concealed ore deposits.

Geochemical Methods

It is also claimed that substantial progress has been made of recent years in the use of geochemical methods in the search for mineral deposits, but so far the evidence is not very conclusive. It sounds rather like a return to methods employed in mediaeval times, when the peculiarities of the vegetation growing on the surface were thought to give indications of lodes and veins in the rocks below, as mentioned by Agricola. Now to-day it is said to have been demonstrated that the soil and ground water in the immediate vicinity of an orebody contain small amounts of the metals present in the deposit; also that the presence of these metals can often be detected by the analyses of the leaves of trees and plants growing on the surface above an orebody. By following up these clues it is claimed that the location of the source of these metals can be determined. Radioactivity methods are also expected to prove helpful in the discovery of new mineral deposits, but it would appear that in all these interesting and fascinating conjectures we must await further enlightenment and proof.

Diamond Drilling

Bore-holes are probably one of the quickest and most helpful methods that can be employed in the search for mineral deposits, provided the rock and mineral formations are favourable for drilling. However, in some formations, such as narrow gold veins with spotty values and lodes of uneven texture, too much reliance should not be placed on assay results obtained from bore-holes, although these holes undoubtedly give useful information, particularly negative information. In the case of ores of a homogeneous character, such as large pyritic or mixed sulphide deposits, the results obtained from bore-holes can generally be relied upon, provided a high percentage of core recovery (90 per cent or more) is obtained.

It is sometimes contended that in diamond drilling, if the core recovery is low, there is always the sludge sample to fall back upon. But this is a dangerous procedure to adopt, as it is not at all certain where in the bore-hole some of the values found in the sludge samples have come from and serious salting of the samples may occur. Only in the most favourable ground for drilling are sludge samples likely to be more or less reliable and there the core recovery has also probably been so high that further samples are not necessary. The importance of surveying bore-holes is often overlooked, but this should be made standard practice. It is surprising how holes become deflected from their proper course.

The introduction of the pre-cast crown and the improved design of core-barrels, together with the higher speed at which the machines can now be run, are the outstanding improvements in current drilling methods. Nevertheless, it is somewhat disappointing that more has not been achieved in this respect, as diamond drilling machines and methods have not really advanced very much during the past few decades. Consequently the drilling of bore-holes is still a rather costly business. It would seem that here also is a promising field for research and improvement.

If the cost of diamond drilling could be brought down to a more reasonable figure there would be a much greater amount of drilling done and the chances of finding new orebodies or zones of mineralization would be correspondingly increased. Some of us would like to see many more drills at work than are in use at present, and while by no means advocating the drilling of wild-cat holes we would be quite willing to take rather longer shots at times and even, in some instances, test what are nothing more than mining or geological ' hunches '. But it should always be remembered that in most cases bore-holes are only indicators and that any mineral discoveries made by their means must be confirmed by underground workings.

Exploration by Means of Sub-surface Workings

Once a mineralized body has been definitely located, it becomes necessary to carry out exploration of the deposit in depth as soon as possible by means of adits, shafts, drifts, etc., the making of which is the everyday task of the mining engineer. It is only necessary here to mention that in more recent times this class of work has been greatly facilitated by the development of the internal combustion engine, improved pumping and ventilating equipment, lighter rock drills such as jack-hammers, drill-rigs for shaft sinking, and so forth.

The early stages of an exploratory programme are much simpler affairs to-day than they were 20 to 30 years ago. Then one generally had to transport boilers and heavy equipment through the bush or over rough country, whereas now much lighter equipment and more efficient means of transport are available. Once the mineral deposit is discovered there should be no undue delay in finding out whether it persists in depth and is of sufficient importance to justify full development in order to bring it to the production stage.

The present problem is to find the deposits. The rest is comparatively easy and can be taken in his stride by the properly qualified and experienced mining engineer.

To return now to the query raised at the beginning of this address—namely, how are we going to find the new orebodies which must and will be found if mining engineers are given the necessary scope and freedom of action? As a contribution towards arriving at a solution to this problem I would like to emphasize the following suggestions, some of which have no doubt been made previously but which cannot be repeated too often:—

(1) Intensive research should be carried out in an attempt to develop new or improved methods of geophysical exploration, whereby some of the uncertainties and weaknesses in existing methods can be eliminated.

(2) Operating companies must be encouraged by governmental assistance in the form of relief of taxation to carry out energetic prospecting for new sources of ore in and around existing mine workings by means of geological, geophysical, and diamond-drilling methods.

(3) Exploration companies should be formed to investigate virgin areas which are known to have potentialities, but where old-fashioned prospecting methods of visual surface examination are by themselves useless. In this case prospecting rights must be granted over large areas to justify the considerable expense and scale of operations which would be required. All such expenses and those incurred in bringing any mineral deposits discovered to the production stage should be allowable for taxation purposes as deductions against any future profits.

If these things are done I venture to predict that important discoveries of new mineral deposits will continue to be made and the present unsatisfactory position as regards the future supply of some of the metals may then assume a very different aspect, *but time is getting short.*

Before I close this address I would like to say a few words to the younger members, particularly our Student members, to whom we must look in the future to carry on the good work of finding new mines. This is fundamentally a young man's job. We older members can talk and preach as much as we like, but we cannot now go out into the field and engage in the rough and tumble of exploratory work, as many of us have done in the past, and without such work no new discoveries will be made. To our younger members, therefore, and also to all those boys of school leaving age who are wondering what particular line of endeavour to pursue, may I say that although it may be true that the ' Age of Adventure ' is no longer with us, as in the time of our forefathers, there is still plenty of scope in the Mining and Metallurgical professions for any young man with an adventurous spirit. He will see the world, get all the thrills he wants and at the same time have the satisfaction of accomplishing something of real value to the human race.

As the poet Blake says, ' Great things are done when men and mountains meet '.

Vote of Thanks to the President

Prof. W. R. Jones, in proposing a vote of thanks to the President, said he wished to congratulate the members on having listened to an address which was of the most profound importance to the mining and metallurgical industry. Mr. Laycock's knowledge of the mineral resources of the world and especially those of lead and zinc gave authority to his disturbing statement that the supply of lead, if used at the present rate of consumption, would decrease rapidly after 14 years and that of zinc after 21 years. Doubtless there would be future discoveries of lead-zinc and other ores, but it did seem inevitable that in the future lower-grade deposits would have to be worked than in the past. That was one of the great problems that faced the mining engineer.

There was an almost unbelievable unawareness on the part of Governments, including our own, of the really urgent necessity of inaugurating campaigns for the discovery of new deposits of minerals of economic importance and no effort should be spared to bring this very prominently before them. The various methods which the President had so clearly outlined for the discovery of new mineral deposits should be very widely studied. The whole address was so copious in information and so provocative in outlook that it would form a most valuable part of the Institution's records.

Mr. G. Keith Allen, in seconding the vote of thanks, said that the standard set in the address was of an order of which Mr. Laycock could well be proud and on which the Institution could congratulate itself, for the address would occupy a high place in its records. The address dealt not only with the fundamentals of the metal-mining industries, but also with one of the essentials in the structure of our civilization, and as such it must be their hope that the President's observations would reach beyond the confines of the Institution to those higher levels where the only effective action was possible.

The resolution was carried by acclamation.

The President, in reply, said that congratulations were always pleasant to receive although one might have some doubts as to whether they were fully deserved. Be that as it may, he thanked them very much indeed. He wished to mention that this address was prepared some time ago, since when the question of the future supply of metals had been referred to by many other speakers, particularly in America. Therefore some of his comments might in the meantime have lost some of their freshness and might have become rather trite. However, it was always some satisfaction to know that other people were thinking along the same lines and that one had selected such a topical subject for an address.

ELECTION OF COUNCIL

Mr. P. T. Halsey then presented the Report of the Scrutineers and the result of the ballot for Members of the Council for the Session 1946–47, which was as follows:

JOHN CALDWELL ALLAN.
LESLIE HUGH BARTLETT (*India*).
HENRY HUGH WHITELOCK BOYES (*West Africa*).
ANGUS LEICESTER BUTLER.
TOM EASTWOOD.
SIR LEWIS LEIGH FERMOR.
WILLIAM HENRY CRANSTOUN GEIKIE.
DONALD GILL.
VERNON HARBORD.
ARTHUR HIBBERT.
LAURENCE CARR HILL.
CHARLES ERNEST JOBLING.
WILLIAM RICHARD JONES.
JULIUS KRUTTSCHNITT (*Australia*).
GEOFFREY MUSGRAVE (*Rhodesia*).
WILLIAM ALFRED CYRIL NEWMAN.
JOHN CARROLL NICHOLLS (*Canada*).
RUSSELL JOHNSTON PARKER (*U.S.A.*).
CYRIL EDWARD PARSONS.
PHILIP RABONE.
JOHN HERBERT RICH (*Malaya*).
JOHN ANTHONY SIDNEY RITSON.
RICHARD HUGH SKELTON.
RALPH SHELTON GRIFFIN STOKES.
DAVID ARNOLD THOMPSON.
WILLIAM GEORGE WAGNER.
ARTHUR JOHN WALTON (*South Africa*).
ARTHUR ROBERT OWEN WILLIAMS.

A vote of thanks was accorded to the Scrutineers and the proceedings then terminated.

[redacted]

Subject to revision.] [*A Paper issued on July* 11*th*, 1946, *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, October* 17*th*, 1946, at 5.0 o'clock p.m.

Anglo-American Magnesium Production.

By P. L. Teed, *Member.*

Introduction

Under the stimulus of national emergency great things have been accomplished in every field of applied science and although nothing else can compare either in effort, result, or implication with the Allied epoch-making developments in nuclear physics, metallurgy also has had a war-time history, indicative of the conspicuous part which it has played. Having been officially associated, both in Great Britain and in the United States, with one of these developments—namely, magnesium production—the author is able to indicate in broad outline what was done in this field and to assert that, under the very difficult conditions under which it was accomplished, it was no mean achievement.

Since Sir Humphrey Davy first isolated magnesium in 1808 the history of its development is some 20 years longer than that of aluminium, yet, in spite of this initial advantage in point of view of time, the production of the lighter metal has greatly lagged behind. That this is so is beyond controversy, but as to the technical reasons for it there are many vigorously held and sharply divided opinions. In Table I, the principal physical characteristics of magnesium are set out, while in the Appendix will be found the chemical compositions and mechanical properties of the British and American alloys which are the subject of official specifications.

Table I and the Appendix indicate the reasons for the use of magnesium as an engineering material and also, because of its outstanding electro-chemical potential, suggest that as a reducing agent and also as a sacrificial cathode in giving protection from corrosive attack it might well, and indeed does, have useful applications. One more and indeed an outstanding characteristic has to be mentioned in order to explain why this former Cinderella

TABLE I

PHYSICAL CHARACTERISTICS* OF MAGNESIUM

Crystalline System	Hexagonal, close packed.
Atomic Weight	24·32
Isotopes	3 (24, 25, and 26)
Atomic Number	12
Density at 20° C., 99·9 per cent Mg.	1·738 gm. per cu. cm.
„ „ Melting Point, Solid	1·666 „ „ „
„ „ „ „ Liquid	1·584 „ „ „
Electrical Resistance at 20° C.:	
Cast	4·36 — 4·49 × 10^{-6} ohms per cu. cm.
Extruded	4·43 — 4·60 × 10^{-6} „ „ „ „
Rolled	4·50 — 4·77 × 10^{-6} „ „ „ „
Thermal Conductivity at 20° C.	0·365 cal. per cu. cm. per sec.
„ „ „ 100°	0·351 „ „ „
„ „ „ 200°	0·345 „ „ „
„ „ „ 300°	0·339 „ „ „
Melting Point	649 — 651° C.
Boiling Point at Atmospheric Pressure	1,094 — 1,120° C.
Latent Heat of Fusion	46·5 cal. per gm.
Specific Heat at 20° C.	0·241
„ „ „ 100°	0·252
„ „ „ 200°	0·262
„ „ „ 300°	0·273
Coefficient of Expansion at 0°—100° C.	25·5 × 10^{-6} per °C.
„ „ „ 0°—200°	26·2 × 10^{-6} „ „
„ „ „ 0°—300°	27·0 × 10^{-6} „ „
Young's Modulus	6·4 × 10^{6} lb. per sq. in.
Shear „	2·5 × 10^{6} „ „
Poisson's Ratio	0·33
Potential in relation to the Calomel Electrode in Seawater	1,400 — 1,500 millivolts.

*The figures given have been taken from numerous sources. Where recognized authorities differ by small amounts, the mean of their results has been used, and where divergence is somewhat greater, its range has been indicated.

of the metals advanced so rapidly in prestige upon the strategic stage. During the Spanish Civil War incendiary bombs, largely composed of a magnesium-rich alloy, were dropped by German aircraft with such effect that military observers watching this dismal prelude to Armageddon realized that a weapon of great potentiality had been disclosed, whose mass manufacture must be the concern of those responsible for aircraft armament in their respective countries. Thus a further stimulus to magnesium production came about, which it can now be said has been responsible for the consumption, in the form of incendiaries, of more than half the metal made by Great Britain and the United States during the war period. Finally, its previous use should not be forgotten, for the capacity of the powder to give light singularly useful for photography and for visual observation has had wide application in flash bombs, flares, signals, and a number of similar devices.

Recent History of Magnesium Production

During the war of 1914-1918 the United States possessed a small magnesium industry, of which the output was mainly consumed in pyrotechnics, but of the seven firms then engaged in this activity only two long survived the conclusion of hostilities, while from 1928 to 1940, American production depended entirely on a single firm, whose output in 1939 was but 3,000 long tons.

The making of the metal in substantial amounts in Great Britain was not begun until 1935, when two companies, using entirely different processes, became producers. In the year of the outbreak of war their combined manufacture amounted to 5,000 tons, a figure which, while at the time a cause both of congratulation and satisfaction, was to become of very modest proportions in the light of developments soon to be put in hand on both sides of the Atlantic.

In the United States conditions both before and subsequent to Pearl Harbour were far more favourable for expansion than in Great Britain, for technical staff, constructional and operational labour, electric power, and raw materials were all much more readily available. Thus it came about that at the conclusion of hostilities the two countries possessed the following respective *proved* magnesium production capacity :—

United States, 268,000 long tons/annum, and
Great Britain, 28,000 long tons/annum—

located in some 20 widely-dispersed units, varying in output from under 2,000 to over 50,000 long tons per annum and operating on a wide variety of processes. The war-time plants were not selected, either as regards their productive processes, or as to the area in which they were located, on the basis of economics; the main consideration was to obtain the maximum amount of magnesium in the shortest space of time. Mistakes have been made. Some processes, so encouraging on theoretical consideration or even from laboratory results, have failed to reach any significant output. Competing claims of other war industries for either labour, electric power, or even materials, have sometimes prevented efficient plants from operating efficiently. Nevertheless, now the curtain has been rung down, it can be said that while in the two Allied countries slightly under £110,000,000 was spent by Government agencies in expanding their sources of magnesium production, the actual combined yield exceeded the estimate. From the

Spring of 1943 onwards metal was brought into being at a rate sufficient, and soon afterwards more than sufficient, to satisfy War requirements.

Processes Used

The processes which were selected for plant expansion were to a considerable extent in satisfactory operation before the national emergency arose, but, for purely logistic reasons, a policy of mere expansion of existing plants or the building of similar units in other localities could not be adopted. New, and in two instances virtually untried, processes had to be supported, as they promised to allow of obtaining metal sooner, although admittedly at greater expense, than would have been the case were a mere duplication of contemporary units undertaken.

As a result of these considerations the appropriate British and American departments sponsored processes operating on the following principles:—

(a) Electrolytic reduction, and

(b) Thermal reduction.

The first-named can be divided into two classes—namely, one in which the cell was mainly fed with anhydrous magnesium chloride and the other whose feed was largely composed of a somewhat hydrated salt. The thermal methods were represented by the following sub-heads, in which the outstanding characteristic was the use of a particular reducing agent :—

(a) Calcium carbide,

(b) Aluminium,

(c) Ferro-silicon, and

(d) Carbon,

the last-mentioned process being developed almost entirely by private enterprise.

Raw Materials

To produce magnesium the primary material required for the electrolytic process is magnesium chloride, while for thermal processes either magnesia or calcined dolomite is essential. These substances have been produced from a variety of sources, the following being the most important :

(a) Magnesite,

(b) Dolomite,

(c) Seawater, and

(d) Natural brine.

Magnesite of sufficient purity was available so long as adequate shipping existed, but as the war at sea grimly developed, putting its ever-increasing premium on tonnage both in Great Britain and in the United States, greater attention had to be paid to the use of indigenous materials, the following being, of necessity, sketchy indications of the many processes used to accomplish this end.

Making Magnesia

While high-grade magnesite was available, all that was necessary was to calcine the material in a gas-, coal-, or oil-fired kiln, but, depending on the nature of the process to be subsequently used, the maximum temperature of calcination varied from 900 to 1,250°C. The carbo-thermic process, which is described later, was exceptional in this respect, as it was able to deal equally well with material whatever its temperature of calcination, provided always this had been sufficiently high to drive off the bulk of the combined carbon dioxide. In at least one instance, an impure magnesite was substantially improved prior to calcination by fine grinding followed by froth flotation.

In both countries during the war period, and also previously, very substantial quantities of magnesia had been made from sea-water by the precipitation of the magnesium salts therein, either as a result of double decomposition reactions with slaked lime of the following type :—

$$MgCl_2 + Ca(OH)_2 = Mg(OH)_2 + CaCl_2$$
$$MgSO_4 + Ca(OH)_2 = Mg(OH)_2 + CaSO_4$$

or by using calcined dolomite as an alternative precipitant, which, certainly as far as its lime content was concerned, had undergone hydration. In both cases the precipitate would be subsequently calcined. However, if slaked calcined dolomite were used as the precipitant the total yield per ton of sea-water treated would be approximately double that obtained with slaked lime precipitation, owing to the presence of the initial magnesia content of the former material in the final product.

To those who have not considered the problem, the recovery of magnesia (and ultimately of magnesium) from seawater has been thought to be something akin to magic and at the same time a mere temporary expedient. Fundamentally it is the almost complete insolubility of magnesium hydrate (about 0·0034 per cent at 15°C. for the *alpha* and 0·008 per cent for the *beta* varieties) which makes this remarkable process so effective. Even with a solution of magnesium salts as attenuated as that which exists in seawater the addition of an alkaline hydrate causes a precipitate almost theoretical in amount. The magnesia content after calcination is as high as 97 per cent if precipitation is carried out with very pure lime, such as is obtained from calcined oyster shells, or somewhat lower if less pure lime or calcined dolomite is used for the purpose.

The chemical analysis given in Table II can be regarded as representative of all seawater undiluted by great fresh water additions—such as occur in the neighbourhood of large estuaries

TABLE II

PRINCIPAL CONSTITUENTS OF SEAWATER

Element	*mg/kg*
Chlorine	18,980
Sodium	10,561
Magnesium	1,272
Sulphur	884
Calcium	400
Potassium	380
Bromine	65
Carbon	28
Strontium	13
Boron	4·6

or coastal glaciers. From this table it can be calculated that per cubic mile of ocean water having a density of 64.11 lb./ft.^{3}, the magnesium content is approximately 5,400,000 long tons, an amount greatly in excess of the total production of this metal from its discovery in 1808 until the present time; consequently, all countries having access to the sea have available to them a calculable but inconceivable quantity of magnesium, a point which may not be without significance as regards future developments in lands not otherwise blessed with metallic resources.

MAKING MAGNESIUM CHLORIDE

There are considerable numbers of methods which have been successfully used for the making of magnesium chloride for the electrolytic process, each of which has been adopted to take advantage of some local condition. When the anhydrous magnesium salt is required it is probable that the employment of the following reaction is as effective as any. Magnesia (produced by a relatively low-temperature calcination of magnesite or of magnesium hydrate) and peat powdered together with some potassium chloride and a little aqueous magnesium chloride to produce an oxychloride binder, are mixed together, pelleted, and lightly calcined to burn out the organic matter, thus leaving the inorganic constituents in an extremely porous condition. The material is then fed to a coke-containing electrically-heated shaft furnace, up which gaseous chlorine is passed over the descending pellets, when the following reaction takes place :—

$$MgO + C + Cl_2 = MgCl_2 + CO.$$

The ultimate charge temperature being in excess of the melting point of the anhydrous chloride (712°C.), this can be readily removed by tapping. Owing to an admirable and most surprisingly speedy piece of wartime research, it became possible to give up the use of the organic spacer, with the result that, apart from the direct saving due to peat no longer being required, very great economies in the use of chlorine were also achieved, the combined effect of which, it has been stated, reduced production costs by 17 per cent.

An alternative method of producing this salt, which has had large-scale application, is to carry out the drying of hydrated magnesium chloride crystals ($MgCl_2 . 6\,H_2O$), made from magnesium hydrate derived from seawater by two or more stage drying (or by a variety of methods yet to be described), taking the dehydration by simple heating to that point at which the hydrous salt breaks up into magnesia and hydrochloric acid. At this stage further drying is carried out by circulating dry hydrogen chloride over the material. This method was effective in producing the end product required, but corrosion difficulties both in the dryer and circulators were extensive. However, since developments in the electrolytic cell have now permitted the use of a magnesium chloride which can be expressed as $MgCl_2 .\ 1{\cdot}25\ H_2O$, the last mentioned method of making the anhydrous chloride is no longer employed.

Yet another process consisted of treating magnesium hydrate, derived from natural brines or seawater, with a solution of calcium chloride and carbon dioxide, when a reaction of the following type takes place :—

$$Mg(OH)_2 + CaCl_2 + CO_2 = MgCl_2 + CaCO_3 + H_2O.$$

After separation of the calcium carbonate from the magnesium chloride by filtration the last-named was concentrated by evaporation and subsequently dried by a variety of methods—such as submerged combustion evaporation, spray or shelf driers, or by a remarkable combination of the last two—in which concentrated liquor, containing about 50 per cent of magnesium chloride, was sprayed on six times its weight of previously-dried chloride. The resulting mixture (which remained solid), was further reduced in water content in a shelf drier to the $MgCl_2 .\ 1{\cdot}25$—$1{\cdot}5\ H_2O$ stage, when it was suitable as feed for the type of cell used for its electrolysis.

A variant of this technique which has been successfully employed is to treat calcined dolomite with an aqueous solution of calcium chloride and carbon dioxide, when the magnesia goes into solution as the chloride while the lime is converted to carbonate. Subsequent to filtration and concentration the hydrated magnesium chloride was dried as has been previously described.

These methods of obtaining magnesium chloride by means of calcium chloride, of course, made their appeal where the latter salt was being obtained as a by-product from some other industry—such as the ammonia-soda process.

A somewhat analogous method, which was employed with good results, was to treat calcined dolomite with by-product hydrochloric acid, to neutralize the resulting solution with magnesia, then to remove the lime by passing carbon dioxide (originally liberated by the calcination) through the liquor and finally to concentrate in the usual way. By-product hydrochloric acid was also employed to dissolve magnesium hydrate obtained by lime precipitation from seawater, while, in a single case, magnesium chloride was made direct from natural brine by crystallization.

Calcination of Dolomite

For the process in which metallic magnesium was made directly from dolomite the mineral was calcined in a gas-, oil-, or coal-fired cement kiln, the choice of fuel depending on the locality in which the plant was erected. This stage of the process really calls for no comment, but there is one point which, although not of technical importance, might be mentioned. When coal firing was used, this led to a substantial rise in the R_2O_3 content of the calcined product, due, no doubt, to ash contamination, but as this still remained below 1 per cent the matter was one of interest rather than of consequence.

Magnesium Production

Electrolysis.—The principal processes used during the war period for the manufacture of the raw material of magnesium production having been described, it remains to outline the methods whereby it was converted into metal. On both sides of the Atlantic, the major contribution towards satisfying the Allied magnesium needs has been made by the electrolytic process. While three techniques actually reached the producing stage, one of them, by the time the state of the war made its continued operation unnecessary, had made no substantial amount of metal and was not likely to do so without further development (which has not been put in hand) and consequently its technique is not described.

While anhydrous magnesium chloride alone can be decomposed electrolytically there are practical objections to attempting this on the manufacturing scale. The melting point of the salt is high—

namely, 712°C.—while its electrical conductivity is low. Both these defects can be reduced by the addition of other chlorides whose decomposition voltage is higher than that of magnesium chloride. As regards what chlorides to add, however, there are strongly divergent views, based to some extent on technical considerations and in part on such practical ones as the local availability of possible materials.

On the technical side, apart from the decomposition voltage and influence on melting point which the addition may have, there is a further important point—namely, its effect on the specific gravity of the electrolyte—for, certainly with cells of the present design, it is essential that the latter must be denser than magnesium itself at the same temperature, because were the reduced metal to sink to the bottom of the cell, this would introduce at least two further difficulties: it would be hard to remove the metal and, further, since sludge accumulates at the base of the compartments, the magnesium would be contaminated with it, thus necessitating an additional and by no means easy purification process.

It is probable that there is no subject connected with the technique of magnesium production by electrolysis on which opinions differ more widely than the composition of the electrolyte. Views have been sincerely and vigorously expressed which are entirely contradictory, although they may nevertheless be soundly based, for the chemistry of the cell is greatly influenced by such complex matters as over-all dimensions, electrode spacing, method of heating the electrolyte, conditions of electrolysis, etc. Undoubtedly one important point is the degree of turbulence of the electrolyte. In general it can be said that, under otherwise equal conditions, with a quiescent bath there is greater tendency for sludge formation than when there is considerable fluid movement. It was due to cell design that one producer, who initially found it essential to use a cell feed composed of anhydrous magnesium chloride, subsequently was able to operate using the $MgCl_2 . 1{\cdot}25\ H_2O$, the manner of preparation of which has already been briefly outlined.

Probably an electrolyte having the following composition has been more extensively used than any other: Magnesium chloride, 25 per cent; calcium chloride, 15 per cent, and sodium chloride, 60 per cent. This material has been decomposed in a steel cell 6 ft. deep, 5 ft. wide, and 11 ft. long, which, in addition to containing approximately 10 tons of fused electrolyte, formed part of the internal cathodes. Current through such a cell was 30,000 to

70,000 ampères, the voltage drop being about 6·8, and the electrolyte temperature about 700°C. This type of cell was and remains unique in that it stood in a gas-fired refractory-brick furnace, which apart from improving the efficiency of electrolysis, allowed of more flexibility in operation than would have been the case were heating due entirely to the ohmic resistance of the electrolyte itself. The anodes consisted of 22 graphite rods 8 in. in diameter and about 9 ft. in length, which, for reasons elaborated later, were consumed at a rate of about one-tenth of a pound per pound of metal produced. The collection of the metal in the cell was accomplished by inverted troughs, placed under the bath level, which caught it as it rose from the cathodes, led it to metal wells situated in the front of the cells, whence it was dipped two or three times daily and cast, without further refinement, into ingot.

Although only the magnesium chloride content of the electrolyte is decomposed, the question of the chemical composition of the cell feed is one of the utmost importance, for the behaviour of material other than the magnesium salt requires the most careful consideration. Should the feed contain compounds of aluminium, copper, nickel, zinc, and silicon, these are reduced, either by electrolysis or by contact with the metallic magnesium, and alloy with it. Iron and boron, although reduced to the elemental state, tend to drop through the electrolyte into the sludge, while manganese to some extent alloys with the floating magnesium, but also in part settles out at the bottom of the cell. Chlorides in the feed, whose decomposition potentials are above that of the magnesium salt, accumulate, thus leading ultimately to the necessity of removing electrolyte by dipping. Water introduced into the cell gives rise to magnesia and hydrochloric acid, some of the first named going into solution, while another portion is converted into anhydrous chloride by chemical inter-action between the carbon anodes and the nascent chlorine there liberated. About half of that which was originally formed would find its way into the sludge, a typical composition of which has been given by one operator as being about 18 per cent magnesia, several per cent metallic magnesium, less than 1 per cent heavy metals, and the remainder entangled electrolyte.

Thus, owing to the necessity of removing sludge and of dipping out electrolyte (which, it may be noted, can be usefully employed as the raw material of foundry flux manufacture), there is great scope for research, with the two-fold object of finding a mixture of low sludge-making propensities and of producing the most effective cell feed. Further, there can be no doubt that when circumstances warrant such development, existing electrolytic cells can be improved in design. In some the surface-volume ratio leads, in spite of insulation, to heat losses capable of reduction, while dimensional modifications and changed electrode spacing might well provide a degree of turbulence in the electrolyte, decreasing sludge formation without reduction in the rate of metal production.

With a cell of the type referred to, the following would be representative of the magnesium chloride for the feed: Magnesium chloride, 73·70 per cent; magnesia, 2·30 per cent; sodium chloride, 1·50 per cent; potassium chloride, 0·25 per cent; calcium, 0·22 per cent; sulphur, 0·17 per cent; manganese, 0·02 per cent; iron, 0·01 per cent. Owing to the combined, and to some extent indeterminate, influence of sludge removal and electrolyte dipping, other salts would have to be added to maintain an acceptable bath composition.

Operating in the way that has been indicated, the metal removed from the cell would have the approximate composition given in Table III. Such metal would be produced in the cell described

TABLE III

	Per cent
Aluminium	0·003
Manganese	0·04
Calcium	Nil
Zinc	,,
Iron	0·03
Silicon	0·005
Copper	0·003
Lead	Nil
Nickel	,,
Cobalt	,,
Silver	,,
Sodium	,,
Boron	less than 0·0001
Cadmium	0·00001 to 0·00004
Magnesium	Remainder

at a rate of about half a ton per diem for an expenditure of electrical energy of 18,000 to 19,000 kWh per long ton, which compares very favourably with the contemporary power consumption required for the manufacture of virgin aluminium. If anhydrous magnesium chloride made with gaseous chlorine derived by electrolysis were used as the cell feed, then there would be a substantial addition to the figure given.

Purification of Crude Ingot.—While metal of the composition already given is entirely suitable for the manufacture of casting alloys, which by the nature of their manner of fabrication are not used in extremely thin sections, there is a feeling, particularly in the United States, that for the making of wrought parts—such as would be employed for the manufacture of sheet—its purity is insufficiently high to resist corrosive conditions of the type which would be experienced in carrier-borne aircraft, float-seaplanes, and flying boats.

Metal for making aluminium-containing alloys for this purpose can be purified by the following long established technique. After the cell metal has been remelted under a suitable flux it is raised to a temperature of 740°C. and manganese equal to about ¾ to 1 per cent of its weight added. This manganese addition can be made in a variety of ways, the most usual being to add the metal in the form of a flux containing manganese tetrachloride. As a result of the exothermic reaction between the magnesium and the manganese salt, the last-named is reduced to metal, the temperature of the melt raised about 10°C., and the manganese taken into solution. The other common method of carrying out this technique is to add the manganese in the form of an appropriate aluminium-manganese hardener. In this case there is, of course, no exothermic reaction and therefore the melt should be slightly hotter than when the addition is made by means of a manganese salt.

Another method of adding the metal which has been used to some extent is to plunge the appropriate quantity of finely-divided electrolytic manganese into the melt. Results obtained in this way have not always been satisfactory, however, success seeming to depend on using finely-ground metal.

Finally, yet another way of introducing the manganese has been employed. A briquette composed of finely-divided electrolytic manganese, manganese dioxide, and magnesium powder is plunged into the melt. Owing to the reaction of the magnesium with the manganese dioxide being exothermic there is a certain degree of turbulence, which appears to give a wide and effective distribution of the addition.

Be the manganese introduced as it may, subsequent to its having been added the melt is allowed to stand while the temperature falls about 70°C. During this period (which, when two tons of metal are being treated, may be of the order of 45 minutes to 1 hour) an iron-manganese complex forms in the melt and settles out. Four-fifths of the melt can then be pumped off, when it will have a composition of the order shown in Table IV. The 'heel' left in the pot can be added to similar material derived from other applications of the purification treatment and the whole retreated with results very similar to those described.

TABLE IV

	Per cent
Aluminium	0·003
Manganese	0·20
Zinc	Nil
Iron	0·002
Silicon	0·005
Copper	0·003
Nickel	Nil
Cobalt	,,
Boron	0·00003
Cadmium	less than 0·0001
Magnesium	99·787

In concluding this outline of the manganese purification technique it should be emphasized that while this was initially employed for the purpose of reducing the iron content of electrolytic magnesium, the process is equally applicable to the treatment of metal derived by other means.

THERMAL PROCESSES

Calcium Carbide Reduction.—One of the pioneer thermal techniques for manufacturing magnesium which was in full-scale operation before the war is that which is generally termed the calcium carbide process, for it was by this substance that the reduction of the magnesia was achieved. The salient features of the method consisted of mixing together in powdered form (85 per cent to pass 200-mesh) magnesia, calcium carbide, and fluorspar in such proportions as concerns the first two as are dictated by the equation :—

$$MgO + CaC_2 = Mg + CaO + 2C.$$

However, as neither the magnesia nor the calcium carbide available were without impurities, due allowance was made for this and a mixture of the following composition was made : Magnesia (containing 85 per cent MgO), 34·5 per cent ; calcium carbide (containing 75 per cent CaC_2), 62·2 per cent, and fluorspar, 3·3 per cent. Such a mixture contained 17·5 per cent of potential metallic magnesium, of which about 65 per cent would be ultimately recovered as ingot metal ; thus, to make 1 ton of magnesium the following quantities were required :—Magnesia, 3·0 tons ; calcium carbide, 5·4 tons, and fluorspar, 0·3 ton. This dry and well-mixed powder was pressed into small circular briquettes 2 in. in diameter and $\frac{7}{16}$ in. in thickness, having an approximate weight of 2½ oz. These were placed in a mild-steel retort about 14 ft. long by 14 in. internal diameter and 1½ in. wall thickness, which was protected on its exterior surface by a welded sheath of heat-resisting steel (22 per cent nickel, 25 per cent chromium). The retort, having been filled with briquettes to a certain height, was closed by a cap about 8 ft. long, which contained two semi-circular mild-steel plates about 34 in. long, which rested on a ledge in the top of the retort and acted as a condenser for the magnesium vapour which was subsequently produced. This retort with its cap, condenser, and charge of briquettes weighed about 1 ton 12 cwt., of which the latter, approximately 3,200 in number, weighed about 4·5 cwt. or 14 per cent of the total.

Twelve briquette-filled retorts of the type described were placed vertically in a producer-gas-fired furnace so that the cap portion containing the condenser plates protruded above the top of the furnace and was subjected to a fan-induced draught. On the retorts being put into the furnace, they were connected to a vacuum pump which gradually reduced the internal pressure to about 1mm. of mercury. When, after about 8 to 9 hours the temperature of the charge had risen to approximately 900°C. the reaction already indicated began to take place, but owing to the low pressure, the resulting magnesium at once vaporized and subsequently condensed on the plates in the fan-cooled portion protruding from the furnace. The decreasing mechanical strength of the steel of the retorts with rise in temperature fixed 1,120°C. as a practical operational limit.

In the war period the time they were maintained at this temperature depended more on the urgency with which magnesium was required than on what it cost. In Table V one aspect of this problem is disclosed, but the relationship of the economics of the process to the furnace cycle is too complicated to be explained in

TABLE V

INFLUENCE OF FURNACE TIME CYCLE ON THE EXTRACTION OF MAGNESIUM

Furnace Cycle, Hours	*Possible Number of Cycles per Week*	*Lb. of Magnesium per batch of 12 retorts*	*Possible Output of one 12-retort Furnace (Tons/annum)*	*Per cent Extraction of Magnesium from Magnesia*
36	4·67	950	102·5	100·0
32	5·25	925	112·5	97·5
28	6·00	900	125·2	94·6
24	7·00	800	130·0	84·2
22	7·64	750	132·6	78·9
20	8·40	700	136·2	73·7
18	9·34	600	130·0	63·2
16	10·50	500	121·6	52·6

this concise manner. At the completion of the selected cycle, the retorts were removed from the furnace (their places being taken by freshly-charged units), and allowed to cool for six hours, after which the caps could be taken off, the condensed magnesium removed, melted under flux, and cast into ingots.

A point of interest is the trouble introduced by the presence of the alkaline metals in the briquettes. These were reduced and condensed in the coolest portion of the cap; consequently, if substantial quantities of the former were present, since their ignition temperature is much lower than that of magnesium, the cooling of the retort had to be prolonged for fear that when letting back the air prior to the removal of the cap the magnesium itself might be set on fire from the burning alkaline metals. This trouble in the plant under consideration could be got over only by having virtually sodium- and potassium-free raw materials or by prolonging the cooling period. In describing a later thermal process, this matter is dealt with again.

Under the influence of the atmospheric pressure acting on the hot retorts, these tended to decrease in length and also to cave inwards, thus greatly reducing their volume. On this happening to a serious extent the retort would be removed from the furnace and after emptying and reheating would be restored approximately to its original shape by inserting a mandrel under hydraulic pressure and subsequently drawing the retort through a die. This method of dealing with collapsed retorts was effective, but involved costly operations with expensive plant. (In referring to a later retort process, a singularly neat and cheap way of restoring collapsed retorts is described).

The calcium carbide process, like all pioneer processes, can be criticised on many grounds. The reducing agent was such that its employment prevented the making of a briquette of high potential magnesium content. The need for charging and discharging the retorts outside the furnace necessitated having an embarrassing number of overhead cranes, while the fact that the retorts could not be discharged until they were almost cold, led to the heat input of every cycle being virtually lost, and the maintenance of the retorts was a matter of great difficulty. Grave as these defects were from the economic point of view, when magnesium was in short supply, one of the plants operating on this process regularly delivered more than its designed output, an achievement deserving both praise and gratitude.

Aluminium Reduction.—In at least three different types of full-scale plant magnesia has been reduced to the metallic state by its reaction with aluminium, in accordance with the following equation :—

$$3MgO + 2Al = 3Mg + Al_2O_3.$$

The magnesia, ground to pass 50-mesh, was mixed with rather finer-blown aluminium powder, made from low-grade scrap. From what was virtually a stoichiometric mixture, to which a small quantity of calcium fluoride was sometimes added, briquettes were made. However, this turned out to be a matter of very considerable difficulty, involving, prior to success being attained, the necessity for greatly modifying the standard types of machine. The briquettes were put into either very ingenious continuously-operating furnaces (an intermittent type), or into fixed retorts, all of which were under vacuum while the briquettes themselves were heated to about 1,250°C. The furnaces and the retorts (as already described in conjunction with the calcium-carbide process) communicated with a cold vacuous space in which the magnesium vapour condensed and, after removal therefrom, was remelted under flux and cast into ingots.

The process had the advantage of using a briquette of high potential magnesium content, but (and for many purposes this was not a matter of consequence) the condensed metal contained all the zinc and about half the lead which was present in the aluminium powder. Where electric heating of the briquettes was carried out, the consumption of power was about 13,000 kWh per long ton.

An alternative demand for aluminium powder led to these magnesium plants being curtailed in their activities, and, as a consequence, they scarcely reached the degree of development which was originally anticipated ; nevertheless, they did demonstrate the complete practicability of the conceptions on which they were based.

Ferro-Silicon Reduction.—Of the thermal reduction processes, that involving the use of ferro-silicon as the reducing agent was the last in the Allied World to get into production, but it has probably been responsible for a yield of higher purity metal than any other.

Two features are outstanding—its use of calcined dolomite as its source of magnesium and of ferro-silicon as its reducing agent. The reaction can be expressed :

$$2MgO.CaO + Si = 2Mg + (CaO)_2\,SiO_2.$$

This has been known for a number of years, indeed as early as 1915, and it is probable that, in the legal sense, it was then no longer novel. Plant for carrying it out on a manufacturing scale was developed from 1934 onwards by the I.G. Farben-Industrie A-G and by at least one of its associated concerns, whilst in the Allied World the reaction had received laboratory investigation by a number of firms. Since, however, under the conditions under which it was studied, it was found that a temperature of about 1,400°C. was necessary to obtain satisfactory yields, the practical difficulties which this involved initially led to alternative processes being employed. That the method became a large-scale producer of magnesium of high purity was due to the appreciation of the fact that if the reaction were carried out in a really high vacuum, temperatures lower than those which had hitherto been regarded as essential could be used satisfactorily. With a vacuum of about 0·05 mm. of mercury, it was clearly demonstrated that, at a temperature of 1,160°C., the reaction proceeded at such a speed as to justify the design and construction of several large-scale plants based on this operational conception.

The following can be regarded as a representative analysis of the calcined dolomite employed: Magnesia, 40·5 per cent; lime, 58·1 per cent; R_2O_3, 0·8 per cent, and insoluble 0·6 per cent.

The ferro-silicon used was, from the chemical point of view, specified as containing not less than 75 per cent Si, and in practice it was generally about 2 per cent above this figure. As in other thermal reduction methods, a small quantity of high-grade fluorspar was added to the mixture. The manner in which this salt functions is obscure, but users of this and similar processes are, in the main, satisfied that its introduction is beneficial.

The dolomite, after calcination, was ground so that 60 per cent would pass a 200-mesh screen, the ferro-silicon being ball-mill reduced wholly to pass a 55-mesh screen; the fluorspar, in general, was slightly finer than the calcined dolomite. This is a point without technical significance, but it so happened that in this degree of subdivision it was readily procurable. The materials were dry mixed in the following proportions: Calcined dolomite, 80·0 per cent; ferro-silicon, 17·2 per cent, and fluorspar, 2·8 per cent. On the basis of the analysis already given the silicon is about 18 per cent in excess of stoichiometric requirements, but practical trials were deemed to justify this apparent extravagance.

Subsequent to mixing in several plants the powder was fed to 'Belgian' rolls and converted into strong hard briquettes, which had the following characteristics: Apparent density (air/oil), 2·03; bulk density, 62·7 lb./ft.^{3}; magnesium content, 19·4 per cent.

In the majority of the plants using this process these briquettes, freed from dust by screening, were charged into paper bags and loaded into a hopper, from which they were thrust into an externally-heated, fixed, horizontally-inclined, and heat-resisting steel retort of about the following characteristics: Overall length, 9 ft. 8 in.; length within furnace, 7 ft. 8 in.; length outside furnace, 2 ft. 0 in.; internal diameter of portion within furnace, 10 in.; internal diameter of portion outside furnace, 11 in.; wall thickness, 1¼ in.; weight of retort, 1,200 lb., and weight of charge, 200 lb. As already described in reference to the calcium-carbide reduction retorts, the portion protruding from the furnace acted as the condenser both of the magnesium and of the alkali metal vapour, but in the case of the ferro-silicon process, the cooling was done by water jacketing.

After the retort had been charged it was closed and connected to a vacuum service, sometimes purely mechanical, but tending, in the light of experience, to be composed of mechanical pumps for pulling down and of mercury or oil-diffusion pumps for holding. Within 20 minutes of closing the retort a vacuum of the order of 0·5 mm. of mercury would be obtained, while before the end of the cycle, which was of nine hours duration, the pressure would be reduced to as low as 0·05 mm.

At the completion of the cycle, the duration of which was fixed by practical considerations, approximately 84 per cent of the magnesium in the charge would be attached to a mild-steel tube fitting into the condenser, while practically all the sodium and potassium would be located on a plate nearer to the closing cap of the retort (and therefore in a cooler position). On letting air back into the retort the cap, together with the plate on which the alkaline metals were condensed, was speedily taken off; then, risk of setting fire to the magnesium having been eliminated, this could be removed without undue haste, the spent briquettes withdrawn, either by hand or mechanically, and the retort recharged as before.

The condensed magnesium, generally known as a 'crown', was of a higher purity than that made by any other full-scale plant and would in general have an analysis of the type indicated in Table VI. The remelting of 'crowns' for making ingots was done by employing a standard flux containing anhydrous magnesium chloride (with sodium, potassium, and, perhaps, other chlorides), when there would be reaction between the calcium in the magnesium and the magnesium chloride in the flux with the result that Table VII would be a representative analysis of the resulting ingot.

TABLE VI

	Per cent
Aluminium	0·005
Manganese	0·002
Calcium	0·01—0·1
Zinc	Absent
Iron	0·002
Silicon	0·004
Copper	0·001
Lead	0·004
Nickel, Silver	Not detectable by spectroscope
Sodium	less than 0·0005
Magnesium	Remainder

TABLE VII

	Per cent
Aluminium	0·005
Manganese	0·002
Calcium	0·004
Iron	0·002
Silicon	0·004
Copper	0·001
Lead	0·004
Nickel, Silver, Sodium	Not detectable
Magnesium	99·982

In the large-scale operation of this process, employing electric heating for the retorts, the consumption of materials and power per long ton of crystal metal were of the following order:

Materials:	
Raw Dolomite	9·3 tons
Ferro-Silicon	1·1 „
Coal for Calcining Dolomite	1·7 „
Power:	
For Heating Retort	11,000 kWh
For Manufacture of Ferro-Silicon	11,000 „
	22,000

Thus the power consumption exceeds that of the electrolytic process, except where anhydrous magnesium chloride is used (necessitating the employment of elemental chlorine derived from the electrolysis of brine for its manufacture). From the commercial point of view the ferro-silicon process is also a larger power consumer than when magnesia is reduced by means of aluminium powder made from low-grade scrap. If, however, the electrical energy employed to produce the aluminium were included in the balance sheet it would compare extremely favourably. Apart from the question of power and materials, the cost of both of which is so greatly influenced by local conditions, the notable fact remains that as far as large-scale magnesium production is concerned, the ferro-silicon process gives a product which is normally purer than that made by any other process so far described.

In concluding this description of one application of this process, mention should be made of the ingenious method which was evolved for dealing with collapsed retorts. This problem has already been referred to in relation to the calcium carbide process, where the effects of atmospheric pressure on the hot plastic retorts was dealt with by removal from the furnace, re-heating, and mechanical processing with expensive tools. In one of the ferro-silicon plants the simple but effective idea was conceived (and subsequently universally adopted) of blowing out the distorted retort *in situ* in the furnace. When this treatment was required, a cap was secured to the retort which was connected to a pneumatic supply at 100 lb./in.^{2}. With skilful application of the air pressure the retort was surprisingly rapidly blown out to approximately its original dimensions. This extremely simple plan led to an enormous saving of man-hours and made the installation of costly hydraulic presses entirely unnecessary.

Carbon Reduction.—For more than half a century it has been appreciated that magnesia could be reduced to metal by means of carbon at temperatures in excess of 1,800°C., in accordance with the following equation :

$$MgO + C = Mg + CO.$$

It was also realized, however, that below that temperature the reaction was reversed and that metallic magnesium was oxidized by carbon monoxide. Consequently it was initially felt that it was impossible to base a process for the production of the metal on the above simple but reversible reaction. Owing, however, to the exercise of much scientific and engineering ingenuity, at least three techniques have been evolved, which, in a large measure, suppress the reversibility of the reaction.

In each of these variants the first stage was the same, or very nearly so. Magnesia, ground so that 85 per cent would pass 200-mesh, was mixed in approximately stoichiometric proportions with some carbonaceous material—such as anthracite, petroleum coke, and/or steam coal—and briquetted or merely worked into a stiff paste with a tar or oil binder and fed into carbon arc furnaces, varying in size from 1,000 to 8,000 kW, some operating with single- and others with three-phase current. As a result of the temperature of the arc the bulk of the feed would be converted into vapour in accordance with the equation and, together with gas brought into the furnace *via* the electrode holders (introduced solely for cooling), would sweep out the furnace, carrying a fair proportion of the original charge in suspension.

At this stage, the techniques, hitherto very similar, differ decisively. In one variant the gases leaving the furnace were 'shock cooled' by large turbulent additions of hydrogen at atmospheric temperature, thus not only rapidly lowering the temperature below that at which 'back reaction' was of significant proportions, but also mechanically separating possible reacting molecules. This hydrogen, when subsequently freed of matter in suspension, would of course be contaminated with carbon monoxide, which would be largely removed by cooling the gas and passing it over activated carbon, after which the purified gas would be re-circulated for shock chilling purposes.

In a locality where natural gas was available, this was used for shock-cooling instead of hydrogen; it was freed from about 0·7 per cent carbon dioxide by mono-ethanol-amine, and of water vapour by activated alumina, when it had the following composition: Methane, 92·0 per cent by volume; ethane, 4·3 per cent; propane, 1·4 per cent, and nitrogen, remainder. Owing not only to contamination with the carbon monoxide of the furnace gases, but also to the hydrogen brought into the furnace *via* the electrode holders, the composition of the outgoing gas would be approximately: Hydrogen, 10·0 per cent by volume; carbon monoxide, 10·0 per cent; methane, 75·0 per cent; ethane, 3·0 per cent, and nitrogen, remainder. Some of this would be added to purified incoming gas and would be recirculated, while the other portion would be sold for kiln firing to a neighbouring cement company.

In the third method of stopping back reaction a standard mineral oil was sprayed into the gases as they issued from the furnace. Some of this oil would be merely volatilized and, owing to its latent heat, would give rise to an increase in the cooling effect, but some would be cracked to such a degree as to constitute about 60,000 cu. ft. of permanent gas per ton of metal produced, and this would ultimately become available for heating or other purposes in other portions of the plant.

Subsequent to whichever of these techniques was employed for stopping back reaction, the cooled gases would pass on to a settling chamber and would ultimately be passed through a bag filter. The following are representative analyses of the product derived by gaseous shock cooling: Metallic magnesium, 47 to 50 per cent; magnesia, 33 to 25 per cent; carbon, 10 to 20 per cent, and ash, 10 to 5 per cent. When oil was employed as the coolant the first product would have about the following composition: Oil, 65·0 per cent; metallic magnesium, 14·0 per cent; magnesia, 5·0 per cent; carbon, 15·0 per cent; insoluble, 0·4 per cent, and R_2O_3, trace.

After freeing this sludge from oil by distillation, the following analysis is typical of the residue: Metallic magnesium, 40·0 per cent; magnesia, 14·8 per cent; carbon, 42·9 per cent; insoluble, 1·1 per cent, and R_2O_3, remainder.

Somewhat surprisingly, for each of the techniques, the consumption of electric power in terms of the *metallic* magnesium content of the product was approximately the same—namely, 5·0 to 6·5 kWh per lb. of metallic magnesium.

The product derived by gaseous shock cooling was pyrophoric, while that obtained from the oily sludge became so on the distillation of the liquid; consequently, the material, however it arose, was extremely difficult to deal with, requiring both ingenious mechanisms and exemplary care in operation. While identical techniques were not adopted, the product in each case was briquetted. In the cases in which gaseous shock cooling was employed about two tons of briquettes were loaded into an electrically-heated retort, weighing with its charge about 11 long tons, and the magnesium content distilled out under reduced pressure. This operation, which required about 2·1 kWh per lb. of magnesium produced, gave a condensate which, on remelting under standard flux, would, under favourable circumstances, produce an ingot of about the composition shown in Table VIII.

TABLE VIII

	Per cent
Aluminium	0·003
Manganese	0·005
Calcium	0·005
Zinc	0·001
Iron	0·001
Silicon	0·001
Copper	0·0005
Lead	0·005
Nickel	Nil
Cobalt	,,
Silver	,,
Sodium	,,
Magnesium	99·9785

Where oil shock cooling was used a different method of treating briquettes was employed. Instead of being distilled *in vacuo* at a relatively low temperature, they were heated at atmospheric pressure to 1,100°C. in a shaft furnace through which hydrogen was circulating. The gas swept the vapours into a condenser where liquid metal condensed and was subsequently tapped off for ingoting.

Summarising this carbo-thermic process (which has been developed almost entirely by private enterprise at an expense in Allied countries of somewhat over £5,000,000), it has, at least in one of the gaseous shock chilling units, attained *technical* success. Metal of higher purity than is given by the electrolytic process has been produced in substantial quantities over long periods at a power consumption of approximately 18,000 kWh per long ton. However, at the present stage of its development, from the briquetting of the arc furnace condensate to the casting of the ingot, it can be doubted whether it is as yet on an economic basis. The retorting procedure was extremely consumptive of manpower, and, since most of the sensible heat of the retort and charge was lost with the treatment of each batch of briquettes, there is considerable scope for achieving economies, such as are believed to be essential if the process is to operate on a competitive basis in time of peace.

Conclusion

In attempting to review within the compass of a single paper the great achievements of a number of teams of technical men inspired with the necessity of producing magnesium with the very minimum of delay and without undue regard to cost, the author has probably undertaken too ambitious a task ; yet since within the space of four years the combined Anglo-American productive capacity was increased from 6,000 to nearly 300,000 long tons per annum, this fact deserves to be put on record, and the technical achievement of men often subjected to unfair criticism described, even if only in the inadequate terms at his command. The overall picture of magnesium production during the war is one of a great task well done.

*** *Extra copies of this paper may be obtained at a cost of 1s. 9d. each, at the office of the Institution, Salisbury House, Finsbury Circus, London, E.C. 2.*

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The Chemical Compositions§ and Mechanical Properties of British and American Magnesium Alloys

Chemical Composition

British

APPENDIX

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Chemical Composition

British

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The Chemical Compositions and Mechanical Properties of British and American Magnesium Alloys

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British

THE INSTITUTION OF MINING AND METALLURGY

SEVENTH ORDINARY GENERAL MEETING

OF THE

FIFTY-FIFTH SESSION

Held in the Rooms of the Geological Society, Burlington House, Piccadilly, W.1,

ON

Thursday, April 11th, 1946.

Mr. G. F. LAYCOCK, *President*, in the Chair.

DISCUSSION

ON

A Survey of the Deeper Tin Zones in a Part of the Carn Brea Area, Cornwall.

By BRIAN LLEWELLYN, *Member*.

The President said that the paper to be submitted for discussion was 'A Survey of the Deeper Tin Zones in a Part of the Carn Brea Area, Cornwall'. He hardly needed to say how much the paper would appeal to all Cornishmen and all those who, like himself, had learned their mining there and had spent some most interesting years in that delightful county. The paper raised many matters of primary importance to the Cornish mining industry and, he felt sure, would lead to a most interesting discussion. He added that they were very pleased to see many friends from Cornwall among their guests.

Mr. Brian Llewellyn, in presenting the paper, exhibited a blackboard sketch (Fig. 4) to illustrate the statements made in the paper concerning the productiveness of lodes, extension in depth, and history of the mines. He went on to say that when it was first suggested by a former colleague that he should present this paper he was somewhat reluctant. He felt that not only did the plan suffer from many imperfections due to war-time conditions, but that the substance of the paper was in the concise terminology of a report which it would be difficult to put into the more narrative form usual in papers published by the Institution. He also felt that he might be laying himself open to a charge of lack of caution, for Cornish mining was a controversial subject which had not hitherto been more than briefly touched upon in any previous paper read before the Institution.

The time seemed opportune, however, to have this theme brought up for discussion. It was of far-reaching importance to all mining engineers and to many others. During the war just ended tin production was in a far weaker position to meet an emergency call

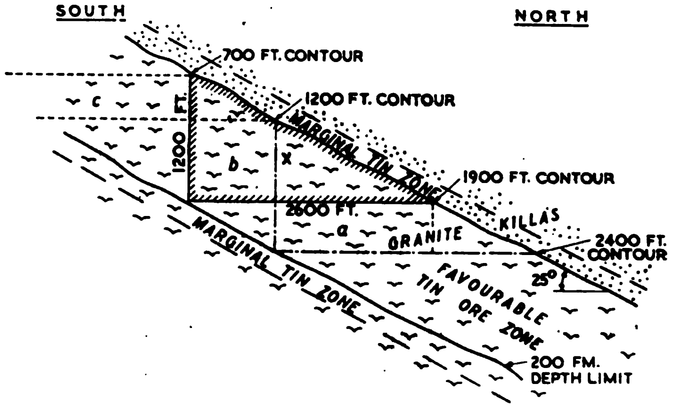


FIG. 4.—Diagrammatic cross section. (The contour lines are those shown on the plan.)

Triangle X: Section through triangular prism referred to on page 12 of paper.

Rectangle a: Area in which all fully blocked out development between 1,900 and 2,400-ft. levels will lie within the favourable depth zone.

Rectangle b: Corresponding area for development between 1,200 and 1,900 ft.

Rectangle c: Corresponding area for development between 700 and 1,200 ft.

than in the earlier war. Now tin was in short supply and so were good tin miners, while, at the same time, it was extremely difficult to forecast the future production of tin or future prices.

Use had been made of all sources of information which were available to him during the time he was engaged upon the work, extending over several months. It was his earnest desire that members who had information on tin mining in the Carn Brea area should take part in the discussion.

From earliest recorded times mining had been the great industry of Cornwall. Was Cornish mining doomed to extinction? He himself believed that Cornish mines could be so modernized that they could compete with any other tin lode-mines anywhere else in the world, through the agencies of low power costs and increased efficiency by higher contract earnings, harder work, and scientific management, provided also that taxation was equitable. Many excellent papers on taxation had recently been published by the Institution and the Council had performed a notable service in urging the necessity for fair treatment of the metal-mining industry in this matter. In the past taxation had been levied without consideration of the needs of the mines and it had been impossible for the managements to provide adequately for the exploratory and other capital expenditure essential for the continuance of the industry. Without this it was impossible to gain full knowledge of their mineral resources. The Carn Brea tin-mining area of Cornwall afforded an instructive example of the potential mineral resources which might be lost to the nation unless means were found to do the requisite exploratory work.

Mr. Robert Annan, after complimenting the author on his paper, said that it was obvious that a great deal of careful and patient research had gone into the preparation of it, and so far from apologising for the clear and concise manner in which he had presented his results, he thought the author was to be congratulated. He knew that quite a number of members present wished to speak on this paper and therefore he would try to be brief, for he could not add very much, if anything, to the technical part of the discussion. He wished, however, to refer to some of the general aspects.

The author reached the conclusion that 'there existed an extensive mineralized area worthy of development'. He started from the familiar premise that copper lodes passed down and became tin-bearing zones in the granite and he quoted from Dr. Maclaren's report on that subject. By the courtesy of the East Pool Company the speaker was able to see that report and in it Dr. Maclaren referred to that same subject in another part and he wished to quote a different paragraph from the one given by the author in the paper because it struck him as expressing Dr. Maclaren's views with great precision. The quotation was as follows :—

> Lodes in the granite will not carry tin ore to indefinite depths ; there is a limit to the downward persistence of the tin-ore shoots. This limit may reasonably be placed at 200 fathoms, not beneath the surface, but beneath the point at which the lode enters or comes into contact with the granite ; beneath this zone of 200 fathoms nothing should be assumed. Weak shoots will not persist so far ; strong shoots may reach that depth ; only the very strongest will pass beyond it.

That favourable zone of granite plunged east and west along the strike of the veins as one went in either direction from the

East Pool shaft and to the north as one proceeded from vein to vein in that direction. He believed with the author that it was a favourable zone and worthy of development. To arrive at an estimate of the gross value the author assumed that the proportionate yield of copper to tin in those veins which penetrated into the zone would be the same as from those which had been worked in the past in both zones. His first comment on that would be that as the value of the tin ore had increased so had the cost of working. One must also assume that as one got to a greater distance from the focus or source of mineralization the strength of the mineralization must be expected to decrease. Whether the lateral extent of tin mineralisation in the granite would be as great as that of copper in the killas he was not so sure; while it might not be true over a larger area, it might very well obtain over the comparatively small area referred to in the paper. Nevertheless, he thought it had to be regarded with a certain degree of caution.

To turn to some of the conditions which would affect the programme of development along the lines advocated by the author, certain points were shown on the plan, notably in the area B. The workings at the bottom of the New Roskear shaft were for the most part in the killas. In area D the rich discovery made in the Tolgus Tunnel from East Pool—now inaccessible—could only be a short distance below the contact and the 1,600-ft. level crosscut in area E also ran out into killas. It cut a lode with promising tin value just below the contact. At all these points it would be necessary to gain depth before any useful development could obtain in those areas.

The next point to which he wished to refer was underground water. As the author rightly pointed out there was very little or no reserve of pumping capacity in the district and development had been hampered by the fear of cutting more water. He had been told on good authority that there was little, if anything, to be gained by changing from the Cornish pumping engine to electrically-driven pumps, but he remained quite unconvinced on that subject. It was true that the charge for electric power in the district was a reproach to any civilized country, but the cost of coal was going up, the quality he was told was indifferent, and the cost of pumping was a very serious burden on the industry. He thought it was clear that the unified system of drainage and ventilation would be of the greatest benefit to the district as a whole, but the capital cost would be a fairly serious matter.

As to the value of the ore which might be developed, the yield in the district for some years before the war averaged about 25 lb.

of 70 per cent tin concentrates per ton of ore milled. Neglecting by-products, that was worth, say, 30s. a ton ore value at a price of £225 per ton for tin. That did not leave a very wide margin of profit. The price of tin to producers today remained a State secret, but he thought that the value of the ore today could not be less than 45s. a ton, which was a very different matter. When Malayan production was lost to the Japanese a shortage of tin extending well into the post-war period became a foregone conclusion and then if ever a long-term policy of development was justified. But during the war the industry came under the influence of the Ministry of Supply and was worked on what was euphemistically termed a short-term policy. The mines were stripped of their skilled labour, the ore was mined in such quantity as was available, and development was restricted without any regard to the future of the industry. He wanted to make it clear, however, that for this no blame attached to the industry itself or to its technical advisers, but it was clear that had a forward development policy been initiated at that time it would have reached those objectives long ago and the cost of carrying out that policy while the mines were in full work would not have been extravagant.

Whether that work could be accomplished now was another matter. It was obvious that the difficulties were much greater now than they were at that time, but he hoped that those who were better qualified to judge the matter than himself would be able to throw some light on the prospects of carrying out that work, which, as he had said, ought to have been carried out long ago and which he hoped it would be possible to carry out in the near future.

Mr. Thomas Pryor said that the paper was lucid, concise, informative, and timely, for there was a great need for the assessment of our domestic non-ferrous mineral resources. In the first place, the area dealt with in the paper was selected because it embraced ground which could be attacked from the workings of existing mines and, in addition, was bounded on the east and west by two modern shafts, which, although now disused, could come later into the scheme if the initial exploration were successful, and thus provide adequate ventilation and good working facilities.

There was general consensus of opinion in agreement with the author that the figure of 1,200 ft. into the granite was a reasonable and conservative estimate of the depth of the favourable zone for tin. When, however, the author took the matter a little further and made a computation that the potential tin ore to be developed might be worth something like £50,000,000 sterling, there were, of course, many assumptions involved which were not capable of precise estimate. One of these assumptions, for example, was that tin would be found below the copper and, again, that tin would be found to persist to similar depth as that of the mines of Groups I and II of the paper (Table II, Plate I).

The lode marked on the map as Reeves's lode, and in its further east extension as 'Old Tom's', according to the paper had produced a great part of the copper output of North Crofty, and was an important producer in Wheal Crofty and North Roskear, thus accounting for a good deal of the copper output of Groups III and IV. Was it not a fact that this same lode, known as 'Old Tom's' lode, had been seen at a number of levels in East Pool mine, but throughout that mine the lode, as far as developed, had been virtually unproductive of tin? If this was correct, it was evidence that, at any rate in the eastern part, tin had not been found in quantity below copper on this line of lode. This would affect the validity of the computation of the possible value of the undeveloped tin ore, arrived at by applying the tin-copper ratio of mines of Groups I and II to the gross copper output of Groups III and IV.

He would like it to be quite clear, however, that he agreed with the author that there were good grounds for supposing that tin would be found below the copper in many lodes of this undeveloped zone, for the most northerly lodes yet worked in the deeper levels of South Crofty were showing good values for tin.

On page 8 the author said that the north-dipping lodes were commonly stronger in character than those dipping to the south and Malcolm Maclaren wrote to similar effect in the *Mining Magazine* of May, 1917. But if the district as a whole were considered, and not merely the area of Plate II, it was the south-dipping lodes which had been of greater economic importance for Dolcoath Main lode and the Great Flat lode both dipped south. Nowadays, in some of the deeper levels, the lodes were getting much steeper, and the distinction between north- and south-dipping lodes was not so clear-cut as it was in former years.

Plate II gave the expected position of the lodes at the 1,900-ft. horizon. Many readers would like more information, but all the information could not be shown on one sheet of paper. If the author had supplemented the paper with other maps similar to Plate II, for the 1,500-ft. and 1,100-ft. horizons, and accompanied those maps with a series of cross-sections, he thought the general argument of the paper would have been made easier to follow.

The Carn Brea area was a very complex district. His personal opinion was that the planning of underground exploration in this district would be greatly assisted by study of a suitable type of mine-model. Since the author wrote his paper a model of suitable type, not involving the use of glass plates, had been under construction at the South Crofty mine. He had seen it a few days previously and in this model were shown the workings of the last 30 years, including the New Roskear shaft workings. Some photographs were on the table and gave an idea of the construction of the model, but no photograph could convey the view of the structural geology afforded by study of the model, which showed up clearly numerous places where further development was most desirable. The model even in its present half-completed stage fully supported Mr. Llewellyn's contention that zones A, B and C of Plate II warranted intensive exploration.

The model did not extend far enough into East Pool to cover the eastern part of Plate II and thus provided no information as regards zones D and E. From numerous visits which he made to East Pool during the years 1942–45, and from a study of the mine plans and records, he had come to the conclusion that zones D and E fully justified the expense of their exploration, but, of course, that expense had been now considerably increased by the recent suspension of work at East Pool. The water had risen in that mine, and the burden of responsibility on South Crofty was therefore greatly increased as the sole remaining active mine in this important district.

The author was certainly correct in stressing the essential need for adequate pumping capacity and good ventilation, but he would not dwell on the author's detailed suggestions for exploration, because before these works could be carried out it was necessary to have come to a decision whether a long-term policy of exploration was to be adopted. Such decisions involved consideration of many important issues outside the scope of the paper. If he might be allowed one digression from the discussion on the paper, it was to say that the recent remarks of the Chancellor in his Budget speech were encouraging, in that he said the Government would find whatever money was necessary to finance useful and practical proposals for developing certain specified areas, although it was to be regretted that Cornish tin-mining areas were not included in the areas specifically mentioned. To give their domestic tin mining a long-term policy was certainly a useful and practical proposal of great advantage to the country. What was needed was the realization that the county did in fact possess important potential reserves of tin ore. The paper was a valuable contribution towards their knowledge of the potential reserves of the Carn Brea area, and he hoped the discussion would show whether or not there was general agreement amongst mining engineers that sufficient potential reserves of ore might reasonably be expected to be found to maintain mining in this district for many years to come. The membership of the Institution included practically all the mining engineers who were qualified to judge the matter. In his opinion, the necessary potential reserves of ore existed, but the work required active exploration of many lodes containing scattered orebodies and for that reason he had stressed the use of suitable models to guide the exploratory work and study the structural geology upon which the dispostion of the orebodies largely depended.

Mr. H. G. Dines* offered his congratulations to the author on his paper and particularly on bringing the West of England up for discussion. He had started by saying that it was not his intention to give a geological paper and yet it seemed that his deductions were based primarily on geological considerations. One criticism he had to make was that the author still adopted a rather rough-and-ready rule-of-thumb idea that the top of the tin zone and the top of the granite were for all practical purposes coincident. The depth zones were believed to have been controlled by the temperature gradient of the country rocks at the time of ore deposition. The granite, no doubt, supplied heat and at the time of the injection of the granite bosses it was possible that the isotherms of the country rocks were roughly parallel with the granite surface, but there was evidence to show that the arrival of the granite masses and of the mineral-bearing solutions were separated by a time interval, during which cooling must have set in, and the isotherms must have settled down, so that, while they sloped in the same direction as the granite surface, they did not slope so steeply ; the depth zones were, in fact, so disposed in many places in the West of England.

It so happened that in the major lode system of Dolcoath, South Crofty and East Pool the top of the tin zone and the surface of the granite were more or less coincident, but northwards the tin zone rose above the granite. This was brought out by the account of the North Crofty mine given in the Appendix, which showed that tin was being worked where the granite surface was at nearly twice the depth. This meant that the potential tin was not so deep as the author had allowed for, which from the point of view of tin mining was, of course, an advantage ; but there

*H.M. Geological Survey.

was, unfortunately, another geological consideration which was not quite so encouraging. The general distribution of the mineral deposits in the West of England led to the conclusion that the mineralizing solutions rose up in certain restricted centres and the oreshoots of each mineral zone, in upward succession, spread to a wider lateral extent around these emanative centres.

The deposits of tin, the deepest zone, were much more confined laterally within their zone than deposits of copper in the zone above. Tin was, therefore, only to be expected below copper deposits actually at the emanative centres. The great lode system of Dolcoath, South Crofty, and East Pool represented the largest emanative centre in the West of England ; nevertheless it had limits—an oval area embracing the territory of the three large mines—and as yet there was no proof that in the area to the north there were rich deposits comparable with those of the emanative centre. As evidence, they had the results of the new Roskear and Tolgus shafts. It could not be denied that those shafts did pass through the copper zone and to about 1,000 ft. below, and it was not to be assumed that that 1,000 ft. was in a barren zone between the tin and copper, for tin was definitely being worked in North Crofty mine at a depth of 1,000 ft. from the surface or less.

Another significant thing in the workings of those shafts was that the only values which attracted attention were those which were found in the cross-cuts southwards—that is to say, towards the emanative centre. It was true that in the case of Roskear the values were actually in granite, but his contention was that the presence of granite was not *ipso facto* significant of the presence of tin ; the granite was several hundred feet below the tin worked at North Crofty. Incidentally, it was learned from the Appendix that, at North Crofty, tin below the copper was worked in the early 1860's, but it had never developed into a tin mine comparable with those of the emanative centre.

Another point made by the author was that the mines in the north were nearly all abandoned when they reached the greenstone. That might have been so in some cases, but not in all. South Roskear was sunk 70 fathoms below the lower greenstone sill ; the workings of North Roskear mine must have been sunk 80 to 100 fathoms, and North Crofty to 120 fathoms below. The plans of these mines showed that the development below the greenstone was quite extensive and the amount of stoping was quite small and generally confined to scattered patches.

He did not wish to give the impression that he considered this paper as being without value—far from it. His criticism was that with all the meticulous work put into Plate II, the author did not seem to have taken account of all the available geological evidence.

During the war the Geological Survey had put a great effort into the investigation of the West of England from the standpoint of practical mining geology and not only had all accessible workings been systematically surveyed, but old mine plans, account books, and other documentary evidence in private possession had been carefully examined and analysed and the information contained sifted out. He wished to express appreciation for the way in which the people of the West Country had co-operated. Only in a few instances had permission to examine such private documents been withheld. In addition to these they had also inspected all the plans in the Mining Records Office. They had been examined in the same way as the private plans and had all been reduced to a standard scale. Plans of over 900 mines were so dealt with, which involved the reduction and examination of close on 4,000 plan sheets. As a result of all this work they now hoped that they had the whole of the obtainable evidence on the mining work in the West of England. He hardly needed to say that this work was due mainly to Mr. T. Eastwood, who, as Assistant Director, had started on the work in the West of England early in 1938.

He agreed with the present author that the northern part of the area was not as well prospected as one would wish. It was a great pity that a boring programme was not carried out from the workings at East Pool before that mine was allowed to close down.

In conclusion he said that in his opinion areas A and B as marked on Plate II were very favourably situated. They were opposite or close to the rich part of the emanative centre, where the tin zone was deepest and practically undeveloped. It was a surprising fact that below a depth of 1,700 ft. on the west side of the great cross-course—that is, area B—there had been no drive of any kind between the south cross-cut from Roskear shaft and the main lode of Dolcoath, which had been worked to a depth of 3,300 ft. Area C was worth prospecting; area D, he thought, was not quite so favourable except, perhaps, westward, and area E was too far north for any sizable runs of workable ore to be expected.

Mr. C. V. Paull said that, as manager of South Crofty, the only mine still working in the area under discussion, he wished to express himself largely in agreement with the author's conclusions. He doubted whether there was a similar area in Cornwall which contained so many lodes, but, unfortunately, the majority lacked continuity both on strike and dip. He could give many instances where, in cross-cutting at different horizons, they had been disappointed to find nothing worthy of the name of lode either in width or value ; on the other hand, they had intersected highly-payable lodes where least expected. This had been his experience during 20 years at South Crofty and it was the experience of his father, Mr. Josiah Paull, during his long period of management.

To quote a classic example, when East Pool was developing the Rogers lode at the 225-fathoms horizon, the west drive going towards South Crofty (and only 600 to 800 ft. distant), the lode was some 20 ft. wide, very strong, and rich in value, and it appeared that it could live on this strike for ever, yet their 225- and 180-fathom cross-cuts north exposed no north-dipping formations, but only two rather poor south-underlying lodes, which had never been stoped, and both these cross-cuts were extended north to the killas junction. From the blanks drawn in these two pioneer cross-cuts it could easily have been assumed that the mine was doomed in depth, but seven or eight productive lodes had been and were still being stoped below the 225-fathom level.

Given sufficient development the South Crofty mine had invariably responded by opening up ample ore reserves to supply the 60-head stamp mill, or a monthly tonnage of approximately 6,000. Unfortunately, since the inception of the present company there had been two major wars, lasting over ten years, and in company with other mines they had suffered from shortage of labour, curtailed development, and depleted ore reserves to meet urgent national requirements. The need of labour was such that the mill was running at two-thirds capacity. A further handicap had been the closing down of neighbouring mines—Dolcoath, Tincroft, Old East Pool, Carn Brea, and, more recently, new East Pool and Agar—and the added burden of pumping their water had been thrown upon his company. All the mines mentioned were in the area covered by the paper. Unfortunately, some of them were holed to South Crofty over widths and lengths impossible to dam and it had been necessary to handle their water in order that South Crofty itself might live. In spite of the extra burden of high pumping charges and a sad depletion of immediate ore reserves, they were still confident that South Crofty could be brought back to a profitable mine again if vigorously developed and provided with electrically-driven pumps and modern mechanical muckers and locomotives.

The New Dolcoath mine immediately joined their western boundary and was now part of South Crofty. It was acquired in 1935, with the idea of extending their drives westwards into

In conclusion he emphasized again that the area so carefully mapped and studied by Mr. Llewellyn was only a small part of the Cornish and Devonshire tin areas, some parts of which held out equal promise to the belt under consideration. He sincerely hoped that these other areas would also be studied with equal interest, in the expectation that one day in the future their revival might be undertaken.

Mr. John H. Trounson said that the author was to be congratulated on the production of a paper that had the merit of taking a bold and comprehensive view of the possibilities still latent in the Carn Brea area. In the past exploration had frequently been conducted on far too limited a scale; economic considerations had, of course, been partly responsible for this state of affairs, but it was undoubtedly true that the narrow and unimaginative outlook of past mine managers had also been much to blame.

Much as one agreed with the author's willingness to take broad views, one could not help but doubt the accuracy of some of the implied statements contained in Table I. If each individual vein had some clearly distinguishable characteristic it would be possible to identify it in widely-separated mines, even though large tracts of undeveloped ground intervened. As matters stood, however, until much more exploration had been carried out it seemed very doubtful whether the common identity of certain lodes in different mines could be established in the manner suggested in Table I. As the author pointed out, however, many of the lodes were probably far more continuous both in strike and in depth than was generally realized. In some quarters there had been a tendency in recent years to regard the deep-seated lodes remote from the centre of the granite outcrop as short 'gash' veins of limited economic importance. Recent developments in South Crofty, however, were throwing doubt on the correctness of this assumption. A good example of a lode hitherto regarded as a relatively short orebody was the No. 4 lode, which occurred near the northern limits of the mine as so far developed—namely, 1,000 ft. north of Robinson's shaft. This lode had now been extensively explored at four horizons in the Robinson's section for a length of over 1,200 ft. and in recent weeks what appeared to be the same lode had been intersected in the 315-fathom cross-cut (1,200 ft. north of Cook's shaft) at a point 830 ft. west of the most westerly drive on it in the Robinson's section. If this proved to be a continuous vein its strike would have been demonstrated to persist for over 2,000 ft., while at the present time it gave great promise of much more extension still further westward. The No. 4 lode was not the only instance that could be given of this type of development; there now appeared to be a distinct possibility that the No. 1 lodes of Robinson's and Cook's sections respectively might prove to be the same vein. Indeed, as the author pointed out, there was great scope for far more lateral exploration, even along the strike of the known lodes.

The paper suggested five zones or areas worthy of intensive development and of these A and B seemed to offer the best possibilities for immediate attention. In consequence of war-time difficulties the only major piece of exploration which it had been possible to carry out at South Crofty in recent years had been the driving of the 315-fathom north cross-cut from Cook's shaft. As previously mentioned, this had resulted in the cutting of the No. 4 lode, which was obviously a major orebody, the 200 ft. of driving to date having exposed an average value of 31 lb. of black tin per ton over a width of 5·5 ft., the greatest width so far seen being 9 ft. In addition to cross-cutting to that lode at higher and lower levels it was clearly desirable that exploration be extended still further north into the author's A zone, where other orebodies of importance might reasonably be expected to exist.

One could not help feeling that in his paper Major Llewellyn had not adequately emphasized the immense possibilities that were latent in zone B. In the Carn Brea area the lodes immediately north of the granite outcrop might be divided into two groups:—

(1) The predominantly south-dipping lodes of the Dolcoath-Carn Brea series of mines.

(2) The 'Entral' lodes, mostly north-dipping, of the East Pool-South Crofty group.

All lodes in the area were intersected by the Great Cross-course under the Tuckingmill valley and, irrespective of dip, seemed to be faulted approximately 400 ft. in a right-hand direction. Whereas the lodes of the first group had been most extensively worked on both sides of the cross-course, the 'Entral' lodes further north had for some obscure reason never been developed in depth west of the fault, although their outcrops in the northern part of the old Dolcoath mine had long been recognized and indeed worked to shallow depths by the 'old men'. The development of these lodes (in zone B) could, with adequate labour, be immediately tackled from the western or Cook's section of South Crofty and later from the deep circular Roskear shaft, which was sunk further westward by the Dolcoath company some years ago. *The importance of* extensive developments in this zone of the Carn Brea area could not be over-estimated, as they offered first-class opportunities for major discoveries that might well produce a great revival of mining within the area.

With reference to the other zones mentioned by the author, that referred to as C certainly seemed to be worthy of further investigation, although it was of secondary importance in comparison with A and B. Experience in the East Pool mine did not seem to offer very hopeful prospects in zone E and he would not personally recommend any further development there at the present time. Zone D, however, was in an entirely different category and it was regrettable that through a combination of circumstances it had been found impossible to resume the development of the magnificent orebody intersected in this area in 1920 at a depth of 1,550 ft. from surface. In view of the position of the killas-granite junction to the north-east the length available on the line of strike of this great lode might be limited, but there would seem to be excellent possibilities, too, in the development of the parallel lodes, some of which were intersected by diamond drilling from the Tolgus Tunnel, in which the great lode was cut. If the East Pool and Agar company were ultimately forced to go into liquidation without being able to tackle this area, it would, from the national point of view, be a most regrettable loss of potential mineral production and employment-giving capacity at a time when both were likely to be sorely needed.

The paper omitted all reference to an interesting and important development that had been taking place in the South Crofty mine during the past year—namely, the discovery of wolfram in considerable quantities, together with coarsely-crystalline cassiterite and great quantities of mispickel in a lode at the very bottom of the mine. The orebody in question had, by reason of its contents, been named the 'Complex' lode and had now been developed at the 335-fathom level of the Robinson's section for a distance of 240 ft., the last 175 ft. of which averaged 69 lb. of wolfram and 15 lb. of black tin per ton over a lode width of 3·3 ft.; in addition very high arsenical values were present. The 335-fathom level was 2,021 ft. from surface, or approximately 1,000 ft. below the killas-granite junction, and, indeed, was at a geological depth where it was thought that the lower limits of the tin zone were being approached. Similar minerals were admittedly encountered in a lode found in cross-cutting north at the 2,000-ft. level of the new Roskear shaft, but, whereas in that case the lode was in the *sedimentary* 'killas' rock, the 'Complex' lode at South Crofty was deep in the granite and as such absolutely unique in the history of Cornish ore occurrence.

The possibilities of major wolfram discoveries in depth were not limited to the ' Complex ' lode alone, as there were numerous other indications of the presence of wolfram in the deeper levels of the mine and especially so in the case of the No. 8 lode of Cook's section, which was more than 1,500 ft. west of the present developments on the ' Complex ' lode.

The discovery of this remarkable lode, which looked likely to ' make ' downwards rather than upwards, added additional interest to the old question whether the tin zone hitherto worked was the lowest horizon at which economic mineralization occurred or whether there were further workable deposits of tin or other metallic minerals at still greater depth. The presence of coarsely-crystalline cassiterite as well as massive mispickel (both typical of the shallow zones of the lodes) added still further interest to the discovery of the ' Complex ' lode.

Whatever theories might be advanced in explanation of this unique formation, it was certain that its economic importance could only be proved by further development in depth ; indeed, such development might compel them to modify several of their present ideas concerning deep ore occurrence in the area. Unfortunately, the present acute, although probably only temporary, shortage of labour had necessitated a suspension of work on the lode. In view, however, of the extremely important information that deeper sinking on that lode might yield with reference to ore occurrence at greater depths throughout the area as a whole, it would seem to be a matter of the greatest interest and value that the sinking of a winze should be commenced on the ' Complex ' lode at the earliest possible moment.

One other point in the paper necessitating comment was the mode of estimating the value of the reserves of tin ore in the area. This seemed undesirably speculative and it would probably be wiser to defer any such estimates until extensive developments had been carried out on the lines indicated in the paper. Anyone conversant with the Carn Brea area was aware that there were still important possibilities latent there, but, inasmuch as there was no substitute for actual underground development, until this had been much extended it would seem wise to defer even general estimates of the value of the metal present.

Professor W. R. Jones said that by far the greatest production of tin had hitherto been from alluvial and eluvial deposits. These surface deposits, when worked to bedrock, ceased to yield cassiterite and there was no second crop. When it was considered that in Malaya one of the many large tin dredgers could in one day recover more tinstone than would accumulate throughout the whole of the Malay Peninsula by natural weathering agencies in a generation, it required little imagination to realise what the result would be in the course of years. Very extensive alluvial areas in Malaya, Siam, Dutch East Indies, and Nigeria had already been worked out, many were approaching exhaustion, and some had passed their optimum production, while others had a life of several years. Inevitably, the history of tin mining would repeat that of gold mining. Within living memory the bulk of the world's gold was from placer deposits ; now it was from reefs and lodes.

It was true that in a few parts of Malaya workable tin lodes occurred, but in that and the neighbouring countries, and in Nigeria, most of the cassiterite in the alluvial and eluvial deposits was derived from stringers and disseminations which could not be worked economically in undecomposed rock.

The writing on the wall was clear enough to those interested in tin mineral resources ; it was that the time would come—and the rising generation would see it—when greater attention would be focused on the few parts of the earth's crust where tin lodes occurred. One of these few was Cornwall. It was for these reasons that the speaker welcomed the paper, which contained much new and valuable information concerning an area where, for its size, more tin lodes had been mined than in any part of the world.

Mr. T. Eastwood desired to say that he believed there was still some good stuff to come out of Cornwall. He thought, however, that in order to get that material out it was necessary to take a very long view indeed. That long view would demand a synthesis of all available information and it had been his policy on the Geological Survey to synthesise all such information so that people might have at their command the knowledge which they would undoubtedly require in order to develop any area.

He was extremely glad to welcome a paper by the author on this area, because his methods were admittedly different from theirs, although on the same general lines. He could not take so rosy a view as Mr. Llewellyn did of the precise amounts of material to be gained, but he thought that if progress were made on the lines of really opening up fairly large blocks of country to a preview, there was some chance of success. He was glad that the paper had been brought forward because he believed that the more views on these lines the better—in other words, two heads were better than one. Although they of the Survey might hold different views from Mr. Llewellyn, there was probably in some cases just as much chance that his views were as right as theirs.

Sir Lewis Fermor said that he had not much knowledge personally of the tin industry of Cornwall, although he had been there once or twice. But he had been to many countries abroad, and everywhere he went he found Cornish miners or Cornish mining engineers and it became evident to all those who travelled that the continuance of British interest in mining industries abroad depended on the continuance of the supply of men trained in Cornwall. Therefore, they must be interested in the problem whether or not Cornish mining industries were going to persist. This depended upon two things—namely, upon whether the mineral was there and, secondly, upon whether the taxation imposed by Government would permit it to be extracted.

He had found it heartening to listen to the paper because it seemed evident therefrom that there was still plenty of tin ore in Cornwall to be got out. What they had to do now was to secure an economically feasible scheme of taxation, so as to permit the extraction of the ore that they all believed was there. For that reason he was glad that the author had read his paper and he congratulated him thereon.

Mr. W. C. C. Rose said that in congratulating the author he would like to say a word on behalf of the Ministry of Supply. It was partly owing to the work of that Ministry during the war that Major. Llewellyn had been able to prepare his paper, with which remark, he was sure, the author would willingly agree.

Mr. A. T. Holman said that he was delighted that someone had come forward and put the case for Cornwall in such a clear way. His uncle, who had been closely associated with Cornish mining for many years, had held strongly that there was a great quantity of tin in Cornwall not yet got out. His own point of view, while in full agreement with this, was more particularly that of the mechanical engineer, and he felt it was vitally necessary to keep Cornish mines going. There would not have been the mining machinery exported from Cornwall or perhaps from England as a whole but for the Cornish mines and the mechanical devices which were developed there. There were four firms in Cornwall exporting very large quantities of material to all parts of the world. In the first place it was due to the fact that Cornish men went out as experienced miners and mining engineers and had asked for the material which they knew. The Cornish mines were an asset which England could not afford to neglect and should be exploited for the sake of the whole country.

Mr. J. O. Allan, after complimenting the author on a very constructive paper, said that intelligent appreciation of the problems relating to Cornish mining was sufficiently unusual to make a well-reasoned thesis on the subject, such as the author had presented, very refreshing reading when compared with so much that had been written. The author wrote that many instances had been recorded of tin above copper. Where tin occurred above copper it would probably have been accessible to the ancients and in his opinion much of the tin reported to have been mined by them must have come from outcrop workings. His reason for this view was that the surface configuration of Cornwall did not lend itself to widespread alluvials. It was possible, therefore, that the production of tin in ancient times was not so important as history would have them believe. Tin alluvials were, however, much larger and more wide-spread on the Iberian peninsula and traces of extensive working existed there. The Cassiterides legend might, therefore, have been an early form of propaganda to discourage possible tin buyers. The mining of tin over the last 80 years, however, together with the Cassiterides story, had led to the over-emphasis of tin in Cornish mining history. For over 200 years the principal metal mined in Cornwall was copper. While tin had been mined in a desultory fashion since very early times, it was the copper mines that gave the real foundation to the tradition of Cornish mining to which the modern industry owed so much. This was clearly brought out in Table II, which gave results of only the small section of the West Country mining area under consideration.

In considering the history of the old mines it must be borne in mind that the price of tin averaged below £100 per ton from about 1815 to 1855, during which period the price of copper fell gradually as low as £60 per ton in 1848. The Dolcoath mine, which in 1845 produced 3,400 tons of copper ore, had dropped to 700 tons in 1855, prior to which year the tin production had been under 500 tons per year. During the years 1850 to 1855 the price of tin doubled and the Dolcoath tin production showed a steady annual increase for 30 years. Thus in the case of the mine that produced 36 per cent of the tin in the area under review it would appear that it was the rise in the tin price at a critical juncture in the history of the mine that enabled it to keep the mine unwatered and carried through as a tin producer. Had the copper been exhausted 30 years earlier it was probable that Cornwall's biggest producer of tin to date would today be in the position of many other of the abandoned copper mines. It was only to be expected that, as was pointed out by the author, the mines nearer the granite outcrop would have less difficulty in reaching the tin zone, but the painful history of attempts to open many of the old mines outside this area had been largely based on stories of old miners and not on such constructive consideration of the broad factors involved as had been given in the paper. The problem was not easy, the difficulties being such that they could only be overcome in a large way with ample capital backing, and it was useless to imagine that Cornish mining could be revived by companies with a capital structure more suitable for a medium-sized grocery business.

Another point was that the author gave the estimated tin production at 275,000 tons of concentrates. He considered that the over-all extraction was not likely to have been more than 60 per cent, so that this represented a total of some 460,000 tons. He could not imagine any organization with the courage and resources necessary to solve the problem being content with a mill treatment that allowed individuals to make a comfortable living out of the mill tailings, nor would they be content to hand their concentration problems to old gentlemen whose idea of a modern concentration machine was beating a tattoo on the sides of half a beer barrel!

In conclusion he again emphasized that in the history of mining in Cornwall copper had been far more important than tin and although the mining of the latter was able under special circumstances to struggle through in a small way, the 30 years of low metal prices in the middle of the last century gave Cornish mining a blow from which it had never recovered. The general world increase in mining after 1860, coming at the end of a long period of depression, dispersed the mining population and had added to the difficulties.

The President said that he was glad that Mr. Annan had mentioned the matter of pumping. He knew that they might be getting on rather dangerous ground if they tried to make a comparison between Cornish pumps and electric pumps, but he was not at all convinced that the Cornish engine was the best form of equipment for that particular purpose. He had no doubt that this question of Cornish pumps *versus* electric pumps had been thrashed out over and over again in Cornwall and it would have been interesting to hear what the final opinions were. He was sorry that the author had not dealt with this pumping question rather more fully; it was only referred to in one place in the paper when it was stated that the pumping capacity should be increased, a sentiment which he was sure all of them would endorse.

He believed he was right in saying that most of the water with which these mines had to deal was surface water and that the amount of water which the mines made in the deeper workings was comparatively small. The author did not say anything about the total quantity of water to be pumped, but from figures which he, the speaker, had himself seen he did not believe that this quantity was really very excessive when compared with the large quantities of water which were pumped from mines in other parts of the world. It would certainly appear that water had been the bogey of Cornish mining, particularly in this area, and that the mines had always had to work under a severe handicap—namely, the constant fear of being drowned out. That was particularly unfortunate, because the richest lodes were often likely to be very wet ones.

Another point only briefly touched upon in the paper was diamond drilling. The author said that drilling alone would not provide sufficient information. In the case of these Cornish tin lodes, where values were liable to be very erratic, this was undoubtedly true and he would like to support that statement strongly. The best way of finding out what was really there was by means of underground workings—particularly cross-cuts and drifts.

Mr. Brian Llewellyn, after stating that he would prefer to make his reply in writing after he had studied the report of the discussion, said that the President's assumption about water in the mines was correct—it was surface water. When more than half the adits were unplotted on any map it was extremely difficult to ensure that all surface water was led away and did not go into the mine.

Mr. C. V. Paull desired to add a word arising from two queries raised during the discussion: In his own particular mine, where they were pumping about 800 to 900 gallons per minute, only about 300 gallons of the total came from the lodes themselves. The remaining 600 gallons came from adjoining abandoned mines; the whole area was supplied with adits, but with so many mines closed the surrounding adits were not patrolled and repaired as they should have been and were now in many cases inaccessible. The adit water flowed direct into the abandoned mines and the re forced the water through the cross-courses into theirs. case of two mines it would overflow and gain direct access continually pumped.

his own mine the water was pumped from 2,200 ft. by means of Cornish pumps, but the speed of those engines was limited by the weight. They had reached the capacity of the two pumps, which were 90 in. and 80 in. respectively. Their total capacity was not great when talking in terms of modern electric pumps. It required 15 men to man each pump—three drivers, three stokers, four men climbing through the shaft, and also the repair gang, whereas to pump the whole mine electrically six or eight men would be sufficient. They had gone into the figures carefully and they themselves were putting forward a proposition that they should transfer to electric pumping; it broke their hearts to think of saying good-bye to Cornish pumps, but electric pumping would be cheaper, owing to the big increase in the cost of labour and coal.

On the proposal of **the President** a hearty vote of thanks to the author for his paper was passed with acclamation.

CONTRIBUTED REMARKS

Mr. Maurice Gregory: Mr. Llewellyn is to be complimented on the consolidation of a large amount of scattered information dealing with the area in question and on the comprehensive plan of the lodes which accompanies it. It is to be hoped that he may find the leisure and inclination to deal with other sections of the county in a like manner.

Much limelight has been brought to bear on the mines of this area around and particularly north of Carn Brea, principally on account of their having been such prolific tin producers in the past: also, their long life has been in a measure due to the persistence of economically-savable tin oxide in the lodes down to great depths. No major lode channel has been reported as having been bottomed; it may have pinched, or, what is more important, the tin oxide may have become so fine as to be difficult to save economically in the dressing plant. I feel I must stress observations in practice where it has been found that the deeper one penetrates the granite the finer and more difficult to save does the tin become and estimates of potential tin ore values should be considered with this in mind when dealing with values below horizons where comparatively recent mining operations ceased in the granite.

Fluctuating prices for tin and the working of the mines on complex- and low-grade ore down to a point where economic reserves were depleted, due to a close working balance between ore values and dividends, were probably prime factors which prevented explorations farther afield and in depth, as the author

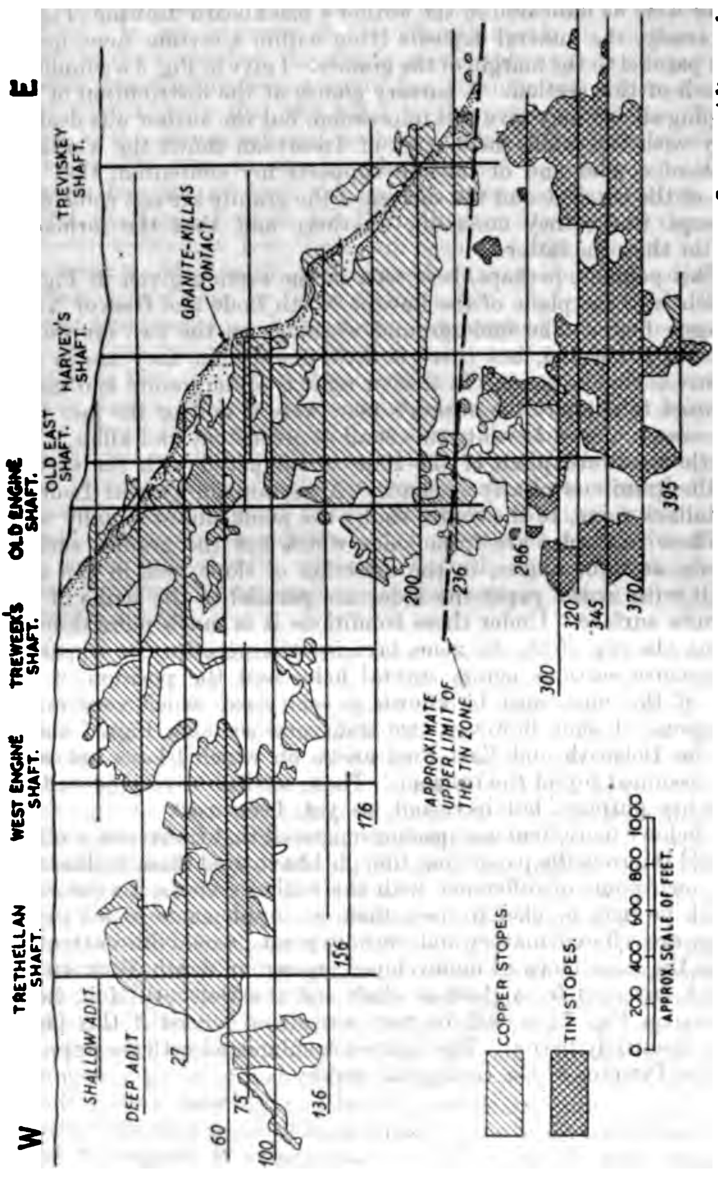


FIG. 5.—Section of the workings of Tresavean mine, showing the relation between the slopes of the granite surface and of the junction between the tin and copper zones.

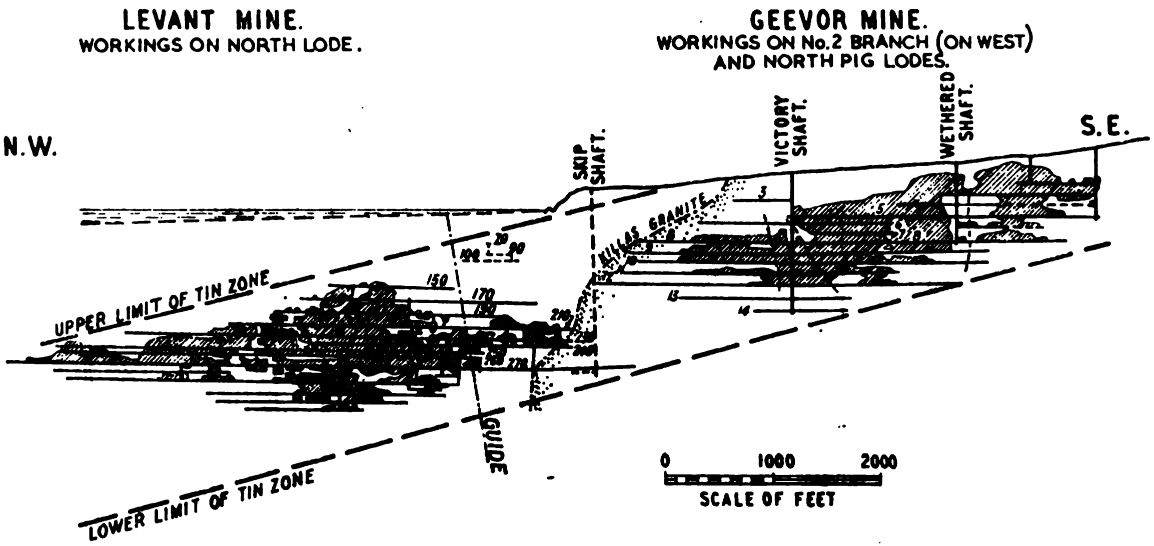


FIG 6.—Section in the plane of North lode, Levant mine, and No. 2 branch and North Pig lodes of Geevor mine, showing the position of the tin zone relative to the granite.

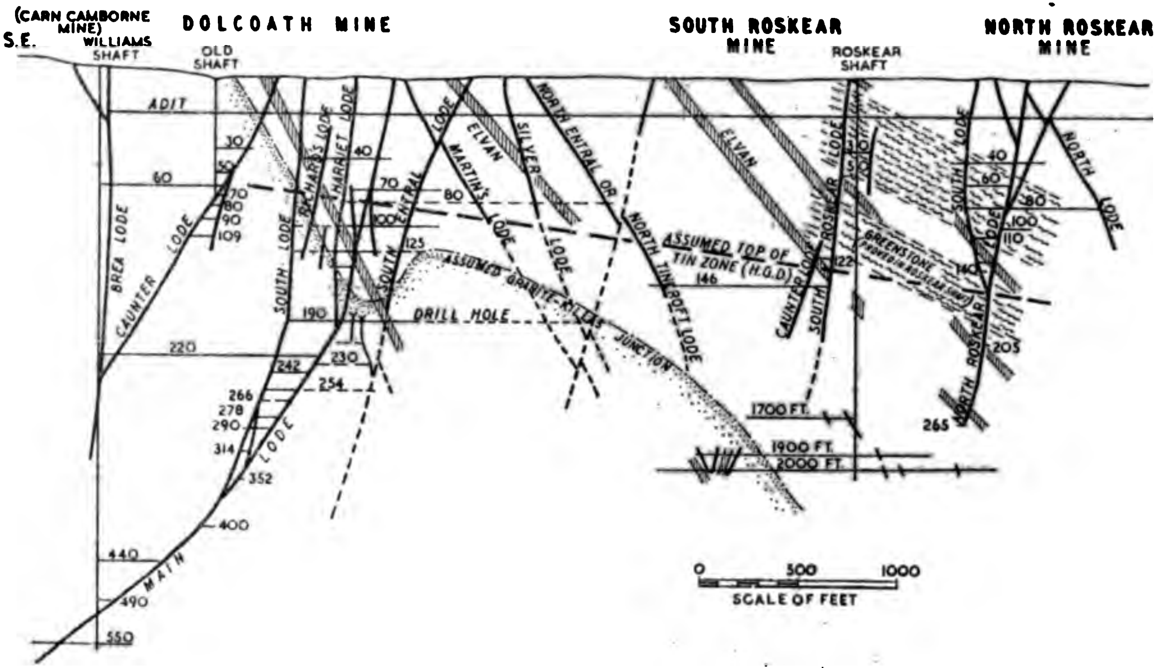
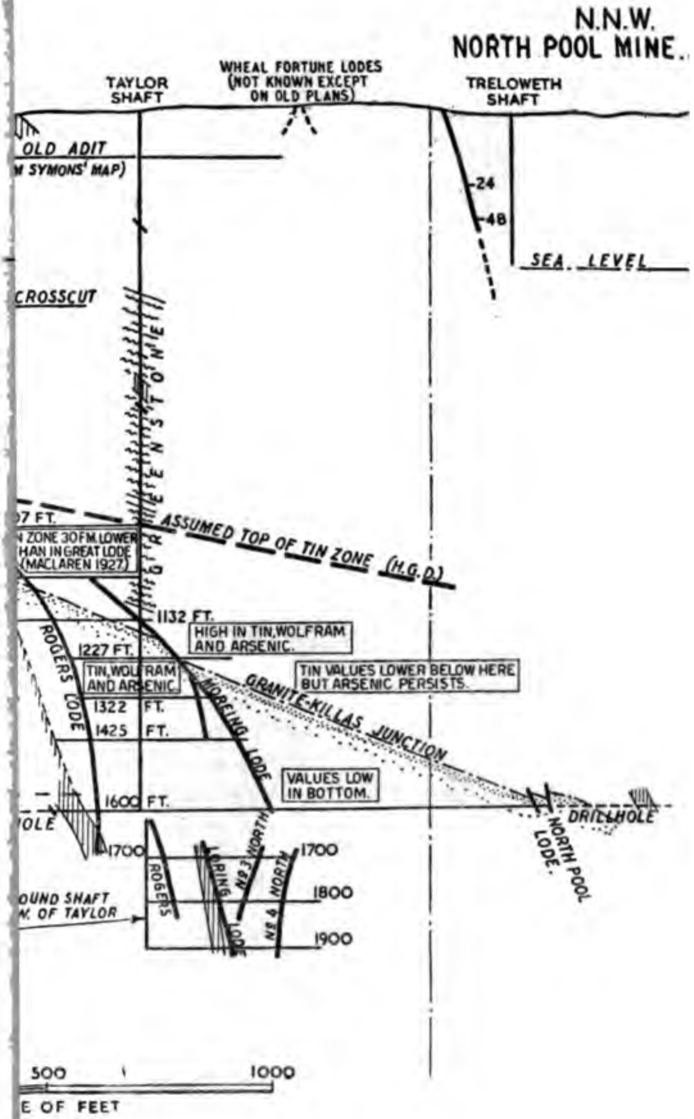


FIG. 7.—Diagrammatic section across the strike of the lodes between Dolcoath and Roskear, based on a section by H. V. Thomas, issued by Dolcoath Mine, Ltd., and on information published by E. H. Davison (1929) on the new Roskear shaft. (Figures represent depths below adit in fathoms, except those at Roskear shaft, which are in feet from shaft collar.)



on of the workings from Taylor shaft projected on to the line of
y by Maclaren in 1918. (Figures represent the depths in fathoms

AUTHOR'S REPLY TO DISCUSSION*

ON

Shaft-Pillars and Shaft-Spaces.

By G. Hildick-Smith, *Member.*

Mr. G. Hildick-Smith: I wish to thank all those who contributed to the discussion on my paper and especially Dr. Keep for introducing it and giving explanations on certain matters raised in the discussion. Dr. Keep's thought-reading regarding the reason for late pillar extraction in rocks of lower tensile strength was entirely correct.

I think perhaps, as usual, some details of actual results may be of more interest to members than quibbles on theoretical grounds, and so I will give an overall reply to the discussion in that way.

Three stages in regard to the work on the shaft-pillar extraction at the Central Shaft, Modder B. Gold Mines, are shown in Figs. 8, 9, and 10. The time factors for the various periods should be noted. It will be seen from Fig. 10 that the conditions as shown have existed for nearly two years. Details of the hanging-wall subsidence, as determined at points near the shaft, are shown in Fig. 11 and those for the surface in Fig. 10. The relative amounts of subsidence shown in Figs. 10 and 11 should be compared.

It is interesting to note that the subsidence of the hanging-wall over the point X, Fig. 10—shown also in the section at X, Fig. 12—has now been sufficient, due to the compression of mat-packs (chocks 4 ft. square, built of 5-in. by 4-in. slabs placed four in each row and 5 ft. apart on strike and dip), to punch the foot-wall and bend steel girders over the main ore bin. Movement above the bin stopped immediately the compressed mat-packs were extracted and replaced by new ones. This would not have happened so soon with a foot-wall rock of higher tensile strength than the shale of which it was composed.

With the total amount of subsidence to date as shown in Fig. 11 no difficulty has been experienced in the shaft itself, as the telescopic arrangements for studdles and shaft runners were effective, provided care was taken to ensure that the bolts could move freely in slots in the timbers and plates.

*Bull. 470, January, 1945.

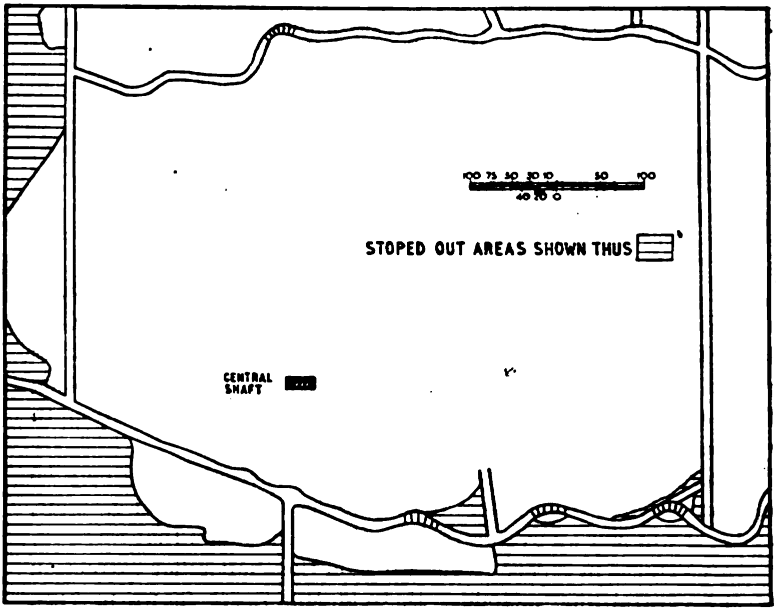


FIG. 8.—Plan as at 31st December, 1940.

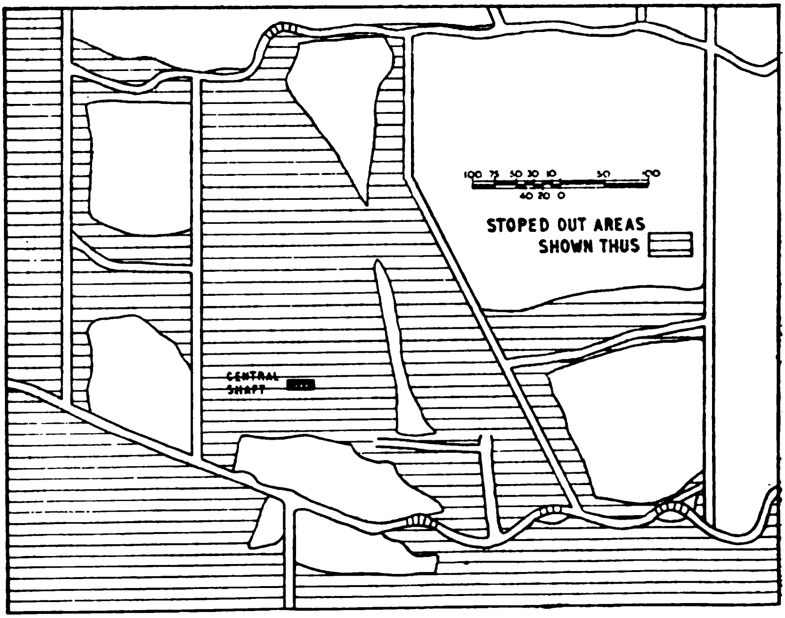


FIG. 9.—Plan as at 31st December, 1943.

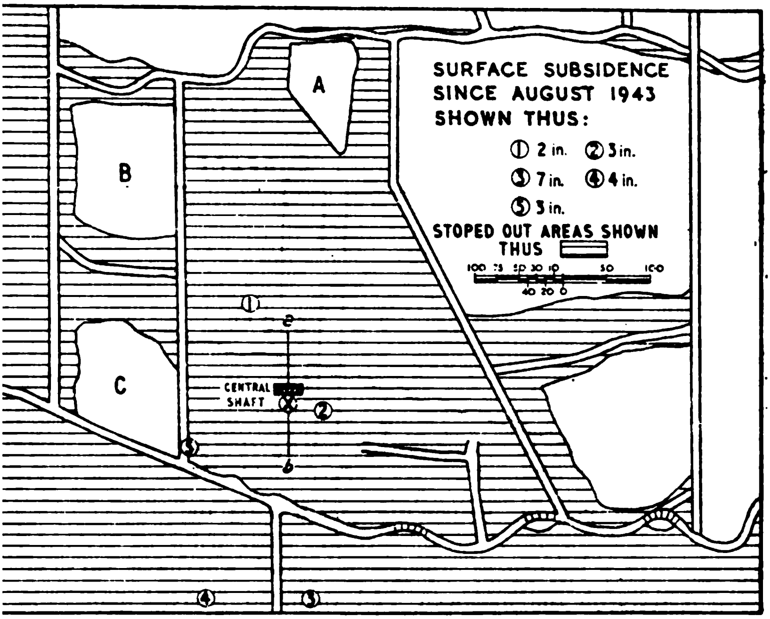


Fig. 10.—Plan as at 30th June, 1944, to date (25th April, 1946).

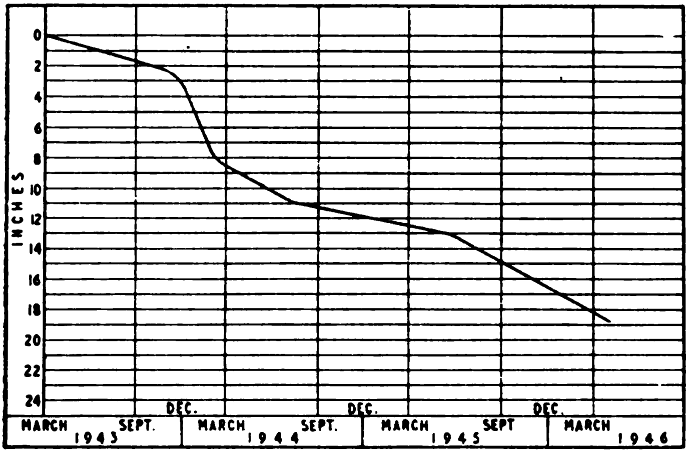


Fig. 11.—Hanging-wall subsidence.

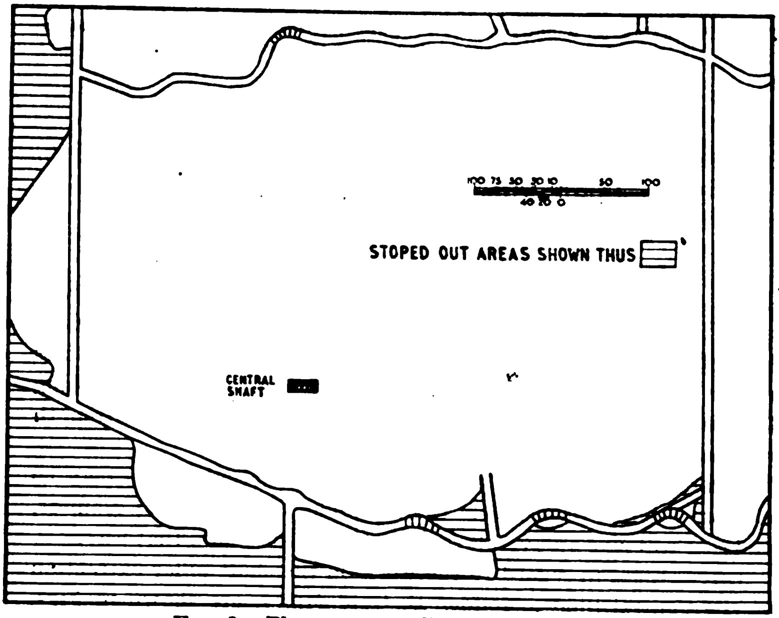


FIG. 8.—Plan as at 31st December, 1940.

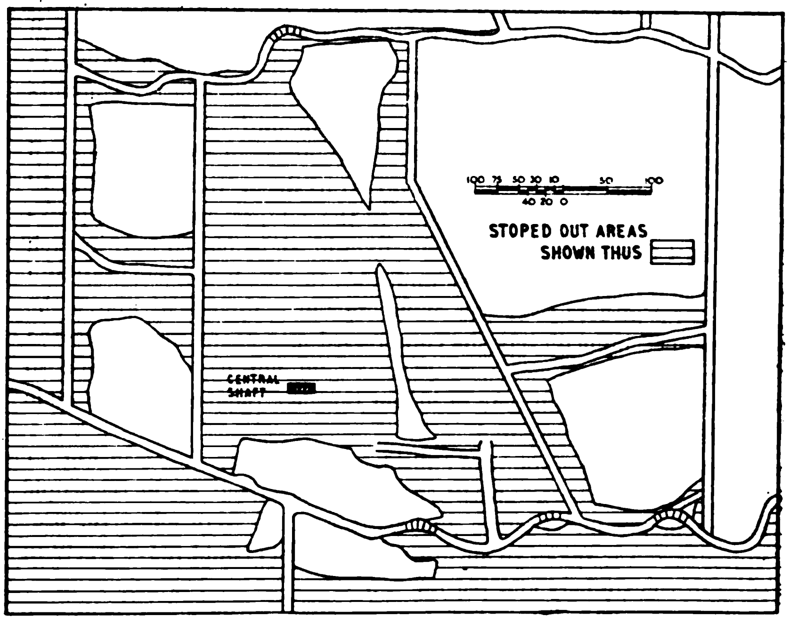


FIG. 9.—Plan as at 31st December, 1943.

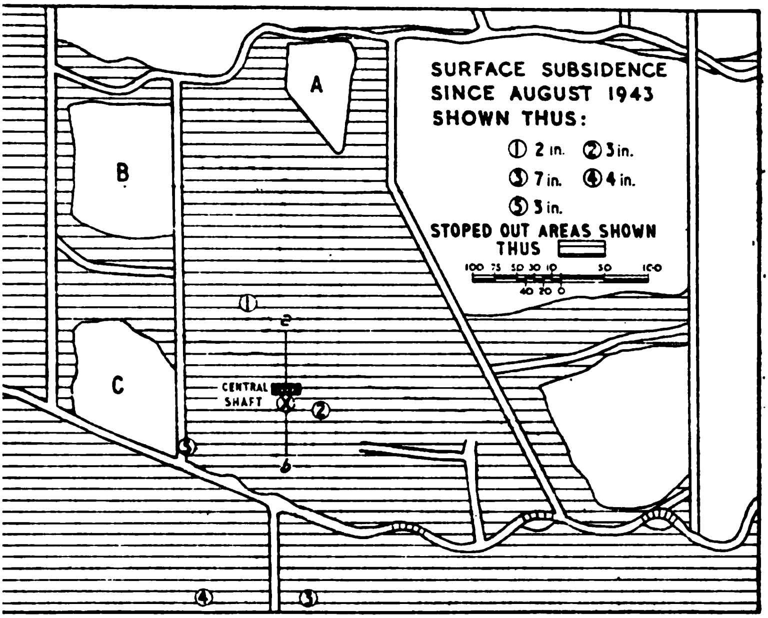


FIG. 10.—Plan as at 30th June, 1944, to date (25th April, 1946).

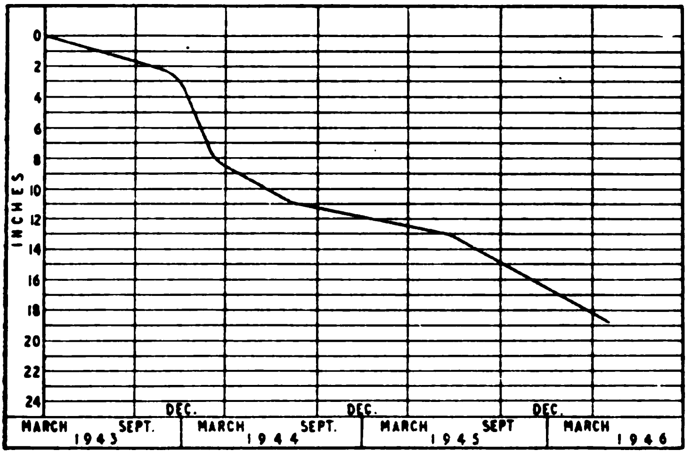


FIG. 11.—Hanging-wall subsidence.

It will be seen in Fig. 10 that the pillars A, B, and C have been left for the time being in order to damp down the hanging-wall subsidence. When either shearing of the hanging-wall in their vicinity or the crushing of the pillars takes place, then they will

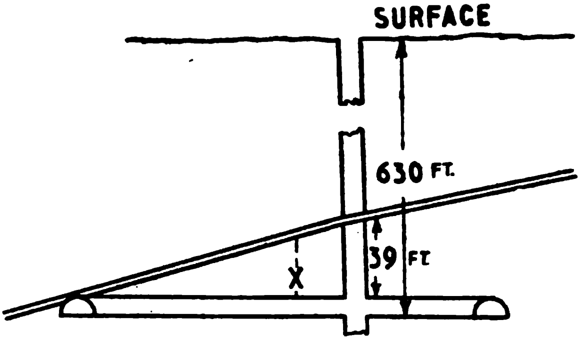


FIG. 12.—Section through a-b of Fig. 10.

be removed. Eventually the hanging-wall will have subsided until it rests on the foot-wall, and then, to prevent punching, the hanging-wall over the necessary area for about 4 ft. thick will have to be mined, or, in other words, similar operations to those necessary in the original stoping will have to be repeated.

AUTHORS' REPLY TO DISCUSSION*

ON

The Mineralogy of Two Cobalt Occurrences in British Columbia.

By H. V. WARREN, *Associate*, and R. M. THOMPSON.

With Appendix by F. A. FORWARD and A. B. LYLE.

Professor H. V. Warren: In reply to Dr. S. W. Smith's comments it need only be said that the cobalt concentrates referred to on p. 7 of the paper were from Hedley and the cobalt analysis given on p. 10 from the Little Gem Prospect. A similar answer may be made to Mr. E. J. Pryor. The run of mine ore to which Mr. Pryor makes reference came from Hedley, whereas the analyses given in Table II came from Little Gem samples. Mr. Pryor quite correctly drew attention to the high assay given in the Appendix for the Kelowna concentrate—namely, 0·88 per cent cobalt. It is now known that these concentrates ran much higher than average grade; the reason for this has yet to be explained.

The authors regret deeply, as did one contributor, the absence of stope assays of cobalt. However, it is only fair to state that, the cobalt being unwanted, for the time at any rate, it was only natural that in view of rising costs and labour shortages the company should attempt to cut out every avoidable expense.

The authors are deeply appreciative of the points brought forward in the discussion, and especially to Professor W. R. Jones for his introduction.

Professor F. A. Forward and Mr. A. G. Lyle: The authors of the Appendix join in expressing their warm appreciation of the thoughtful discussion contributed by the members present and by those who added written comment. In replying, it should be pointed out that the Appendix represents only a very much abbreviated account of a series of investigations and that, as the account was intended to suggest a possible economic use of the cobaltiferous ores described in the paper, no attempt was made to describe the details of the cobalt extraction scheme. The Appendix deals only with Kelowna Exploration concentrate and does not apply to the treatment of Little Gem ore.

***Bull.* 471, March, 1945.

The chief matter of concern to those who took part in the discussion was the possible difficulty which would be encountered in conducting the 'interrupted' roast on a commercial scale. During the course of the investigation much thought was given to this phase of the problem and, in consultation with manufacturers of roasting equipment, it was generally agreed that the double roast could be carried out in a single roaster of the 'Edwards' or 'flat-hearth' type. A similar operation has been performed in a plant treating tin sulphide ores using *three* stages of roasting. It is simply necessary to instal a baffle wall at a suitable distance from the feed end of the roaster, heat the chamber so provided to a temperature of 800°C. and withdraw the hot products of combustion which would carry the arsenic oxide (possibly also arsenic vapour) that distills so readily from arsenopyrite. We proposed having the next-hearth-in-line at a slightly lower level than the hearth of the first chamber, in order that the high-sulphur low-arsenic calcine could feed by gravity under the baffle wall. The second hearth would consist of water-cooled plates instead of bricks and would, in turn, be isolated from the third section (sulphating) by a baffle wall. The hearth of the third section was to be slightly lower than the second to permit cooled calcine to flow under the wall to this hearth. Air admitted to the third section was to be carefully controlled and, at the discharge end, some heat was to be applied to effect decomposition of the small amount of ferric sulphate that had formed.

In brief, the roaster would be an Edwards-type machine divided into three sections by baffle walls, with the hearths of the second and third sections stepped below the level of the first and second respectively, the second hearth consisting of water-cooled plates instead of bricks. It is not necessary to use a muffle as suggested by Dr. Levy, though, by using a muffle, it is possible to recover the arsenic as As vapour instead of as As_2O_3—a scheme which has much to recommend it.

The decomposition of the arsenopyrite by distillation is easily effected, without removing much of the sulphur. The analysis of the low-As calcine was as follows: Gold, 1·77 oz./ton; cobalt, 1·49 per cent; arsenic, 0·6 per cent, and sulphur, 21·0 per cent. This material is very soft, porous, and magnetic, resembling pyrrhotite.

The sulphating roast temperature can be readily controlled—very little air need be admitted—the less the better as long as there is sufficient to effect oxidation of the sulphur and iron. A high SO_2 content of the gases over the roasting material should be readily maintained. The extraordinarily large surface area presented by the porous 'pyrrhotite' permits efficient use of oxygen in the air at the low temperatures conducive to SO_3 formation. The oxidation is highly exothermic, but the rate of reaction can be controlled by providing conditions—such as high SO_2 in the gases over the hearth—that tend to establish a desirable equilibrium and retard the burning rate. The large surface area of the porous burning material assures that the furnace will not 'go out' if it is operated below 600°C. Ferrites would form at higher temperatures but there is no reason to expect that temperatures of the order of 800°C. need be encountered in the sulphating stage. Most of the ferrous and ferric sulphate formed can be decomposed by heating the discharge end of the sulphating chamber to about 650°C.—well below the decomposition temperature of cobaltous sulphate (about 725–750°C.)

Cobalt was recovered from the leach solutions by the following procedure :—

(1) Precipitate the ferric iron with calcium carbonate (ferrous sulphate *is* difficult to oxidize on a practical scale, but consideration was given to both the SO_2 method and electrolytic oxidation).

(2) Precipitate copper by controlled addition of sodium carbonate.

(3) Precipitate cobalt by addition of sodium carbonate to the copper-free solution.

The authors appreciate especially the suggestions offered by Mr. R. John Lemmon with respect to the leaching procedure. Unfortunately there has been no opportunity to test these proposals in the laboratory during recent months, but it might be observed that in no instance after water or acid leach was any undue consumption of cyanide observed. Apparently the cobalt salts present which are insoluble in the leach solutions are also insoluble in cyanide solutions. The use of 1 per cent acid increased the Co extraction, but also increased the soluble iron. Consequently, sodium sulphate (or bisulphate) was used in some tests to increase the *water-soluble* Co without causing a parallel increase in Fe solubility. We have not tried the precipitation of cuprous oxychloride as suggested and are not prepared to venture an opinion on the feasibility of this proposal.

In conclusion the authors wish to thank all those who have been kind enough to discuss the Appendix and would welcome further suggestions should they occur to any of those who read the paper or the ensuing discussion.

No. 480. SEPTEMBER, 1946.

Bulletin
of
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CONTENTS.

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The Institution of Mining and Metallurgy

(*Founded* 1892—*Incorporated by Royal Charter* 1915.)

Bulletin No. 480.

SEPTEMBER 12TH, 1946.

LIST OF CONTENTS.

[NOTE.—*The Report of Discussion at the First General Meeting of the Session and Author's Reply to Discussion on a Paper previously submitted are also attached hereto, together with the following new Papers :*

Billingham Mine.
By G. ELAND STEWART, *Associate.*

An Experimental Investigation of the Effects of High Temperatures on the Efficiency of Workers in Deep Mines.
By ANTHONY CAPLAN, M.D., M.R.C.P., and J. K. LINDSAY, *Member.*

The Geology and Opencast Mining of the Jurassic Ironstones of Great Britain.
By W. DAVID EVANS, *Associate.*

Notes on the Estimation of Tonnage and Grade of Some Chromite Dumps.
By N. W. WILSON, *Associate.*]

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Committee on Mining in Great Britain.

NOTICE OF GENERAL MEETING.

The SECOND ORDINARY GENERAL MEETING of the Fifty-sixth 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, OCTOBER 17th, 1946, at 5 o'clock p.m.

The following Papers will be submitted for discussion :

Anglo-American Magnesium Production.

By P. L. Teed, *Member.*

(*Published with July,* 1946, Bulletin.)

An Experimental Investigation of the Effects of High Temperatures on the Efficiency of Workers in Deep Mines.

By Anthony Caplan, M.D., M.R.C.P., and J. K. Lindsay, *Member.*

(*Copy attached hereto.*)

Light refreshments will be provided at 4.15 p.m. for members and friends 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 Council desire to remind members that in addition to the more comprehensive types of papers for discussion at General Meetings, they welcome for publication in the *Bulletin* short notes recording data or describing technical experience, which may be of general interest and value. Such notes are governed by the same rules in regard to acceptance as ordinary papers, but would be open for discussion by correspondence.

FIFTY-SIXTH SESSION : 1946–1947.
DATES OF SUBSEQUENT MEETINGS.

The following dates have been provisionally fixed for subsequent General Meetings of the Institution during the Session 1946–47 :

Thursday,	November 21st, 1946.	Thursday,	March 20th, 1947.
,,	December 19th, 1946.	,,	April 17th, 1947.
,,	January 16th, 1947	,,	May 15th, 1947.
,,	February 20th, 1947.		

VISIT TO BILLINGHAM.

As announced in the July *Bulletin,* arrangements have been made for a small number of members of the Institution to visit the Billingham anhydrite mine, sulphuric acid plant and sulphate plant of Imperial Chemical Industries, Ltd., on Wednesday, September 25th. The accommodation reserved is not quite fully booked up, and members are invited to write to the Secretary immediately if they wish to join the party.

NUFFIELD FOUNDATION FELLOWSHIPS AND SCHOLARSHIPS FOR THE ADVANCEMENT OF EXTRACTION METALLURGY.

The Nuffield Foundation is prepared, for a period of five years, to award a number of fellowships and scholarships with the object of advancing research and training in extraction metallurgy. The awards will be as follows :—

(a) *Travelling Fellowships.*—Five Travelling Fellowships will be offered in each year for members of the teaching staff of universities and approved schools of mines and metallurgy in Great Britain and other parts of the British Commonwealth and Empire, to enable them to visit important mining and metallurgical centres in the long vacation, and to assist them in evolving improved teaching methods. The value of a Fellowship will be up to £500. This sum includes travelling expenses. The tenure of the Fellowship will be for approximately three months. Both the amount and the tenure of the Fellowship award will be determined, according to individual needs, by the Nuffield Foundation in consultation with the Institution of Mining and Metallurgy.

(b) *Travelling Post-Graduate Scholarships.*—Five Travelling Post-Graduate Scholarships will be offered in each year for junior members of the profession who are graduates of universities and approved schools of mines and metallurgy in Great Britain and other parts of the British Commonwealth and Empire, and who have specialised in extraction metallurgy. Candidates will be selected not necessarily on account of their order of merit in examinations, but having regard also to their personality and general suitability. The value of a scholarship will be up to £500, to be spread over the period of special training, which should not normally exceed six months. The sum of £500 includes travelling expenses.

(c) *Vacation Scholarships.*—Ten Vacation Scholarships for mining and metallurgical students at universities and approved schools of mines and metallurgy in Great Britain and other parts of the British Commonwealth and Empire will be offered in each year to enable such students to travel by air to important mining and metallurgical centres for vacation work. The value of a Vacation Scholarship will be up to £200, to cover the cost of air transport from the students' homes to the mines, mills, or smelters in the country of study, and return to their homes.

All Fellowships and Scholarships mentioned above will be awarded by the Nuffield Foundation in consultation with the Institution. Applications must be received by the Nuffield Foundation by 1st November of the year preceding the year in which the applicant is desirous of obtaining an award. For example, for Fellowships and Scholarships to be granted for 1947, applications should be received by the Nuffield Foundation by 1st November, 1946.

Full particulars and forms of application for these Fellowships and Scholarships are obtainable from the Secretary of the Nuffield Foundation, 12–13, Mecklenburgh Square, London, W.C.1, to whom they should be returned when completed. When applying to the Nuffield Foundation for application forms, candidates must state clearly whether they are desirous of being awarded a Travelling Fellowship, Travelling Post-Graduate Scholarship, or Vacation Scholarship. The requisite application form—they differ in all three cases—will then be sent to the candidate.

THE GEOLOGY AND MINERAL RESOURCES OF BRITISH GUIANA.

The fourth lecture in the series ' Recent Progress in Geological Investigation and Mineral Developments in the Colonies ', arranged by the Mineral Resources Department of the Imperial Institute, will be given on Thursday, September 26th, at 3 p.m. in the Cinema Hall of the Imperial Institute, South Kensington, S.W.7 (East Entrance), for which no tickets of admission will be required. Mr. S. Bracewell, B.Sc., A.R.C.S., D.I.C., Director of the Geological Survey of British Guiana, will lecture on ' The Geology and Mineral Resources of British Guiana ', and the Chair will be taken by Sir Frank Stockdale, G.C.M.G., C.B.E.

BALLOTING LIST FOR ELECTION OF MEMBERS OF COUNCIL.

The Council wish to draw the attention of Members and Associates to Section IV, clause 6, of the By-Laws, which contains the following provision : ' The Council shall receive the name of any Member or Members, submitted in writing by any Member or Associate previous to November 1st in any year, and shall decide by ballot upon the inclusion thereof, or otherwise, in the balloting list '.

Any Member or Associate who wishes to suggest a name for inclusion in the balloting list for the election of Members of Council for the Session 1947–48 is requested to notify the Secretary of the Institution before November 1st, 1946.

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 July 11th, 1946 :—

To Membership—

Dumbleton, Norris Ayliff (*Gatooma, Southern Rhodesia*).
Truscott, Percy Hugh (*Kitwe, Northern Rhodesia*).
Yeo, Leonard (*Pontyclun, Glamorganshire*).

To Associateship—

Batty, Ralph Frank (*Hove, Sussex*).
Bennetts, Thomas William (*Bukuru, Northern Nigeria*).
Dickson, Alastair Francis (*Grootfontein, South West Africa*).
Dudman, Ian Stanley (*Van Ryn, Transvaal*).
Marsh, Alan (*Gatooma, Southern Rhodesia*).
Parish, Godfrey Woodbine (*London*).

The following have applied for admission into the Institution since July 11th, 1946 :—

To Membership—

Talbot, Geoffrey William (*Buxton, Derbyshire*).

NEWS OF MEMBERS—*continued.*

Mr. H. N. B. DOVER, *Student*, having been demobilized, has rejoined the staff of Assam Oil Co., Ltd.

Dr. K. C. DUNHAM, *Member*, has received the honorary degree of Doctor of Science of the University of Durham.

Mr. P. P. EDWARDS, *Associate*, is returning to England from India.

Mr. A. FAIRFAX-SCOTT, *Associate*, has returned to England from Nigeria.

Mr. E. A. FISHER, *Associate*, has returned to Malaya and has resumed employment with The Eastern Smelting Co., Ltd., Penang, as assistant works manager.

Mr. F. H. FITCH, *Student*, has left for Perak after furlough in England, to rejoin the Geological Survey, Malaya.

Mr. J. H. FRENCH, *Member*, has left the Guinea Fowl Mines Training Camp to take charge of all milling operations on the Connemara mine, Southern Rhodesia.

Mr. H. F. C. GREENWOOD, *Associate*, has returned to Malaya to resume his consulting engineering practice at Ipoh.

Mr. C. E. GREGORY, *Member*, has been demobilized from the Australian Imperial Force and has accepted the post of general manager to the Adelaide Chemical and Fertilizer Co., Ltd.,

Mr. S. V. GRIFFITH, *Member*, has left the Gold Coast for Nigeria.

Mr. ALAN C. HARRISON, *Member*, has returned to the Burma Corporation, Ltd., Namtu, from England.

Major G. F. HATCH, *Associate*, has recently been demobilized from the army and has joined Selection Trust, Ltd. (This note is in amendment of that published in the previous issue.)

Mr. G. L. HATHERLY, *Student*, has left England to take up duties with New Consolidated Gold Fields, Ltd., Johannesburg.

Mr. J. HAYS, *Student*, has left England to join the staff of the Rhokana Corporation, Ltd.

Mr. J. F. G. R. HEYWOOD, *Member*, formerly manager of Consolidated Main Reef mine, is now general manager of Crown Mines.

Dr. G. V. HOBSON, *Member*, is now Assistant Director of Opencast Coal Production, Ministry of Fuel and Power, in charge of the development section.

Mr. J. G. HORNE, *Associate*, has arrived in England on leave from Nigeria.

Mr. A. HUDDLESTON, *Associate*, has been transferred from the Gold Coast Geological Survey to the Mining and Geological Department, Kenya, as geologist.

Mr. J. H. JACKSON, *Associate*, has been demobilized on his return from Palestine.

Mr. H. E. JEFFERY, *Associate*, is home on leave from the Kolar Gold Fields and will be returning to India towards the end of the year.

Mr. M. C. KAIN, *Associate*, has returned to Thai Tin Syndicate, Ltd., Penang, from New Zealand.

NEWS OF MEMBERS—*continued.*

Mr. C. S. Kneebone, *Member*, has left South Australia and returned to Malaya.

Mr. J. V. Lake, *Member*, has resigned his post as general manager of Al Consolidated Gold, N.L., but remains as consulting engineer to the company.

Mr. R. Landcastle, *Student*, has been demobilized and is leaving England for Tanganyika Territory on his appointment as an inspector of mines.

Mr. H. N. Lightbody, *Associate*, has been appointed manager to the Rhodesian Iron and Steel Commission's iron and limestone mines at Que Que.

Mr. J. K. Lindsay, *Member*, is in Scotland on leave from India.

Mr. W. H. J. Luck, *Student*, has been demobilized from the Navy and has now completed a refresher course at Camborne School of Mines.

Mr. D. H. McCall, *Associate*, is now in Scotland on leave from the Kolar Gold Field.

Mr. J. B. Mackie, *Associate*, has been appointed technical adviser to the Chinese Tin Mines Loans Committee, Malaya, for the rehabilitation of the Chinese mining industry.

Mr. G. C. Morgan, *Associate*, has left England for West Africa.

Mr. E. N. G. Morris, *Associate*, is returning from Frontino gold mines and expects to arrive in England before the end of the year.

Mr. P. A. Nicholls, *Student*, has been discharged from the Royal Australian Engineers, A.I.F., and has been appointed surveyor to Hill 50 Gold Mine, N.L., Western Australia.

Mr. T. V. O'Hare, *Associate*, has left England on his return to Thai Tin Syndicate, Ltd., Renong.

Mr. G. E. Osborne, *Student*, has left England to join the staff of Pahang Consolidated Co., Ltd.

Mr. D. de V. Oxford, *Associate*, has left the Roan Antelope copper mines to take up an appointment as assistant consulting engineer to Falcon Mines, Ltd., Bulawayo.

Mr. A. M. Pilter, *Associate*, has returned to Malaya.

Mr. G. H. Pinfield, *Associate*, has been demobilized and has joined the staff of Imperial Chemical Industries, Ltd.

Mr. R. S. H. Richards, *Member*, has arrived in England on two months' leave from Portugal.

Mr. F. L. Rickard, *Associate*, has been demobilized from H. M. Forces.

Mr. J. E. Robson, *Associate*, has arrived in England on leave from the Gold Coast.

Mr. J. V. Ronaldson, *Associate*, has been appointed assistant controller (metallurgy) to the Control Commission, Germany.

Mr. G. A. Schnellmann, *Associate*, has joined the staff of Millom & Askam Hematite Iron Co., Ltd., as geologist.

Mr. R. H. Skelton, *Member*, has returned to England from Western Australia.

Mr. A. Sloss, *Associate*, has left England to take up the appointment of superintendent of mines, Cyprus Asbestos Co., Ltd.

NEWS OF MEMBERS—*continued.*

Mr. W. L. Stewart, *Member*, for five years mining engineer and representative of the Iron and Steel Control (Home Ore Department) in the N.W. District, has returned to Spain.

Mr. E. H. Taylor, *Associate*, has been demobilized and is now at Gopeng, Malaya.

Mr. D. A. Temple, *Student*, has resigned his position as works metallurgist at Airspeed, Ltd., Portsmouth, in order to undertake research work under Professor Wesley Austin at Cambridge University.

Mr. R. C. Vivian, *Student*, has left Tanganyika to join the staff of Nchanga Consolidated Copper Mines, Ltd.

Mr. H. B. Watson, *Associate*, has received an appointment with United British Oilfields of Trinidad, Ltd. and has left England.

Mr. F. H. Way, *Associate*, has left England on his return to Malaya.

Mr. T. A. Wellsted, *Associate*, is at present in England from India.

Mr. G. Benn White, *Member*, is leaving India shortly on retirement.

Mr. G. A. Whitworth, *O.B.E.*, *Member*, has recently been appointed Principal of the School of Metalliferous Mining (Cornwall), which position he has filled since the death in 1941 of the late Mr. H. Standish Ball.

Mr. W. R. Williams, *Associate*, has accepted the post of geologist with the Government of Iraq, and expects to leave soon for Baghdad.

Mr. L. S. Wilson, *Student*, has joined the London Engineering Department of Selection Trust, Ltd.

Mr. L. Wilton, *Student*, has arrived in South America to join the staff of Frontino Gold Mines, Ltd.

Mr. R. B. Woakes, *M.C.*, *Member*, has left England on a short professional visit to Cyprus.

Mr. W. G. Woolston, *Associate*, has returned to England from India.

Mr. F. O. Wright, *Associate*, has left England to join the staff of Sierra Leone Chrome Mines, Ltd.

INFORMATION REQUIRED.

The Secretary would be glad to receive news of the following Members, Associates and Students from whom no communications have reached the Institution for several years :

A.—*Reported as having been prisoners of war or internees.*

Germany : Lt. the Hon. D. C. Feilding, R.E., *Student* ; Lt. R. H. S. Haydon, R.E., *Associate*.

Java : M. C. FitzHerbert, R.A.F., *Student*.

Siam : L/Cpl. W. L. H. Morrison, F.M.S.V.F., *Associate*.

B.—*Members whose last address was in an enemy or enemy-occupied country.*

Belgium : J. G. J. Parfondry, *Associate*.

Burma : G. L. Loonba, *Student* ; G. C. J. Rotter, *Student*.

Channel Islands : R. B. D. Jackson, *Student* ; C. P. Journeaux, *Student*.

INFORMATION REQUIRED—*continued.*

CHINA : K. Y. Kwang, *Member.*
DENMARK : R. Underwood Jarvis, *Member.*
GERMANY : P. Trotzig, *Member.*
HOLLAND : M. A. Provily, *Student.*
ITALY : A. Lheraud, *Associate* ; F. J. Wydler-Hollis, *Associate.*
JAPAN : H. Hunter, *Associate.*
KOREA : K. I. Yun, *Member.*
MALAYA : W. J. D. Kloezeman, *Associate* ; J. Thomson, *Student* ; A. T. Wood, *Student.*
MANCHURIA : B. J. Bryner, *Associate.*
NETHERLANDS EAST INDIES : N. J. Kuiper, *Associate* ; A. B. M. Meyer, *Student.*
PHILIPPINE ISLANDS : M. M. Aycardo, Jr., *Student* ; F. Garrido, *Member* ; A. B. Rowe, *Associate* ; A. Wellhaven, *Member.*
SIAM : A. Buranasiri, *Student* ; A. L. Fredericks, *Associate* ; R. M. Hannah, *Student* ; C. C. W. Liddelow, *Member* ; P. Sukhum, *Associate.*

ADDRESSES WANTED.

E. A. Banning.	L. W. Elsum.	A. McCall.
J. S. Burns.	A. W. Eyre.	E. I. Robinson.

OBITUARY.

Sergeant Alexander John Bell, F.M.S.V.F., is reported by one of his fellow-prisoners to have been shot by the Japanese after an attempt to escape from one of the railroad camps. He would then have been about 30 years of age. He was a student at Ballarat School of Mines and Industries, Victoria, Australia, from 1929 to 1932, and was granted the Diploma and admitted an Associate of the School in 1934. For the first 3 months of 1933 he worked as cyanide shiftman at the Golden Ridges cyanide plant of New Guinea Goldfields, Ltd., subsequently holding the position of chief assayer until December, 1935. After two months at Edie Creek Gold Mining Co., N.L., Mr. Bell transferred first to the Edie Creek assay office of New Guinea Goldfields, Ltd., and later to their mill and cyanide plant. At the end of 1936 he took up the appointment of metallurgist in charge of Edie mine pilot mill and plant under Enterprise of New Guinea Gold and Petroleum Development, N.L., in whose service he remained until 1939, when he left New Guinea and joined the staff of Raub Australian Gold Mining Co., Ltd., Pahang, Malaya.

It is known that he was a sergeant in the F.M.S.V.F. and apart from a report of his imprisonment at Moulmein camp, Burma, nothing had been heard of him until the news of his death. It is understood that he left the prisoner-of-war camp to contact some natives who had promised to guide a party to the border, but on finding that they intended to sell the party to the Japanese he endeavoured to return to camp to warn his friends but was captured by a Japanese sentry. He is reported to have been shot when he refused to reveal his associates. He was elected to Associateship of the Institution in 1939.

OBITUARY—*continued.*

Sverre Blekum died in 1943 at the age of 57. He was a Norwegian and studied mining at the University of Christiania from 1907 to 1912. He began his career as assistant chemist with Professor Tarup, and in 1914 went as geologist to the Spitzbergen Coal and Trading Co.'s mines at Advent Bay, Spitzbergen, for six months. He held the position of chemist at Knaben mines for fifteen months from 1915 to 1916 and was subsequently appointed manager of Ornehommen molybdenite mines, Flekkefjord. He specialized in the mining and flotation of molybdenite and from 1918 until his death was manager and technical director of A/S Knaben Molybdængruber, Norway. He was elected to Associateship of the Institution in 1918.

Sergeant Alexander Burns, F.M.S.V.F., is reported to have died in 1943 of sickness, probably cholera, while a prisoner of war, shortly after being sent to the Japanese rail road in Siam. He was 29 years of age. When elected to Studentship of the Institution in 1939 he was in the fourth year of his course at the Otago University School of Mines, and early in 1940 he joined the staff of Raub Australian Gold Mining Co., Ltd., at Raub, Pahang, where he attained the position of mill manager. No news of him had been received after the Japanese occupation of Malaya, and it has since been learned that he was captured at the fall of Singapore.

George Ernest Collins died at the age of 76 in hospital in Ouray, Colorado, on May 4th, 1946, of injuries received two days earlier when the mine bus in which he was returning from inspecting a mine of the American Smelting and Refining Co. left the highway and plunged down a steep boulder-strewn slope. He was born in Truro, Cornwall, and studied the profession under his father, the late Mr. J. H. Collins, a founder-member and past-president of the Institution, and was made a partner in July, 1889, of the firm J. H. Collins & Son. His career in America, the whole of which was centred on mining in Colorado, began in 1895 as general assistant to Mr. A. L. Collins at Central City and joint manager of the Central Development Syndicate, Ltd. From 1898 to 1900 he was manager of Reynolds and St. George mines of The Gold Reefs of Georgia, Ltd., and then took charge of other large mills and mines of the Central Development Syndicate. In 1901 he was managing the Polar Star mine, San Juan County, Colo., a property in which he subsequently acquired a controlling interest. Since 1902 he had maintained a practice as consulting mining engineer with an office in the Boston Building, Denver. Between 1902 and 1910 he managed various properties in Gilpin and Clear Creek Counties ; in 1909 he was instrumental in opening up valuable tungsten deposits in Boulder County, and also assumed the managership of the Argo tunnel, Clear Creek County, which was completed in 1910, in which year he also became manager of the Druid mine, Gilpin County, with a controlling interest. A few years later he took up the position of manager of the Mary Murphy Gold Mining Co., at Romley, Colo. This company acquired, in 1917, the Red Mountain mines at Ouray, which had since been worked continuously on a small scale under Mr. Collins's management. He took an active interest in the introduction and adaptation of rock drills to mining and the use of

OBITUARY—*continued*.

steel sets instead of timber. He was also prominent in fostering early developments of flotation and electro-static mineral separation. Among his writings are two papers which he contributed to the *Transactions* of the Institution—' Vein structure at the Reynolds mine, Georgia ' (vol. 9, 1900–1) and ' The relative distribution of gold and silver values in the ores of Gilpin County, Colorado ' (vol. 12, 1902–3).

He was elected an Associate of the Institution in 1895 and was transferred to Membership in 1900. He was also a member of the American Institute of Mining and Metallurgical Engineers and member and chairman of the Committee of Professional Conduct of the Mining and Metallurgical Society of America ; he was also an ex-president of the Colorado Scientific Society and an ex-governor of the Colorado branch of the American Mining Congress.

Hereward Bosworth Hall is presumed to have died on or about February 13th, 1942, as a result of enemy action during the evacuation from Singapore. He was a student at Armstrong College, University of Durham, from 1919 to 1922, and obtained a B.Sc. degree in non-ferrous metallurgy. He was employed from 1924 on general metallurgical chemistry, mainly on tin ores, slags and metal, by the Straits Trading Co., Ltd., of Singapore, first as assistant chemist for four years, then as chemist for ten years. In 1935 he was designated works chemist and a year later was appointed works assistant to the company. In 1939 Mr. Hall returned to England in the position of works manager to The British Tin Smelting Co., Ltd., at Liverpool, but rejoined the Straits Trading Co. in Singapore in 1940. No news of him having been received since the fall of Singapore, the Colonial Office have reluctantly concluded that there can no longer be any hope of his survival.

Mr. Hall was elected to Associateship of the Institution in 1928.

George Allan More died on June 12th, 1946, at Neutral Bay, Sydney, N.S.W., at the age of 67. He was born at Singleton, N.S.W., and received his training at Sydney University from 1896 to 1900, graduating with the degree of B.Eng. (Min. and Met.). He began his career at the Mt. Cobalt Black Snake gold mine, Queensland, and after varied experience left to join Overflow Mines, N.L., N.S.W., as assayer and surveyor. From 1902 to 1906 he was employed by Tasmania Gold Mining and Quartz Crushing Co., Ltd., and for fifteen months in 1906 and 1907 was managing the Primrose mine at Rosebery, Tasmania. In the latter part of 1907 he acted as consulting engineer at the Colebrook copper mine, and in January, 1908, was appointed manager of the Mt. Lyell Comstock mine. After eight months Mr. More resigned, and carried out mine examinations in Australia and New Zealand, but at the beginning of 1909 took up the position of chief mining engineer to Great Cobar, Ltd., N.S.W. He left ten months later to become mine manager for North Mt. Boppy, Ltd., operating in Hargraves District, N.S.W., where his particular work was to develop a gold mining prospect for the company. This property was finally closed down at the end of 1912 and Mr. More resumed private examination and reporting. He was appointed chief mining engineer to Great Cobar, Ltd., in 1915, but, pending re-starting of operations, made valuations for the

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

The address of the Library is now 424 Salisbury House, London, E.C. 2, and books (excluding periodicals) may be borrowed by members on personal application or by post.

AMERICAN BUREAU OF METAL STATISTICS: YEAR BOOK, 1945. N.Y.: The Bureau. 1945. 112 p. (*Presented by the Director of the Bureau.*)

BRITISH STANDARDS: No. 229, 1946. Flameproof enclosure of electrical apparatus. 45 p. 3s. 6d.

CANADA: PROCEEDINGS OF THE STANDING COMMITTEE ON NATURAL RESOURCES ON THE ECONOMIC VALUE OF METALLIFEROUS MINES IN CANADA. Ottawa: Govt. Printer. 1946. 202 p.

CANADIAN INSTITUTE OF MINING AND METALLURGY: TRANSACTIONS. VOL. 48, 1945. Montreal: The Institute. 1946. 794 p.

COAL INDUSTRY NATIONALISATION BILL. (As amended in Committee.) London: H.M.S.O. 1946. 1s.

CONTROL OF IMPOUNDING STRUCTURES ON ORE DEPOSITION, by Robert A. Mackay. Reprinted from *Economic Geology*, Vol. 61, Jan.–Feb., 1946. 33 p. (*Presented by the Author.*)

ELECTROCHEMICAL SOCIETY: TRANSACTIONS. VOL. 87. N.Y.: The Society. 1946. 609 p.

FOREST OF DEAN COALFIELD. REGIONAL SURVEY REPORT. Ministry of Fuel and Power. London: H.M.S.O. 1946. 58 p. 1s. 3d.

GEOLOGICAL SOCIETY OF SOUTH AFRICA: TRANSACTIONS. VOL. 48, JAN.–DEC., 1945. Johannesburg: The Society. 1946. 234 p. £2 2s.

GT. BRITAIN. BOARD OF TRADE: REPORT OF THE ENQUIRY ON THE BALL CLAY INDUSTRY. W. R. Jones, *chairman of committee*. London: H.M.S.O. 1946. 14 p. 3d.

HANDBOOK OF NON-FERROUS METALLURGY, 2ND EDN. VOL. 1.—PRINCIPLES AND PROCESSES. D. M. Liddell, *ed.-in-chief*. N.Y.: McGraw Hill. 1945. 656 p. 39s.

INSTALLATION AND MAINTENANCE OF BOILER HOUSE INSTRUMENTS. Ministry of Fuel and Power. London: The Ministry. 1946. 24 p. (Fuel Efficiency Bulletin No. 45.)

NEW MEXICO. BUREAU OF MINES AND MINERAL RESOURCES: BULLETIN NO. 24. BUILDING BLOCKS FROM NATURAL LIGHTWEIGHT MATERIALS OF NEW MEXICO, by Donn. M. Clippinger. Socorro: The Bureau. 1946. 40 p.

RAPID SURVEY OF COAL RESERVES AND PRODUCTION; A FIRST APPRAISAL OF RESULTS. Gt. Britain. Fuel Research Board. London: H.M.S.O. 1946. 23 p. (Survey Paper No. 58.) 9d. (*Presented by the Director of Fuel Research.*)

REPORTS OF THE BRITISH AND COMBINED INTELLIGENCE OBJECTIVES SUB-COMMITTEES (B.I.O.S. and C.I.O.S.), FIELD INFORMATION AGENCY, TECHNICAL (F.I.A.T.), AND JOINT INTELLIGENCE OBJECTIVES AGENCY (J.I.O.A.). Various reports have been presented by the Department of Scientific and Industrial Research and are listed under the Index of Recent Articles.

LIST OF ADDITIONS TO THE JOINT LIBRARY—*continued.*

REVIEW OF THE TECHNICAL PROBLEMS OF THE POLISH COAL MINING INDUSTRY, by G. A. Reicher. 1944. 32 p. (Typescript pamphlet.) (*Presented to the Institution of Mining Engineers by the Author.*)

SOME UNUSUAL WOLFRAM AND CASSITERITE LODES, by O. A. Jones. Brisbane: University of Queensland. 1941. 6 p. (Queensland University. Dept. of Geology. Paper No. 8.) 1s. (*Presented by the Author.*)

SPECIAL RULES FOR HÆMATITE IRON ORE MINES. Metalliferous Mines Regulation Acts, 1872 and 1875. 1946. (*Presented by Lt. Col. A. Hibbert, D.S.O., M.C.*)

TANGANYIKA. GEOLOGICAL DIVISION. BULLETIN NO. 15. EXPLANATION OF THE GEOLOGY OF DEGREE SHEET NO. 17 (KAHAMA), by D. R. Grantham, B. N. Temperley, and R. B. McConnell. Tanganyika: Geological Division. 1945. 39 p. map (scale 1/250,000). 4s. (50 cents).

TIMBER PESTS. London: Timber Development Association. 1946. 60 p. biblio. (*Presented by the Timber Development Association.*)

TIMBER SEASONING. London: Timber Development Association. 1946. 112 p. biblio. (*Presented by the Timber Development Association.*)

TIN, TUNGSTEN, AND MOLYBDENUM DEPOSITS OF AUSTRALIA. An annotated bibliography with a discussion of the geology of the deposits and of problems connected with their mining, by O. A. Jones. Brisbane: Queensland University. 1943. 49 p. (Queensland University. Dept. of Geology. Paper No. 9.) 3s. (*Presented by the Author.*)

MAPS.

ASHCROFT, BRITISH COLUMBIA. Geological Survey Preliminary Map 46–8. Scale: 1 in. = 2 ml. Ottawa: Bureau of Geology and Topography. 1946.

BRULE MAP AREA, ALBERTA. Geological Survey Preliminary Map 46–5. Scale: 2 in. = 1 ml. Ottawa: Bureau of Geology and Topography. 1946.

CANOL ROAD, TESLIN RIVER TO MACMILLAN PASS, YUKON. Geological Survey Preliminary Map 45–21A and Paper 45–21, 2nd ed. Scale: 1 in. = 4 ml. Ottawa: Bureau of Geology and Topography. 1946.

ENTRANCE, WEST OF FIFTH MERIDIAN, ALBERTA. Geological Survey Map 843A. Scale: 1 in. = 1 ml. Ottawa: Bureau of Geology and Topography. 1945.

SALMON ARM, BRITISH COLUMBIA. Geological Survey Preliminary Map 46–7. Scale: 1 in. = 2 ml. Ottawa: Bureau of Geology and Topography. 1946.

SUSSEX, KINGS AND QUEENS COUNTIES, NEW BRUNSWICK. Geological Survey Map 845A. Scale: 1 in. = 1 ml. Ottawa: Bureau of Geology and Topography. 1945.

TAKLA, CASSIAR DISTRICT, BRITISH COLUMBIA. Geological Survey Map 844A. Scale: 1 in. = 4 ml. Ottawa: Bureau of Geology and Topography. 1945.

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.

ALLOYS AND METALLOGRAPHY.

ALLOYS—MANGANESE COPPER.—Heat contents above 25° C. of seven manganese-copper alloys. B. F. Naylor.—*U.S. Bur. Min. Rep. Invest.* No. 3835, Wash., D.C., Jan., 1946 13 p.

METALLOGRAPHY — RUSSIA. — Russian metallography: Pt. 2—The electron atom characteristics of phases; Pt. 3—Discussion of berthollide phases. G. Stanley Smith.—*Metal Ind.*, Lond., Vol. 68, 1946; June 28, pp. 516–18; June 21, pp. 495–98. 6d. ea.

ANALYSIS AND CHEMISTRY.

ANALYSIS — ANTIMONY. — The micro-analytical chemistry of antimony. F. T. Beaumont.—*Metallurgia*, Manchr., Vol. 33, Dec., 1945, pp. 101–03. 2s.

ANALYSIS—CHROMIUM.—Note on determination of minute quantities of chromium in igneous rocks. A. J. van der Merwe.—*Trans. Geol. Soc. S. Afr.*, J'burg, Vol. 48, Jan.–Dec., 1945, pp. 15–16. £2 2s.

ANALYSIS — METALLURGY — POLAROGRAPHIC.—Applications of the polarograph to metallurgical analysis. G. W. C. Milner.—*Metallurgia*, Manchr., Vol. 33, April, 1946, pp. 321–23. 2s.

CHEMISTRY — DENSITY. — The micro-determination of density. Cecil L. Wilson.—*Metallurgia*, Manchr., Vol. 33, Jan., 1946, pp. 157–60. 2s.

CHEMISTRY — WATER SOFTENING. — Water treatment developments. R. P. Donnelly.—*Colliery Engng.*, Lond., Vol. 23, Aug., 1946, pp. 207–10. 1s.

CHEMISTRY—WOOD CARBONIZATION—GERMANY.—Wood carbonization industry of Germany.—B.I.O.S. Final Report No. 444, 1945, 60 p.

ASSAYING AND SAMPLING.

ASSAYING—IRON—SPECIFIC GRAVITY METHOD.—Rapid specific-gravity method for estimating the iron content of Birmingham, Ala., red ores. I. L. Feld, G. D. Coe, and Will H. Coghill.—*U.S. Bur. Min. Rep. Invest.* No. 3838, Wash., D.C., Jan., 1946, 6 p.

COAL.

ANTHRACITE—WASTE.—Disposal of solids in breaker waste water by impounding in surface basins. W. C. Muehlhof and L. D. Lamont.—(*A.I.M.E. Tech. Pub.* No. 1959) *Coal Tech.*, N.Y., Vol. 1, Feb., 1946, 9 p. $1.60.

COAL—*continued*.

AYRSHIRE — NEW CUMNOCK. — Some features in the development of the main coal seam in the New Cumnock coalfield. John C. George.—*Iron Coal Tr. Rev.*, Lond., Vol. 152, June, 1946, pp. 1101–09.

BY-PRODUCTS — HYDROCARBONS. — Stickstoffwerk, Hibernia Wanne-Eickel, Ruhr (Recovery of hydrocarbons from coke-oven gas).—C.I.O.S. Item No. 30, File No. XXXIII-30, 1945, 5 p.

BY-PRODUCTS—SULPHUR.—Sulphur recovery from spent purifier oxide: Ruhrgas A.G.—Herwarth Strasse, Essen.—C.I.O.S. Item No. 30, File No. XXXIII-29, 1945, 4 p.

BY-PRODUCTS — TAR — GERMANY. — Gesellschaft für teerverwertung m.b.H., Duisberg, Meiderich and Castrop-Rauxel (Tar distillation plant).—B.I.O.S. Final Report No. 480, Item No. 30, 55 p.

CAPE PROVINCE.—Coalfields of the Cape Province—fresh examination to be made.—*S. Afr. Min. Engng. J.*, J'burg, Vol. 57, Part 1, June 1, 1946, pp. 341–42. 6d.

COAL GAS—STORAGE.—Les nouvelles techniques dans la compression du gaz de houille carburant. M. Pignot.—*Mem. Soc Ingenieurs Civils, France*, Paris, June–Aug. 1945, pp. 217–26.

COKE—COKE OVENS—DIDIER TYPE.—Study of slanting type Didier coke ovens, Stadtische Werke, Karlsruhe.—F.I.A.T. Final Report No. 566, Nov., 1945, 7 p.

COKING PRACTICE—HIGH TEMPERATURE GASIFICATION—LURGI PROCESS.—Lurgi high pressure gasification.—B.I.O.S. Final Report No. 521, Item No. 30, 1946, 56 p.

COMPOSITION — VEGETABLE CONSTITUENTS.—Observations on the vegetable constituents of coals. William C. Darrah.—*Econ. Geol.*, Lancaster, Pa., Vol. 36, Sep.–Oct., 1941, pp. 589–611. $1.

DISTILLATION — HYDROGENATION — CATALYSTS.—Manufacture and regeneration of catalysts at I.G. Farbenindustrie, Ludwigshafen, Oppau.—B.I.O.S. Final Report No. 422, 1945, 9 p.

DISTILLATION — HYDROGENATION — FISCHER TROPSCH PROCESS.—Ruhr-Chemie A.G. Sterkrade Holten: interrogation of Dr. O. Roelen.—B.I.O.S. Final Report No. 511, Item No. 30, 1945, 25 p.

DISTILLATION — HYDROGENATION — GERMANY.—Report on investigations by fuels and lubricants teams at the I.G. Farbenindustrie A.G. Works, Ludwigshafen and Oppau, Item No. 30, File No. XXX-103, 1945, 67 p.

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COAL—*continued.*

DUST—DETERMINATION—Apparatus for the rapid assessment of dustiness of mine air. J. Ivon Graham and H. A. Stanley.—*B.C.O.R.A. Quart. Bull.*, Lond., Dec., 1945, pp. 6–7.

DUST—SUPPRESSION.—Dust suppression (pneumokoniosis); experiments at a Kent colliery. F. H. Price.—*Trans. Instn. Min. Engrs.*, Lond., Vol. 105, Pt. 5, Feb., 1946, pp. 244–67. 6s.

DUST — SUPPRESSION — WATERING.—Dust abatement by sprinkling and spraying in coal mines.—*Coal Age*, Albany, N.Y., Vol. 51, June, 1946, pp. 90–7. 35 cents.

DUST—SUPPRESSION—WET CUTTING.—The suppression of dust by wet cutting in the Nottinghamshire and Derbyshire coal field.—*Colliery Guard.*, Lond., Vol. 173, July 5, 1946, pp. 29–31. 10d.

EUROPE.—The position and organisation of the European coal mining industry.—*Min. Industr. Mag. S. Afr.*, J'burg, Vol. 34, May, 1946, pp. 291–95. 1s.

EUROPE (N.W.).—Coal in N.W. Europe—post-hostilities position. W. R. Gordon.—*Colliery Guard.*, Lond., Vol. 173, July 26, 1946, pp. 114–19. 10d.

GAS MANUFACTURE—GERMANY.—Gas utilities in Germany.—C.I.O.S. Item No. 33, File No. XXXII-52, 1946, 34 p.

GEOLOGY — CHINA — SIKANG — PAIKUOWAN.—Geology of Paikuowan coal field, Huili, Sikang. V. C. Juan.—*Bull. Geol. Surv. China*, Pehpei, Chungking, No. 35, April, 1942, pp. 31–3. (English summary.)

GERMANY — UPPER SILESIA. — Coal mining in German Upper Silesia. John David.—*Colliery Guard.*, Lond., Vol. 173, July 19, 1946, pp. 106–10. 10d.

HAULAGE — EDINBURGH — WHITEHILL COLLIERY.—Haulage reorganisation at Whitehill colliery. R. McAdam.—*Colliery Engng.*, Lond., Vol. 23, 1946; July, pp. 184–7; Aug., pp. 203–6. 1s. ea.

HAULAGE—THIN SEAMS.—Supply handling for efficiency in trackless coal mining.—*Coal Age*, Albany, N.Y., Vol. 51, June, 1946, pp. 80–5. 35 cents.

HOISTING—SKIPS.—Seven years' experience of skip winding. E. Hart.—*Trans. Instn. Min. Engrs.*, Lond., Vol. 105, Pt. 6, March, 1946, pp. 271–85. 6s.

HOISTING—SKIPS.—The skip winding of coal. John W. Wardell.—*Colliery Engng.*, Lond., Vol. 23, July, 1946, pp. 168–72. 1s.

KENTUCKY—HOPKINS CO.—MOSS HILL NO. 9 MINE.—Special equipment boosts efficiency at trackless operation. R. R. Richart.—*Coal Age*, Albany, N.Y., Vol. 51, July, 1946, pp. 83–7. 35 cents.

COAL—*continued.*

LANCASHIRE — MOSTON COLLIERY. — Reorganisation at Moston colliery.—*Colliery Engng.*, Lond., Vol. 23, Aug., 1946, pp. 197–98. 1s.

LEICESTERSHIRE — DONISTHORPE COLLIERY.—Donisthorpe colliery; the use of power loading for development work.—*Colliery Guard.*, Lond., Vol. 172; June 28, 1946, pp. 867–71; Vol. 173, July 5, 1946, pp. 1–3. 10d. ea.

LOADING — MECHANICAL — HUWOOD LOADER.—Coal-face loading machine.—*Compr. Air Hydr. Engng.*, Lond., Vol. 11, May, 1946, pp. 9–12. 1s.

LOADING — MECHANICAL — HUWOOD LOADER.—Coal-face loading machine.—*Compr. Air Hydr. Engng.*, Lond., Vol. 11, May, 1946, pp. 9–12. 35 cents.

MINING METHODS — MECHANICAL. — Mechanized coal mining. Interaction of machines and mining methods. J. I. Davies.—*Compr. Air Hydr. Engng.*, Lond., Vol. 11, May, 1946, pp. 5–9. 35 cents.

MINING METHODS — MECHANICAL. — Mechanized coal mining. J. I. Davies.—*Compr. Air Hydr. Engng.*, Lond., Vol. 11, May, 1946, pp. 5–9. 1s.

MINING METHODS — MECHANICAL — HAZARDS.—Attainment of greater safety in coal-mine mechanization. D. C. Jones.—(*A.I.M.E. Tech. Pub.* No. 1963) *Coal Tech.*, N.Y., Vol. 1, Feb., 1946, 5 p. $1.60.

MINING METHODS — MECHANICAL — THICK SEAMS.—Mechanical mining in thick seams. George B. Higinbotham. — *Min. Congr. J.*, Wash., D.C., Vol. 32, June, 1946, pp. 39–40. 20 cents.

MINING METHODS — MECHANICAL — THIN SEAMS.—Mechanical mining in thin seams. John J. Snure.—*Min. Congr. J.*, Wash., D.C., Vol. 32, June, 1946, pp. 38–39. 20 cents.

MINING METHODS — MECHANICAL — U.S.A.—Mechanization in coal mining makes rapid progress. A. L. Toenges.—*Min. Metall.*, N.Y., Vol. 27, July, 1946, pp. 393–95. 50 cents.

MINING METHODS — OPENCAST — DRAGLINES.—The large dragline in coal stripping. R. M. Dickey.—*Min. Congr. J.*, Wash., D.C., Vol. 32, June, 1946, pp. 49–50. 20 cents.

MINING METHODS — OPENCAST — KENTUCKY.—Strip mining two seams. R. Paul Mauger and Clayton Ball.—*Min. Congr. J.*, Wash., D.C., Vol. 32, June, 1946, pp. 48–49. 20 cents.

NOVA SCOTIA—INVERNESS.—Inverness coal field. J. P. Messervey.—*N.S. Dep. Min. Annu. Rep.*, 1945, pp. 189–95.

•NOVA SCOTIA—PORT HOOD.—The Port Hood coal area. G. Vibert Douglas.—*N.S. Dep. Min. Annu. Rep.*, 1945, pp. 247–71.

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INDEX OF RECENT ARTICLES—*continued.*

INDEX OF RECENT ARTICLES—*continued.*

GEOLOGY AND ORE DEPOSITS—*continued.*

ORE DEPOSITS—GENESIS — OREGON (S.W.)—The paragenesis of some gold and copper ores of South-western Oregon. Wayne Russell Lowell.—*Econ. Geol.*, Lancaster, Pa., Vol. 37, Nov., 1942, pp. 557-95. $1.

ORE DEPOSITS — GENESIS — QUEBEC —ABITIBI CO.—Paragenesis of the ore of the Normetal mine, Abitibi County, Quebec. Henry M. Putman.—*Econ. Geol.*, Lancaster, Pa., Vol. 38, June-July, 1943, pp. 313-22. $1.

ORE DEPOSITS — GENESIS — TUNGSTEN — COLORADO. — The origin of the tungsten ores of Boulder County, Colorado. T. S. Lovering.—*Econ. Geol.*, Lancaster, Pa., Vol. 36, March-April, 1941, pp. 229-79. 75 cents.

ORE DEPOSITS—GOLD.—The alkali sulphide theory of gold deposition. F. Gordon Smith.—*Econ. Geol.*, Lancaster, Pa., Vol. 38, Nov., 1943, pp. 561-90. $1.

ORE DEPOSITS—MAGMAS.—Magmas and ores. Alan M. Bateman.—*Econ. Geol.*, Lancaster, Pa., Vol. 37, Jan.-Feb., 1942, pp. 1-15. $1.

PETROLOGY — GENESIS — TILLITE—NAMAQUALAND.—The age of the Numees tillite relative to the Nama system. John de Villiers.—*Trans. Geol. Soc. S. Afr.*, J'burg, Vol. 48, Jan.-Dec., 1945, pp. 135-42. £2 2s.

PETROLOGY — IGNEOUS ROCKS.—Comments on the origin and major structural control of igneous rocks and related mineral deposits. G. F. Loughlin.—*Econ. Geol.*, Lancaster, Pa., Vol. 36, Nov., 1941, pp. 671-97. $1.

PETROLOGY — IGNEOUS ROCKS — GRANITES.—Granite and ore. Augustus Locke.—*Econ. Geol.*, Lancaster, Pa., Vol. 36, June-July, 1941, pp. 448-54. $1.

PETROLOGY — IGNEOUS ROCKS — LAMPROPHYRES.—The lamprophyre problem. H. G. Smith.—*Geol. Mag.*, Hertford, Vol. 83, July-Aug., 1946, pp. 165-71. 6s.

VENEZUELA.—Venezuela—the Venezuelan Andes and the coastal and interior ranges. John C. Davey.—*Min. Mag.*, Lond., Vol. 75, Aug., 1946, pp. 77-87. 1s.

GOLD.

CONCENTRATION — JIGS — PORTUGUESE EAST AFRICA.—Gold recovery by Deco jig in Portuguese East Africa. L. L. Colin.—*Deco Trefoil*, Denver, Vol. 10, May, 1946. 4 p.

GEOLOGY.—Gold mineralization in minor igneous intrusions. W. W. Moorhouse.—*Econ. Geol.*, Lancaster, Pa., Vol. 37, June-July, 1942, pp. 318-29. $1.

GEOLOGY—CANADA.—The mechanism and environment of gold deposition in veins. William H. White.—*Econ. Geol.*, Lancaster, Pa., Vol. 38, Sep.-Oct., 1943, pp. 512-32. $1.

GOLD—*continued.*

GEOLOGY — CANADA. — Zoning and district variations of the minor elements in pyrite of Canadian gold deposits. P. E. Auger. —*Econ. Geol.*, Lancaster, Pa., Vol. 36, June-July, 1941, pp. 401-23. $1.

GEOLOGY — MONTANA — BUTTE — HIGHLAND MINE.—Gray quartz breccia orebody of the Highland mine, Butte, Montana. R. C. Newcomb.—*Econ. Geol.*, Lancaster, Pa., Vol. 36, March-April, 1941, pp. 185-98. 75 cents.

GEOLOGY — TANGANYIKA — LUPA GOLDFIELD.—Microscopic study of some ores of the Lupa Goldfield, Tanganyika Territory, East Africa. David Gallacher.—*Econ. Geol.*, Lancaster, Pa., Vol. 36, March-April, 1941, pp. 306-23. 75 cents.

GEOLOGY—U.S.A.—The mechanism and environment of gold deposition in veins. William H. White.—*Econ. Geol.*, Lancaster, Pa., Vol. 38, Sep.-Oct., 1943, pp. 512-32. $1.

NORTHWEST TERRITORIES - YELLOWKNIFE.—Mineralogy of the Negus and Con mines, Yellowknife, Northwest Territories, Canada. G. Carman Ridland.—*Econ. Geol.*, Vol. 36, Jan.-Feb., 1941, pp. 45-70. 75 cents.

ONTARIO—LITTLE LONG LAC.—Gold ores of the Little Long Lac area, Ontario. H. S. Armstrong.—*Econ. Geol.*, Lancaster, Pa., Vol. 38, May, 1943, pp. 204-52. $1.

ONTARIO — POWELL TOWNSHIP—MATACHEWAN MINE.—Matachewan Consolidated mines: a brief description of geology, mining and milling. D. C. McLaren.—*Canad. Min. J.*, Gardenvale, P.Q., Vol. 67, July, 1946, pp. 693-707. 25 cents.

SPAIN.—Gold in Spain.—*Min. World*, Lond., Vol. 151, July 20, 1946, p. 41. 6d.

IRON AND STEEL.

CHINA — SIKANG — HSIAOHEITSING.—Hsiaoheitsing iron ore deposit near Maokupa. Huili district, Sikang. Y. C. Cheng, Y. C. Juan, T. C. Chow and H. K. Hu.—*Bull. Geol. Surv. China*, Pehpei, Chungking, No. 35, April, 1942, pp. 7-10. (English summary.)

CHINA — SIKANG — TSAITSUKOU.—Iron ore deposit of Tsaitsukou, Taofu, Sikang. Y. C. Cheng, K. S. Tsui and T. C. Chow.—*Bull. Geol. Surv. China*, Pehpei, Chungking, No. 35, April, 1942, pp. 13-28. (English summary.)

CHINA — SIKANG — TSIENSHAN. —Iron ore deposits near Tsienshan, Paihuasu, Huili district, Sikang. Y. C. Cheng.—*Bull. Geol. Surv. China*, Pehpei, Chungking, No. 35, April, 1942, pp. 11-12. (English summary.)

FOUNDRY PRACTICE—CUPOLAS.—The basic-lined cupola.—*Engineering*, Lond., Vol. 162, July 5, 1946 pp. 19-20. 1s.

INDEX OF RECENT ARTICLES—*continued.*

IRON AND STEEL—*continued.*

GEOLOGY.—Origin of banded iron deposits —a suggestion. W. G. Woolnough.—*Econ. Geol.*, Lancaster, Pa., Vol. 36, Aug., 1941, pp. 465–89. $1.

GEOLOGY — CHINA — KWEICHOW — KUANYINSHAN.—Geology of Kuanyinshan limonite deposit, Shuicheng district, Kweichow. C. J. Peng.—*Bull. Geol. Surv. China*, Pehpei, Chungking, No. 35, April, 1942, pp. 29–30. (English summary.)

GEOLOGY — MAGNETITE. — Supergene magnetite. John S. Brown.—*Econ. Geol.*, Lancaster, Pa., Vol. 38, March-April, 1943, pp. 137–48. $1.

GEOLOGY — MINNESOTA — MESABI RANGE.—Geologic interpretation of magnetic exploration on the Mesabi Range, Minnesota. R. H. B. Jones.—(*A.I.M.E. Tech. Pub.* No. 2038), *Min. Tech.*, N.Y., Vol. 10, July, 1946, 13 p. $1.25.

GEOLOGY — ONTARIO — STEEP ROCK LAKE.—Hydrothermal replacement in deep seated iron ore deposits of the Lake Superior region. Hugh M. Roberts and Melville W. Bartley.—*Econ. Geol.*, Lancaster, Pa., Vol. 38, Jan.–Feb., 1943, pp. 1–24. $1.

GEOLOGY — QUEBEC — SEPT-ILES AREA.—Titaniferous magnetite deposits of Sept-Iles, Quebec. Carl Faessler and G. M. Schwartz.—*Econ. Geol.*, Lancaster, Pa., Vol. 36, Nov., 1941, pp. 712–28. $1.

MAGNETITE — CHINA — SIKANG — LUKU.—The Luku magnetite deposit, Mienning, Sikang. Y. C. Cheng, T. C. Chow, and H. K. Hu.—*Bull. Geol. Surv. China*, Pehpei, Chungking, No. 35, April, 1942, pp. 1–6. (English summary.)

METALLURGY — SPONGE IRON. — The low temperature gaseous reduction of magnetite ore to sponge iron. O. George Specht, and Carl A. Zapffe.—(*A.I.M.E. Tech. Pub.* No. 1960) *Metals Tech.*, N.Y., June, 1946, 41 p. biblio. $1.35.

PHILIPPINE ISLANDS—LARAP.—The iron deposits of Larap, Philippine Islands. F. H. Kihlstedt.—(*A.I.M.E. Tech. Pub.* No. 2001) *Min. Tech.*, N.Y., Vol. 10, May, 1946, 22 p. $1.35.

SMELTING — REDUCTION. — Selective reduction of iron in chromite by methane-hydrogen and similar gas mixtures. F. S. Boericke.—*U.S. Bur. Min. Rep. Invest.* No. 3847, Wash., D.C., Feb., 1946, 14 p.

WESTERN AUSTRALIA—YAMPI SOUND —COCKATOO ISLAND.—Cockatoo Island iron ore development.—*Iron Coal Tr. Rev.*, Lond., Vol. 153, Aug. 9, 1946, pp. 237–38. 9d.

LEAD AND ZINC.

CONCENTRATION — QUEBEC — VAL D'OR.—The de-leading of zinc concentrate at Golden Manitou mines. D. A. Livingstone.—*Canad. Min. Metall. Bull.*, Montreal, No. 410, June, 1946, pp. 362–66. $1.

LEAD AND ZINC—*continued.*

GEOLOGY — MISSISSIPPI VALLEY. — The Mississippi Valley type lead-zinc deposits and the problem of mineral zoning. R. M. Garrels.—*Econ. Geol.*, Lancaster, Pa., Vol. 36, Nov., 1941, pp. 729–44. $1.

IDAHO — COEUR D'ALENE DISTRICT.—Activity in the Pine Creek area. Harry W. Ingalls.—*Min. Congr. J.*, Wash., D.C., Vol. 32, July, 1946, pp. 40–4. 20 cents.

NEW SOUTH WALES—BROKEN HILL.—Post-mine leaching of galena and marmatite at Broken Hill. M. D. Garretty and Roland Blanchard.—*Econ. Geol.*, Lancaster, Pa., Vol. 37, Aug., 1942, pp. 365–407. $1.

ZINC — GERMANY. — George von Giesche's erben Magdeburg (zinc mining and smelting company).—C.I.O.S. Item No. 21, File No. XXXI-56, 1945, 6 p.

LIGHT METALS.

ALUMINIUM — ALUMINA. — Pilot-plant investigations preparation of alumina from potassium alum. C. T. Baroch, A. W. Hackwood and R. G. Knickerbocker.—*U.S. Bur. Min. Rep. Invest.* No. 3845, Wash., D.C., Feb., 1946, 26 p.

ALUMINIUM — ALUMINA. — Production of alumina. D. D. Howat.—*Mine & Quarry Engng.*, Lond., Vol. 12, Aug., 1946, pp. 41–6. 1s.

ALUMINIUM-BAUXITE—DUTCH EAST INDIES.—Origin and mining of bauxite in Netherlands-India. R. W. Van Bemmelen.—*Econ. Geol.*, Lancaster, Pa., Vol. 36, Sep.–Oct., 1941, pp. 630–40. $1.

ALUMINIUM — BAUXITE — OREGON. — Ferruginous bauxite deposits in north-western Oregon. F. W. Libbey, W. D. Lowdry, and R. S Mason.—*Econ. Geol.*, Lancaster, Pa., Vol. 41, May, 1946, pp. 246–65. $1.

ALUMINIUM — METALLURGY — SECONDARY METAL.—Secondary aluminium. F. H. Smith.—*Metallurgia*, Manchr., Vol. 33, Feb., 1946, pp. 207–09. 2s.

BERYLLIUM.—Production and uses of beryllium and its alloys. Graham Oldham.—*Min. J.*, Lond., Vol. 227, July 6, 1946, pp. 531–32. 8d.

MAGNESIUM — METALLURGY. — Anglo-American magnesium production. P. L. Teed. —*Bull. Instn. Min. Metall.*, Lond., No. 479, July, 1946, 23 p. 1s. 9d.

METALLURGY AND CYANIDING.

CYANIDING — MERRILL CROWE PROCESS.—Precipitation of gold by the Merrill-Crowe process. J. G. Vivian.—*Min. Mag.*, Lond., Vol. 75, July, 1946, pp. 26–9. 1s.

CYANIDING — ONTARIO — PORCUPINE. —Naybob 1945 Gold Mines Limited. W. J. Stain.—*Deco Trefoil*, Denver, Vol. 10, April, 1946, pp. 3–6.

INDEX OF RECENT ARTICLES—*continued.*

INDEX OF RECENT ARTICLES—*continued.*

INDEX OF RECENT ARTICLES—*continued.*

MINING (General)—*continued.*

PROSPECTING — GEOPHYSICAL PROSPECTING — GRAVIMETRIC METHOD — GT. BRITAIN.—Gravimeter test in a British oilfield.—*Min. Mag.*, Lond., Vol. 75, July, 1946, pp. 21–6. 1s.

PROSPECTING — GEOPHYSICAL PROSPECTING — HOLLAND. — Geological and geophysical studies during the war on behalf of the associated collieries of the Netherlands. L. U. de Sitter.—*Nature*, Lond., Vol. 158, July 13, 1946, pp. 48–50. 1s. 6d.

PROSPECTING — GEOPHYSICAL PROSPECTING — MAGNETIC METHODS. —A new aid to geophysics. Alvin W. Knoerr.—*Engng. Min. J.*, N.Y., Vol. 147, June, 1946, pp. 71–5. 50 cents.

PROSPECTING — MAGNETIC METHODS. —The airborne magnetometer.—*Min. Congr. J.*, Wash., D.C., Vol. 32, July, 1946, pp. 48–50. 20 cents.

PROSPECTING — RARE ELEMENTS. — Rare element prospecting in pegmatites. T. T. Quirke and H. E. Kremers.—*Econ. Geol.*, Lancaster, Pa., Vol. 38, May, 1943, pp. 173–87. $1.

PUMPING.—Practical aspects of pumping sands, slurries and slimes. William B. Stephenson.—(*A.I.M.E. Tech. Pub.* No. 2013) *Min. Tech.*, N.Y., Vol. 10, May, 1946, 7 p. $1.35.

SIGNALLING — HAULAGE. — Haulage signalling. W. Saville.—*Min. Elect. Mech. Engr.*, Manchr., Vol. 27, July, 1946, pp. 21–9. 2s. 6d.

SUPPORT.—Rock pressure and roof support: Pt. 5. Z. S. Beyl.—*Colliery Engng.*, Lond., Vol. 23, Aug., 1946, pp. 213–16. 1s.

SUPPORT — STEEL — U.S.A. — Safe support provided by hitching and steel timbers. J. V. McKenna.—*Coal Age*, Albany, N.Y., Vol. 51, July, 1946, pp. 78–82. 35 cents.

SUPPORT — SUBSIDENCE. — Ground movement adjacent to a caving block in the Climax molybdenum mine. John W. Vanderwilt.—(*A.I.M.E. Tech. Pub.* No. 2000) *Min. Tech.*, N.Y., Vol. 10, May, 1946, 10 p. $1.35.

SUPPORT — SUBSIDENCE — ROCK BURSTS.—Rockburst incidence, research and control measures. W. T. Robson.—*Trans. Canad. Inst. Min. Metall.*, Vol. 49, 1946, pp. 347–74. (*Canad. Min. Metall. Bull.*, Montreal, No. 411, July, 1946. $1.)

SURVEYING — AERIAL. — Aerial photographs speed reconnaissance work. John Edwin.—*Engng. Min. J.*, N.Y., Vol. 147, June, 1946, pp. 88–9. 50 cents.

TUNNELLING — GROUND CONSOLIDATION.—Ground consolidation in railway works. H. J. B. Harding and R. Glossop.—*S. Afr. Engng.*, Lond., Vol. 57, Aug., 1946, pp. 180–81. 1s.

MINING (General)—*continued.*

TUNNELLING — PRESSURE. — Pressure on the lining of circular tunnels in plastic soils. (Tunnels of between 13 ft. and 31 ft. diameter.) D. P. Krynine.—*Proc. Amer. Soc. Civ. Engrs.*, Vol. 72, No. 2, Pt. 3, Feb., 1946, pp. 1048–65.

TUNNELLING — WASHINGTON. — Rock tunnelling. Tunnel driving in the Roslyn Cle Elum Field in central Washington. (8 ft. × 14 ft. tunnels.) Thomas Murphy.—*Min. Congr. J.*, Wash., D.C., Vol. 32, July, 1946, pp. 22–5. 20 cents.

VENTILATION — AIR CONDITIONING. —Mining and equipment problems. Pt. 4. Ventilation and air conditioning.—*Gold Mining Record*, Lond., Vol. 15, July, 1946, pp. 176–77. 1s.

VENTILATION—DUST—AIR SAMPLING. —Micro and semi-micro methods for the investigation of airborne mine dusts. N. M. Potter.—*Colliery Guard.*, Lond., Vol. 173, 1946, Aug. 2, pp. 145–49; Aug. 9, pp. 179–84. 10d. ea.

VENTILATION — DUST — PARTICLE SIZE.—Particle size by spectral transmission. Emerson D. Bailey.—*Industr. Engng. Chem., Analyt. Ed.*, Vol. 18, June, 1946, pp. 365–70. 50 cents.

VENTILATION — DUST — PARTICLE SIZE — SPECIFIC SURFACE. — Rapid method for determining specific surface of fine particles. Alphonse Pechukas and F. W. Gage.—*Industr. Engng. Chem., Analyt. Ed.*, Vol. 18, June, 1946, pp. 370–73. 50 cents.

VENTILATION — FANS. — Les ventilateurs. R. Chappuis.—*Chim. et Industr.*, Paris, Vol. 55, May, 1946, pp. 344–54.

VENTILATION—FANS—AXIAL FANS.—Contra-rotation axial flow fans. F. C. Lynam and S. P. Hawes.—*Engineer*, Lond., Vol. 182, July 5, 1946, pp. 15–17. 1s.

MINOR METALS.

BARIUM — BARYTES — ARKANSAS. — Strip mining of barytes at Magnet Cove involves problems peculiar to the climatic conditions of the Arkansas region. O. J. Benston.—*Min. Congr. J.*, Wash., D.C., Vol. 32, June, 1946, pp. 26–28. 20 cents.

CHROMIUM — BRAZIL — BAIA.— Layered or banded chromite at Compo Formoso, Baia, Brazil. W. D. Johnston, Jr., and H. Capper A. de Souza.—*Econ. Geol.*, Lancaster, Pa., Vol. 38, June–July, 1943, pp. 287–97. $1.

CHROMIUM—CHROMITE—ORE DRESSING — BENEFICIATION. — Beneficiation of Montana chromite concentrates by roasting and leaching. R. R. Lloyd, O. C. Garst, W. T. Rawles, J. Schlocker, E. P. Dowding, W. M. Mahan and C. H. Fuchsman.—*U.S. Bur. Min. Rep. Invest.* No. 3834, Wash., D.C., Feb., 1946, 37 p.

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Subject to revision] [*A Paper issued on September* **12th**, **1946**.

Billingham Mine.

By G. Eland Stewart, *Associate*.

Introduction

The following brief description of the Billingham mine was prepared in connection with a visit to the mine by members of the Institution.

The mining methods now employed at Billingham have remained fundamentally unchanged since the inception of scraper loading in 1931. Through the years up to 1939 a steady improvement in details was maintained, so that the position at the outbreak of war was that a satisfactory output per man-shift underground was being achieved and a strong case could not have been made for the heavy capital expenditure necessary to re-equip the mine with modern American stone-mining machinery.

The shortage of suitable labour and the rising costs of production now indicate the necessity for a thorough scheme of modernization at Billingham, but this scheme, when carried to completion, will make another and quite a long story, one not within the scope of the present paper.

Historical

It is well known that one of the important sources of German strength in the 1914 war was that the first plant for the manufacture of synthetic ammonia (from which nitric acid for explosives can be made) was put into operation early in 1914. In view of the British command of the sea our explosive industry could rely on the importation of nitrate of soda from Chile for this purpose.

As time went on the Government became more and more desirous that the country should be independent of overseas supplies. A group of chemists was set to work in 1916 in University College, London, and made great progress with the research and planning of a factory. A site was purchased at Billingham, but, owing to the increasing pressure on industrial facilities generally, little progress could be made before the war finished.

As a peace-time venture a large synthetic ammonia works must form the basis of a fertilizer industry and it was for this purpose that Messrs. Brunner, Mond & Co. took over the project in 1920.

A subsidiary company—Synthetic Ammonia and Nitrates, Ltd.—was formed to develop the process and site, and along with the rest of the Brunner, Mond Group it became, in 1926, one of the founding companies of Imperial Chemical Industries.

Anhydrite

Nitrogen from the air forms a raw material for the manufacture of fertilizers, as well as for explosives, and is fixed in the form of ammonia by a high temperature and pressure process. Very briefly this consists of circulating practically pure nitrogen and hydrogen at 500°C. and at 3,700 lb. per sq. in. pressure in the presence of a catalyst, when a combination of the gases forms ammonia. The fertilizer (ammonium sulphate) is produced as follows:—25 per cent ammonia solution is allowed to react with carbon dioxide, forming ammonium carbonate, which in turn reacts with ground anhydrite, producing ammonium sulphate (in solution) and calcium carbonate (chalk, as solid). The ammonium sulphate solution is concentrated to form crystals, while the chalk is used for making cement and another fertilizer—called 'Nitro-Chalk'. Sulphate of ammonia is a very stable and safe form of nitrogen storage; it is in great demand now for the starved agricultural land of Europe, which is the main reason for the increased outputs required.

By a process of roasting in kilns with other materials anhydrite is made to give off the gas sulphur dioxide, which is oxidized in the presence of a catalyst to produce sulphuric acid. The clinker produced is used for the manufacture of cement. Ground anhydrite is used with certain accelerators or retarders to produce various grades of building plasters, plaster blocks, etc. Ground anhydrite is used direct for the control of weevil in grain, as an insecticide, as a filler in paint, and has numerous other uses in small quantities.

Geology

The anhydrite deposit worked at Billingham occurs near the middle of the South Durham and North Yorkshire Permo-Triassic succession. The presence of rock salt and anhydrite among the rocks concealed beneath the surface glacial deposits was known for many years from the brine wells near Middlesbrough. In 1926 a boring at Billingham, put down in search of a supply of cooling water, proved beneath the rock salt, at a depth of 800 ft., a bed of anhydrite 20 ft. thick and of 90 per cent $CaSO_4$ content.

The anhydrite as mined is a blue-grey stone, translucent when *fractured* to a thin edge. It has a specific gravity of 2·9 and a

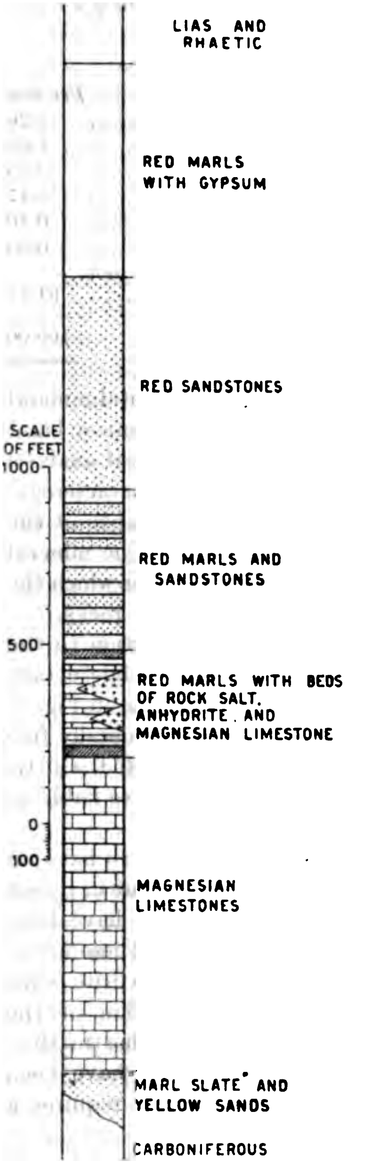

FIG. 1.—Generalized section of the Permo-Triassic rocks in Durham.

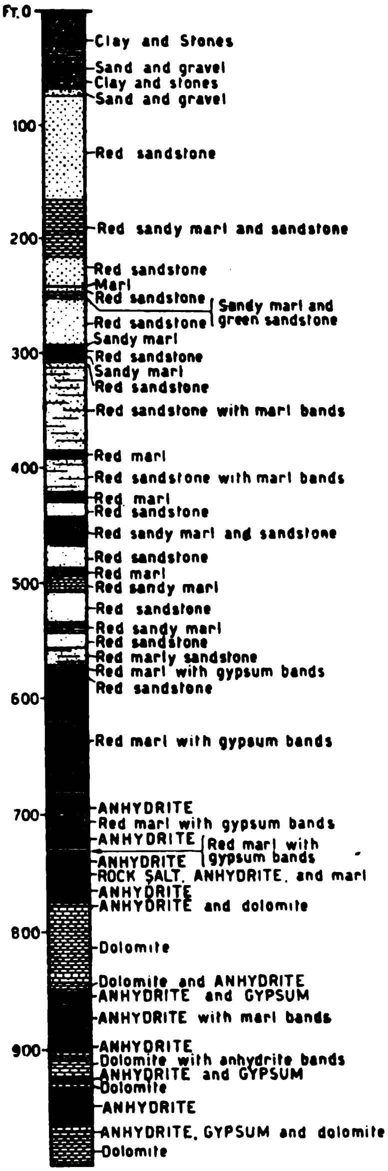

FIG. 2.—Stratigraphical succession compiled from No. 2 shaft and borehole 'B'.

hardness of 3 to 3·5 on Moh's scale. The bulk analysis of the mine output averaged over a year is: $CaSO_4$ 90·5 per cent, SiO_2 2·1 per cent, and NaCl 0·006 per cent.

TABLE I

	Per cent
Muscovite	3·26
Biotite	1·63
Paragonite	0·35
Quartz	0·17
Hornblende	0·10
Fluorite	0·05
Anhydrite, with water-soluble minerals and traces of others not exceeding 0·01 per cent each	94·44
	100·00

The anhydrite contains a large variety of minute detrital mineral grains. A quantitative analysis of a sample taken from a band conveyor over a period of 16 days gave the mineralogical analysis shown in Table I. The quartz grains show the characteristic rounding of wind-blown sand, and the relative abundance of the micas is also clear evidence of wind transportation. The mineral assemblage as a whole indicates the arid conditions under which the grains were derived and transported from their parent rocks.

The position of the Billingham anhydrite bed in relation to the general succession in the district is shown in Fig. 1, while details of the strata from shaft and bore-hole sections are given in Fig. 2. The anhydrite seam is, of course, perfectly dry and generally free from gas, but when the underlying dolomite is penetrated, a little water and some methane and sulphuretted hydrogen are liable to be released.

Up to 1939 the anhydrite workings proved the seam to have an average inclination of 1 in 19 in a direction S. 47°E., but in recent years great variations in direction and amount of dip have been met with, as well as red marls intruding in the roof. These intrusions are associated in their contact with the anhydrite with slickensides and other evidence of plastic deformation of the deposits in response to tectonic stress. These and other matters relating to the geology of the Billingham anhydrite have been studied by members of the staff, but such a subject requires a paper to itself.

MINING

Shafts.—The sinking of two shafts, 18 ft. inside diameter, was commenced on September 3rd, 1926, and completed in 14 months. In sinking through the sandstones 600 gallons per minute of water were made in each shaft, which was dealt with by pumps; the shafts are lined with reinforced concrete through the water-bearing measure and with plain concrete below. Although the small diameter of the shafts is responsible for quite 95 per cent of the power input required at the ventilation fan, represented by 130,000 cu. ft. per minute at 2·3 in. W.G., the unit power costs have been so low that no case could be made for larger shafts at much increased capital cost in order to save on ventilation running costs.

After passing through the anhydrite at 778 ft. from the surface the sinking was continued 73 ft. into the magnesian limestone, or dolomite, to provide depth for the gravitation of the anhydrite rock produced through a 180-ton bunker and 4-ton measuring boxes to the shaft skips. Spillage from the skips is collected on a steel diaphragm at the shaft bottom and diverted down a chute to a standard mine tub.

Sinking through the dolomite provided the only difficulty, when gas consisting of 94 per cent methane and 3 to 4 per cent of sulphuretted hydrogen was encountered. Auxiliary fans and canvas tubing were used to sweep out the gas and arrangements made for testing the conditions at the shaft bottom every hour. It should be noted in passing that canaries were useless, as they dashed themselves to pieces with fright at the noise of the sinkers' drills, while guinea pigs continued to 'eat their heads off' in gas concentrations above the danger limit for men doing arduous work—i.e., one part H_2S in 10,000. Actually, the sinkers were withdrawn when the sulphuretted hydrogen reached a concentration of 0·009 per cent. The men were found to vary considerably in their resistance to the gas and some continued to have sore eyes and throats when working to this limit. The present method of detecting sulphuretted hydrogen in the mine is as described in the Department of Scientific and Industrial Research publication 'Methods for the Detection of Gases in Industry', Leaflet No. 1, Hydrogen Sulphide (Revised Edition) 1943.

Skip Winding.—The installation of a 850 h.p. 3,300-volt straight a.c. induction motor skip winder at No. 1 downcast shaft, with skips holding 4½ tons of anhydrite, allowed of the originally-projected output of 1,000 tons per shift being easily obtained from the small shafts and with three shifts working an output of 18,000 tons per week. Actually it has been proved that a maximum winding capacity of 1,500 tons per 8-hour shift is possible when there are no chokes or other causes of delay. During an early period of the war, when the mine was operating a 9-hour shift, outputs in excess of 1,600 tons per shift were obtained.

The winder, built by Scott and Hodgson, has a 9-ft. parallel drum lagged with oak six inches thick, giving a rope speed of 980 ft. per minute. The 350 h.p. motor drives through two trains of double helical gears. Allen-West liquid controllers give acceleration to full speed in ten seconds, with a peak horse-power of 500. Since installation, an extra bearing has been provided for the first motion shaft and double-tangent keys substituted in the intermediate shaft for the original sunk keys. The double tangent keys have proved very efficient. The overwinder is of Burroughs design and recently proved completely efficient when an accidental overwind occurred.

The cage winder at No. 2 up-cast shaft is similar to No. 1 skip winder, except that it is built to the opposite hand : 1¼-in. diameter locked-coil ropes are used at both shafts for periods up to five years and are capped with ' Reliance ' capels.

Crushing Plant

The run-of-mine anhydrite, up to 18-in. cube, is tipped into a head-frame bunker of 100-ton capacity, the headframe being 94 ft. high to the centre of the 18-ft. diameter pulleys. From the bunker the stone, controlled by Ross chain feeders, passes through Hadfields size 7½ gyratory crushers reducing to 3-in. diameter. Passing rotary screens the 1½-in. ring size passes to a 3,000-ton storage bunker, while the oversize is reduced in Symons disc crushers to one inch cube, which is then passed to the storage bunker. Dust control within the plant is satisfactorily maintained by a Visco-Beth apparatus.

The handling of the inch cube anhydrite from the 3,000-ton storage bunker to the grinding mills of the ammonium sulphate plant is by bi-cable ropeway, and from the crushing plant to the sulphuric acid plant by band conveyor.

Underground Layout

The mine workings (Fig. 8, Plate I) are bisected by a locomotive haulage road 18 ft. wide, called the West level, which was set out in 1928 at a slight angle to the strike of the seam to give a grade of 1 in 80 in favour of the loaded tubs. It is now known that a grade of 1 in 200 would have been better, since the number of empty tubs hauled inbye is limited by the load on the locomotive, while the number and speed of the trains of full tubs brought outbye is limited by the braking capacity of the locomotive, especially when the rails are wet due to condensation in the hot summer months. The track is 30-in. gauge and laid with 40 lb. per yard rails for the two 6-ton English Electric trolley locomotives. From this West Level rope haulage inclines, 18 ft. wide and 9 ft. high, to rise and dip are set off every 120 yards. The dip haulages were from the first built with frames to carry two drums, although for the first nine years they were fitted with single drums only and operated as direct-rope haulages. When the dip workings passed through an area of changing dip, forming a ' swilly ', it was possible to add a second drum to the dip haulages and convert them into ' main and tail ' units. These haulages are 150 h.p., to give a rope pull of four tons at 6 m.p.h., which on average gradients enables a 25-ton pay load to be hauled. The rise haulages, having to raise empty tubs only, are 60 h.p., but since the duty of pulling sets of 20 full tubs of 33 cwt. gross each round curves from the loader levels into the haulage plane is severe these haulages are fitted with a two-speed gear. On the bottom gear the speed is 2 m.p.h., enabling the 60 h.p. motor to deal adequately with this duty. Since the lowering of 20 full tubs with a gross weight of 33 tons down the incline at 6 m.p.h. is severe on the brakes, these rise haulages are fitted with two brake paths to the drum and double-caliper brakes.

It may be pointed out that, since the rope tension due to hauling at 2 m.p.h. and to braking on the rise haulages is the same as the rope tension due to hauling on the dip haulages, the drum shafts were specified to be the same diameter. The necessity for this was apparently not obvious to some haulage manufacturers, who questioned the necessity of putting as big a drum shaft into a 60 h.p. haulage as into a 150 h.p. haulage.

From these haulage planes levels are turned off at 20-yard centres to form loading levels 18 ft. wide and the full height of the seam, which on the average has been 18 ft.; extreme heights of 27 ft. have been worked. To rise and dip off these loader levels cross-cuts are driven 18 ft. wide and 9 ft. high in the first working, at 20-yard centres, forming pillars 42 ft. square, thus giving an extraction of 51 per cent in plan area. These cross-cuts are later stoped in two lifts to the full height of the seam.

In the loader levels a top heading is driven 8 to 9 ft. high and 3 yards or more in advance of the bench.

Drilling

Drilling.—Drilling on the headings is generally by Holman S.L. 10A machines mounted on 3-in. vertical columns, the slides being arranged for 4-ft. steel changes. Some CP. 50 auto-feed drifters are in use and Holman auto-feed drifters are on order; 25 holes are drilled in the heading, using a V-cut, as shown in the drawing (Fig. 4). The bench is drilled with similar machines mounted on a 4½ in. horizontal bar; the correctly drilled round should pull 7 ft.

The cross-cuts are driven using the 8-in. vertical columns, but a shorter round is drilled to pull 5 ft.

Much trouble was formerly experienced with bent jack screws when using standard drill columns, so special columns with an external barrel nut are now made, with buttress thread for jacking, which cured the trouble.

All drilling has to be dry, because anhydrite dust has similar characteristics to plaster of Paris; any moist anhydrite would set in tubs and bunkers, and with 0·5 per cent moisture it will not grind in the ball-mills. This means that much dust is produced in drilling, but the systematic X-ray of miners over the last 16 years has proved the dust to be innocuous. Through the years much research and experimentation has been carried out on dust-trapping devices without any marked permanent success, the most successful being the Pyrene dust trap using merino-wool filters similar to the material used in the Mark IV respirator. Some work has been done with electric and compressed air rotary drills using tungsten carbide bits, and this work is being continued with two American coal drills, the CP. 700 and the Jeffrey A6.

Respirators are available to the drillers, the present issue being the American 'Dustfoe', but only about 10 per cent of the men will wear them.

The cross-cuts are drilled with Holman Golden Arrow auto-twist stopers.

Hollow and solid 1-in. hexagon alloy drill steels are used to drill the 6,000 to 7,000 ft. daily, consumption being one third of that when plain carbon steel was used. Rose bits and 4¼-in. shanks are standard, each bit drilling 140 ft. between sharpening. No success has been achieved with detachable bits.

Blasting.—Blasting of rounds of up to 50 holes is normal, using on the average 2½ lb. of Polar Ammon. Gelignite per hole in the heading and the old type of delay action detonators, up to No. 4 delay. In dry conditions there is no reason to use the more expensive gasless delay detonator. The arrangement of delays is shown in the sketch (Fig. 4).

Exploders of various types are in use—Atlas 50-shot, Dreadnought 50-shot, and Schaffler 100-shot; the Schaffler 100-shot is the best machine, giving the most rapid rise of voltage independent

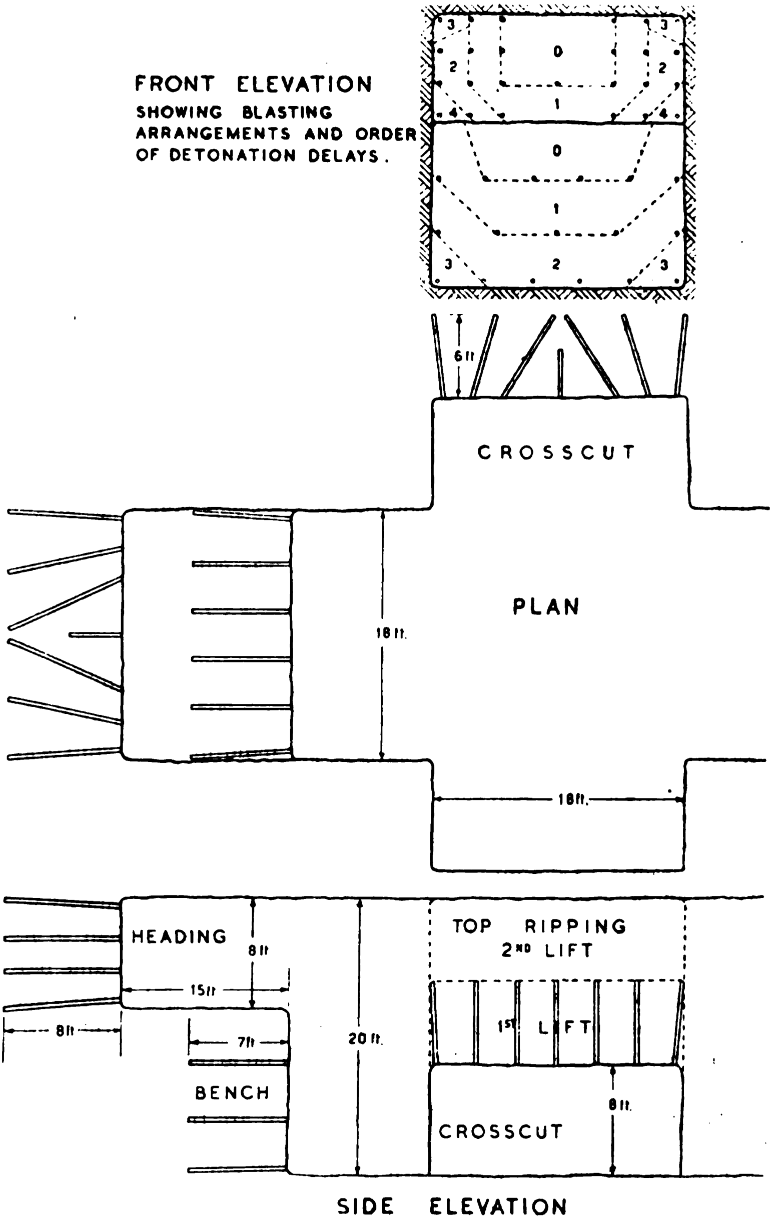
FRONT ELEVATION
SHOWING BLASTING ARRANGEMENTS AND ORDER OF DETONATION DELAYS.
0
1
2
3
4
6 ft
CROSSCUT
PLAN
18 ft.
18 ft.
HEADING
8 ft
15 ft
8 ft
7 ft
BENCH
20 ft.
TOP RIPPING
2ND LIFT
1ST LIFT
CROSSCUT
8 ft.
SIDE ELEVATION

Fig. 4.

of the operator and is used where more than 50 shots in a round are necessary. Plain clay is used for stemming, the holes being stemmed to the mouth. The stemming is produced in an extrusion machine at the surface and conveyed underground in special clay bogies. Much of the 15 per cent moisture required in the clay during the extrusion process is lost in transit underground before being used.

Exhaustive tests have been carried out with sand-clay stemming, proving that for the conditions obtaining nothing was to be gained by this complication, which caused excessive wear of the extrusion machine.

To prove conclusively that stemming deep holes to the mouth with clay and using high explosive was giving practically 100 per cent efficiency, the author carried out a series of tests in which holes were stemmed with Ciment Fondu and allowed to set for 24 hours. In these conditions the explosive was to all intents and purposes sealed within solid rock, indicated by the fact that blocks of anhydrite were found after blasting with the cement stemming still adhering to the rock; whereas the normal consumption of explosive is at the rate of 0·6 lb. per ton, cement stemming improved this figure to 0·59 lb. per ton only, which was within the limits of experimental error.

Loading

The method of loading up to 1931 was by hand shovelling into the 25 cwt. capacity tubs, which limited the size of tub and rail gauge used. By this method 12½ tons per hand filler per shift was maintained.

Trials were then made with Holman and Sullivan scraper loaders with partial success, but the rock-loading duty proved too arduous for the standard design of scraper loader and the mine was forced to build loaders to a modified design to obtain the robustness required. Whereas, on the purchased loaders, clutches had to be relined every 10 to 12 shifts, main shafts were sheared, and ball bearings were spilled into the epicyclic gearing with disastrous results, the loaders built at Billingham have, during the war, operated for periods up to three years with no attention other than oiling.

The main points of the Billingham design are the use of an air-cooled nitrided clutch race of ample area and diameter, rigid caliper clutch shoes, heavy full-type roller bearings, alloy steel shafts, rigid fabricated steel bedplate, and very ample oil reservoirs feeding the roller bearings. The bearing seats are ground to fine interference limits and the bearings heated to 70°C. in oil and shrunk on the shafts. The drive is by roller chain from a 30 h.p. electric motor: ⅝ in. diameter flattened-strand ropes with wire core were standard before the war to pull a 44 in. skip weighing ⅝ ton and hauling ¾ ton of stone. The 8 in. diameter face snatch blocks are built of solid alloy steel, the pulley being mounted on a plain bronze grease-lubricated bush, after experimenting with many designs using roller bearings; when a roller bearing of ample size was used the snatch block was too heavy for practical use.

For the five years before the war these scraper loaders averaged 205 tons per shift of 6½ to 7 hours working time, but with present labour the ouput varies from 140 to 160 tons per shift. The loaders are operated by four men and a chargehand, there being one driver, one tugger man, and two men changing tubs.

Sullivan 7½ h.p. Turbinair tuggers are found most useful for handling tubs in the loader levels, although when grades become steep the Holman 14 h.p. Vee-Haulage becomes necessary, and has been found most reliable.

Ventilation

Ventilation is by a 18 ft. 6 in. Walker Indestructible fan placed underground, exhausting 180,000 cu. ft. per minute to the surface. The water gauge on the fan is 2.8 in., but 2·2 of this is due to the 18 ft. diameter shafts and only 0·1 in. is required to circulate the air through the roads of such large areas in the mine.

These levels of large area present a great difficulty in preventing leakage and expansion into the old workings and brattice cloths have to be erected outbye of the haulage inclines to keep the circulating air in the face area. Now that the working face is approaching a mile from the shafts less than half the total quantity of air reaches the main split at the end of the West Level. This situation stresses the advantage of a panel system of working where it can be adopted, but the present method of driving loader levels continuously prevents the formation of panels.

A commencement has been made to build a brick stopping across the mine near to the present working face, to cut off all the old workings, and in effect to form a new mine. Plans are in hand, however, for a panel system of mining in a new area, to be operated on trackless mining methods, when all the existing methods and equipment will be made obsolete—but this will be the subject of another paper in the years to come.

*** *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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An Experimental Investigation of the Effects of High Temperatures on the Efficiency of Workers in Deep Mines.

By Anthony Caplan, M.D. (Lond.), M.R.C.P. (Lond.), and J. K. Lindsay, *Member*.

Introduction

During the past few decades workers in many parts of the world have carried out investigations on the important subject of the effect of hot humid surroundings on the health and efficiency of industrial workers. There still exists, however, some divergence of opinion as to what constitutes comfortable or uncomfortable conditions and what is the best index as a unit of comparison.

Climatic conditions, racial characteristics, and acclimatization all play an important part in the problem. Thus conditions which would be oppressive to miners in a temperate climate need not necessarily be oppressive to men working in the tropics; this especially applies to the indigenous population.

On the Kolar goldfield, in common with other deep-mining centres, the problem of hot humid conditions underground constituted a serious threat to the continued efficient working of the mines at depth, but the installation of air-cooling plants on surface has so materially improved conditions at vertical depths of 8,000 to 9,000 ft. below the surface that operations can now be carried on without undue effect on the health or efficiency of the workers. The success of a cooling plant, however, is largely dependent on an efficient distribution of the cooled air to each working point and as the workings proceed in depth a general but gradual and progressive deterioration in conditions will again be experienced. In addition it may be pointed out that at the present time the mining of ore at depth and under temperature conditions comparable with those on the Kolar goldfield is confined to a few mining localities only. The time is approaching, however, when shallow deposits will become exhausted or present orebodies followed down to greater depths. The problem of working efficiency at high temperatures will then become of increasing economic importance to many mining fields throughout the world.

For these reasons, investigations were carried out to determine a satisfactory index which could be adopted as a measurement of comfort and efficiency for workers in deep mines and the point at which a serious falling off in efficiency might be expected.

Description of Experimental Procedure

Selection of Method.—The selection of a method by which the efficiency of the worker could be determined presented some difficulty. No ergometer was available to measure directly the foot-pounds of work; in any case such an instrument was not thought to be suitable for experimental purposes with the Indian labour available. Several alternatives were then reviewed by which it would be possible to calculate the foot-pounds of work, but as the investigation was primarily concerned with the relative working efficiency under varying atmospheric conditions, it was considered that a knowledge of work done in terms of foot-pounds was not essential. It was also felt that if men performed work they were accustomed to in their daily routine more reliable results would be obtained; hand-drilling was therefore decided upon as a simple and straightforward method by which work done could be measured and compared in terms of inches drilled per hour.

It was important that apart from the personal factor the only variables should be atmospheric conditions and not the drilling medium. The hornblende schist, which forms the wall rock of the quartz veins, varies considerably in hardness and is generally softer and easier to drill at depth. The local granite, however, is relatively consistent in hardness and texture and any slight variation is of little practical consequence when using a large number of blocks. Granite was therefore selected as the drilling medium.

Selection of Men.—Six men of average physical fitness daily engaged in the drilling of rock by hand and acclimatized to hot and humid conditions up to about 93°F. wet-bulb temperature were selected. No particular care was taken to pick men of exceptional physique or ability to withstand high temperatures. They were chosen from the ordinary run of mine workmen and for their record of regular attendance at work, as they would then be less liable to upset the investigation routine by absenteeism.

Method of Conducting Tests.—As a preliminary to carrying out

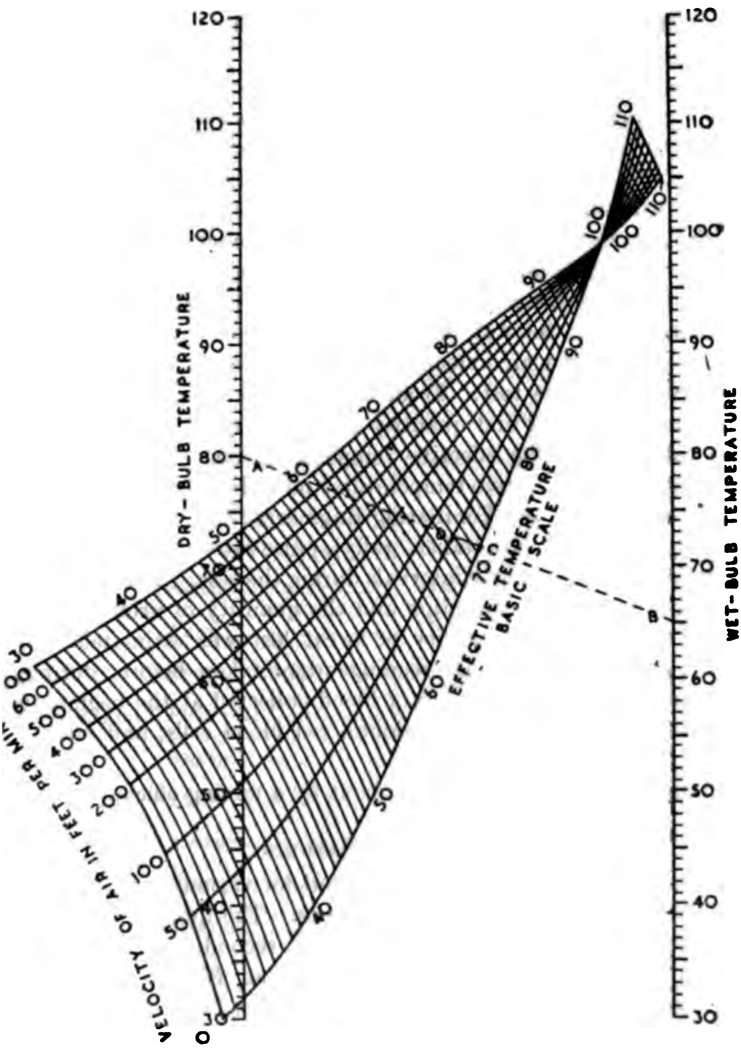

FIG. 1.—Chart giving the basic scale of effective temperatures.
[Reproduced by kind permission of the American Society of Heating and Ventilating Engineers].

the experiments, a temperature survey was made of the downcast and upcast air currents, from which convenient working points were selected to give a range of wet-bulb temperatures between 70°F. and 97°F. and a dry-bulb range of approximately 80°F. to 125°F. The tests were commenced at the bottom of the downcast airway and then continued at any of the pre-selected points in either the downcast or upcast airway as was at the time convenient.

The granite blocks selected were of a size suitable for transporting to the various levels. The men drilled in pairs, one man holding the drill and the other using the hammer. The men worked without clothing and no attempt was made to influence the drilling speed, division of labour, or number of rest periods taken. The duration of each test was three hours and at the end of each hour the number of inches drilled was measured.

Recordings of the wet- and dry-bulb temperatures, wet kata cooling power, and air velocity were taken before and at hourly intervals during the test. From these figures grains of moisture per pound of dry air and quantity of air in cubic feet per minute were calculated. The effective temperature was computed from the basic chart of effective temperatures published by the American Society of Heating and Ventilating Engineers (Fig. 1).

The body temperature (taken orally), pulse rate, and weight were taken before and at hourly intervals during the test. Records were also kept of water consumption and the quantity of urine passed. From these data the quantity of sweat lost was estimated. In certain cases blood-pressure readings were taken before, during, and after the test and samples of blood collected at the beginning and end of the test for haemoglobin, red blood cells, and plasma chloride estimations.

All tests were supervised throughout by a mining official.

Analysis of Results

A total of 80 tests was carried out, but for various reasons complete data were available in only 71 tests. The number of inches drilled by the three pairs during the first, second, and third hour in each test was averaged and in addition an average was taken of the total number of inches drilled per hour by the three pairs for the three-hour period. This is illustrated in Table I.

The average number of inches drilled for each of these periods (underlined in Table I) in each test was then plotted against the wet-bulb temperature, effective temperature, and wet kata cooling power (Figs. 2 to 4). Curves were then drawn by an independent *and expert* mining engineer to represent the mean of the findings in each group of tests. It can be noted that difficulty was experienced in drawing a satisfactory curve for the wet kata readings. The curves for wet-bulb temperature, effective temperature, and wet kata cooling power for the first hour, second hour, third hour, and total three-hour shift were then charted on one graph for comparison (Fig. 5).

TABLE I

SPECIMEN TESTS

		Test Number			
		52	*62*	*18*	*16*
Dry bulb, °F.		82·7	106·1	121·1	121·3
Wet bulb, °F.		82·2	86·2	93·1	94·9
Effective temperature		79·5	90·6	97·2	98·7
Wet kata cooling power, millic./sec.		12·36	7·35	7·67	5·12
Inches drilled					
First hour:	First pair	12·75	11·25	8·0	6·25
	Second „	11·5	10·25	6·25	4·25
	Third „	11·25	6·25	7·00	4·75
	Average	11·83	9·25	7·08	5·08
Second hour:	First pair	9·75	9·75	8·75	5·0
	Second „	13·5	11·25	9·75	3·25
	Third „	12·25	6·00	9·00	2·25
	Average	11·83	9·00	9·17	3·5
Third hour:	First pair	10·25	8·75	8·00	Men unable to work
	Second „	10·75	7·5	7·75	
	Third „	10·25	7·0	7·25	
	Average	10·42	7·75	7·67	0
Average of three pairs for three hours		11·36	8·67	7·97	2·86

At the beginning of the investigation no decision was reached upon a standard of 100 per cent efficiency. With this object in view a number of tests was carried out in a main haulage level where conditions were extremely comfortable (dry bulb, 85° to 86°F., wet bulb, 73° to 75°F., effective temperature, 65° to 69°, and wet kata cooling power, 85 to 32). As will be seen from the graphs it was found that the work done here was less efficient than in many other places with higher wet-bulb and effective temperatures and lower wet kata cooling power. These tests were excluded from the mean curve and their significance is referred to later. When the

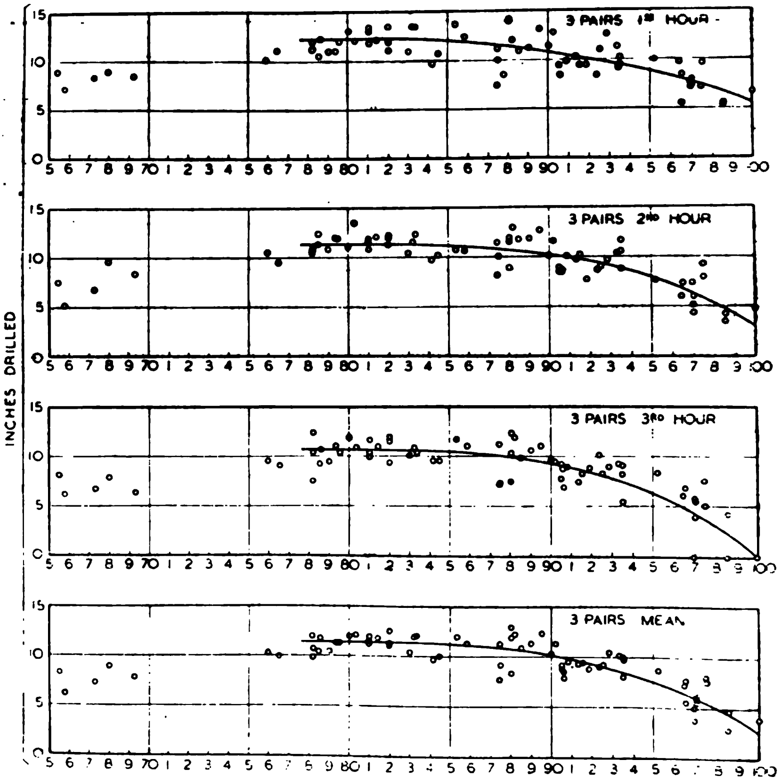

FIG. 2.—The relationship between work done and wet-bulb temperature °F.

graphs were drawn it was decided to use as the 100 per cent efficiency line the average of work done under the most favourable underground conditions during the first hour. The amount of work done in inches drilled was then converted into percentages of efficiency.

The relationship between working efficiency and the wet-bulb temperature, effective temperature, and wet kata cooling power is shown in tabular form in Table II and graphically in Fig. 5.

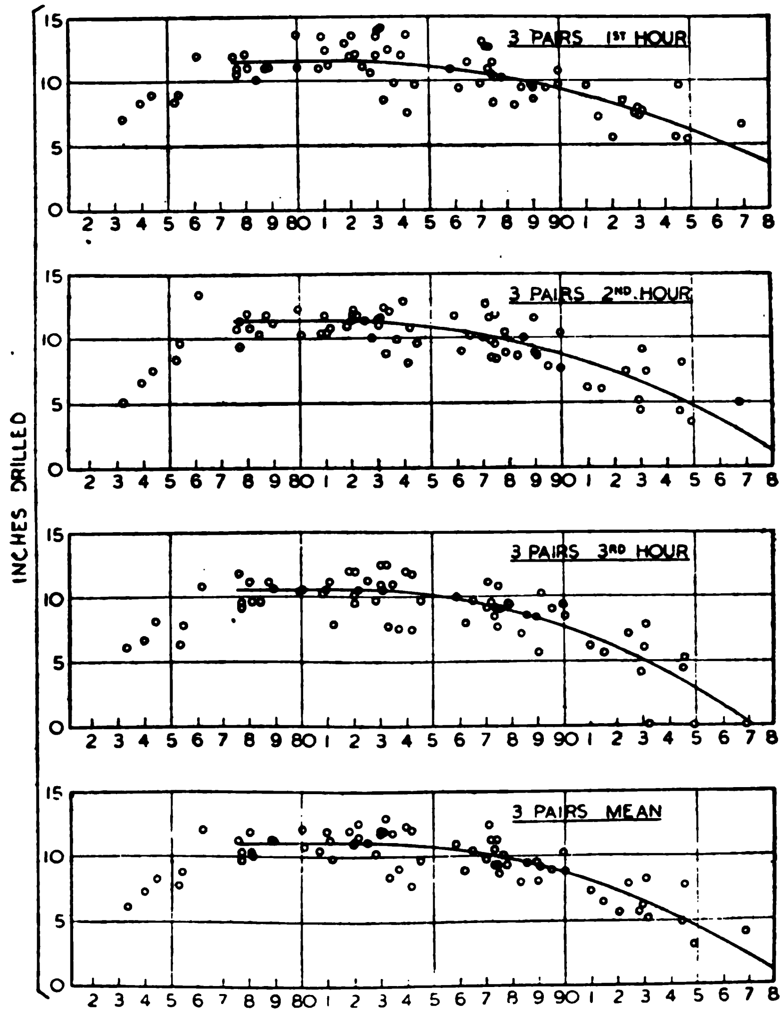

Fig. 3.—The relationship between work done and effective temperature.

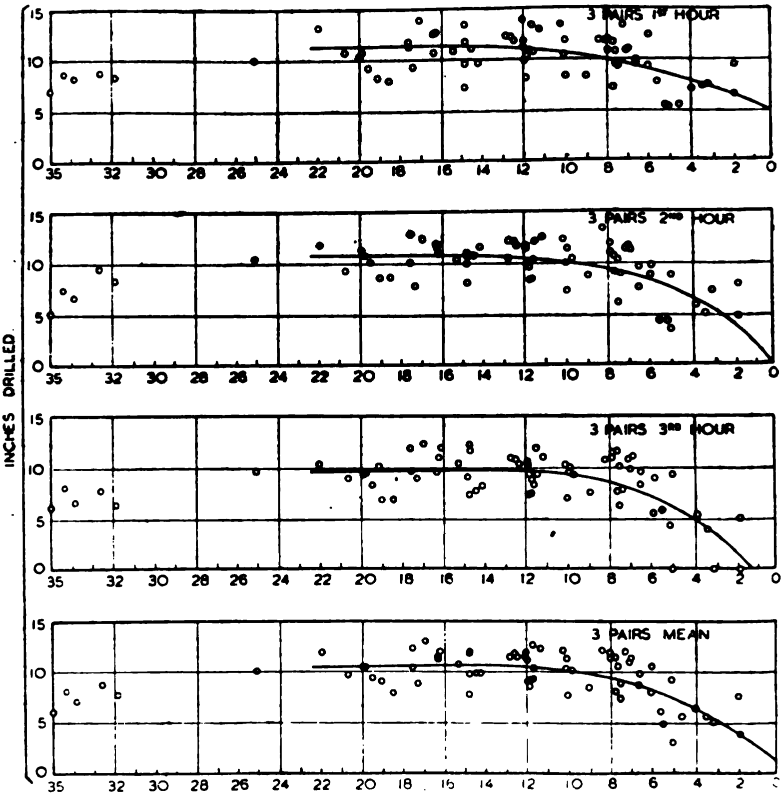

FIG. 4.—The relationship between work done and wet kata cooling power.

Examination of the working efficiency curves shows the following:

TABLE 11

THE RELATIONSHIP BETWEEN WORKING EFFICIENCY AND WET-BULB TEMPERATURE, EFFECTIVE TEMPERATURE AND WET KATA COOLING POWER

Efficiency, Per cent	First Hour			Second Hour			Third Hour			Mean Three Hours		
	Wet-Bulb Temp., °F.	Effective Temp., °	Wet Kata Millic./Sec.	Wet-Bulb Temp., °F.	Effective Temp., °	Wet Kata Millic./Sec.	Wet-Bulb Temp., °F.	Effective Temp., °	Wet Kata Millic./Sec.	Wet-Bulb Temp., °F.	Effective Temp., °	Wet Kata Millic./Sec.
100	83·0	84·5	14·0	—	—	—	—	—	—	—	—	—
90	87·2	89·1	9·0	85·7	87·0	10·7	83·5	—	—	85·2	86·4	11·7
80	89·5	92·2	6·6	88·2	91·2	7·7	86·6	88·1	9·5	88·0	90·5	8·2
70	91·4	94·5	4·7	90·1	93·5	6·0	88·7	91·0	7·4	90·0	92·6	6·4
60	93·0	96·5	3·1	91·6	95·0	4·8	90·1	93·0	6·0	91·5	94·5	5·1
50	94·4	98·0	1·8	93·0	96·3	3·9	91·3	94·4	5·0	92·6	95·9	4·1
40	95·6	99·4	0·5	94·0	97·4	3·0	92·5	95·5	4·2	93·7	97·1	3·4
30	96·8	—	—	95·1	98·4	2·3	93·5	96·5	3·5	94·8	98·1	2·5
20	98·0	—	—	96·0	99·2	1·8	94·3	97·4	2·9	95·6	99·0	1·8
10	—	—	—	96·9	—	1·1	95·2	98·1	2·3	96·5	99·9	1·0
0	—	—	—	97·6	—	0·7	96·0	98·8	1·9	97·2	—	0·4

Wet-Bulb Temperature Curve—Fig. 5A

First Hour.—There is no change in the efficiency until the temperature rises above 83°F. when the output falls, and at temperatures between 90° and 91°F. it is reduced to 75 per cent. The decline is then a little more rapid and with temperatures in the region of 94° to 95°F. the capacity for work is 50 per cent and at 96·8°F. only 30 per cent.

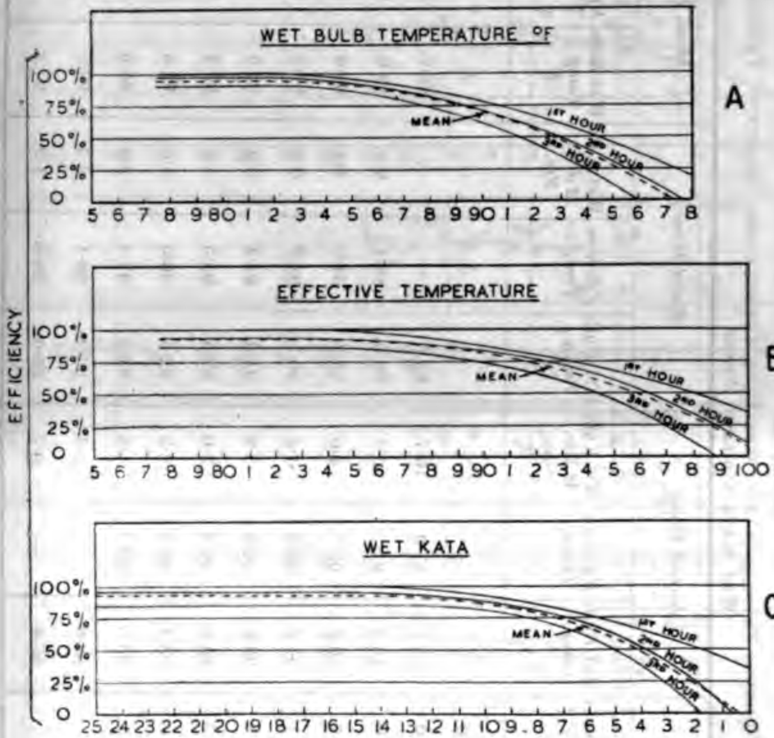

FIG. 5.—Working efficiency curves for first, second, third hour and mean.

Second Hour.—In working places with a temperature range of 78° to 83°F. the efficiency is 96 per cent of that in the first hour. A falling off takes place after 83°F., and the curve follows closely parallel to that of the first hour to a temperature of about 90°F., when the efficiency is reduced to 75 per cent. The fall is then more rapid and at 93°F. the efficiency is 50 per cent, at 96°F. 20 per cent, and at 97°F. 10 per cent.

Third Hour.—The efficiency in the third hour at the lower range of temperatures is 90 per cent, and a further fall is apparent at 83°F. The efficiency drops to 75 per cent when temperatures of 87° to 88°F. are reached. From this point the ability to perform work diminishes more rapidly than in the first and second hours, and at temperatures between 91° and 92°F. the output is only 50 per cent, and at 94°F. 25 per cent. At a temperature of 96°F. the working capacity is nil.

Three Hours.—The curve of the mean performance over the three-hour period follows very closely that of the second hour until temperatures rise above 92°F., when the fall in efficiency is a little more marked.

Effective Temperature Curve—Fig. 5B

First Hour.—The efficiency falls when temperatures reach 84° and continues to fall slowly until at 93·5° the output is reduced to 75 per cent. Beyond this point the fall is somewhat more steep and at 100° the efficiency is 35 per cent.

Second Hour.—The efficiency in the lower temperature readings is about 95 per cent of that in the first hour and a fall commences at 83°. The curve closely follows that of the first hour, with a progressive deterioration in working capacity, until at 92·4° the efficiency is 75 per cent. Above this temperature the decline is more rapid and at 96·3° the efficiency is 50 per cent and falls to 30 per cent when temperatures approach 98·4°.

Third Hour.—The efficiency is about 85 per cent up to 84° to 85°, when a fall occurs. The curve then runs closely parallel with those of the first two hours, until, at 90°, the efficiency is 75 per cent. Beyond this point there is a marked decline and the output is reduced to 25 per cent at 97°.

Three Hours.—The mean curve is similar to that of the second hour until temperatures of 90° are reached when there is a slight divergence. The curve then follows that of the second hour and at 95·9° and 98·5° the efficiency is 50 per cent and 25 per cent respectively.

Wet Kata Cooling Power Curve—Fig. 5c

First Hour.—The curve follows a straight line up to a wet kata cooling power of 13·8, when a falling off is observed. The decline is gradual until 5·7 when the efficiency is 75 per cent, and at 1·8 the efficiency is 50 per cent.

Second Hour.—The second-hour curve commences at 95 per cent efficiency and follows the first-hour curve very closely until an index of 7·5 is reached. From this point the fall is a little more rapid, giving an efficiency of 75 per cent and 50 per cent when the index is 6·9 and 3·9 respectively.

Third Hour.—The efficiency up to an index of 12·5 is 85 per cent, when a falling off occurs and below 8 the decline is greatly accelerated. At an index of 8·2 the efficiency is 75 per cent, at 5·0, 50 per cent, at 3·2, 25 per cent, and at 1·9 it is nil.

Three Hours.—The mean curve is slightly lower and almost identical with that of the second hour until an index of 3·0 is reached, when it crosses that of the second hour. At an index of 4·1 the efficiency is 50 per cent and at 2·2 it is 25 per cent.

Humidity

In Fig. 6 the grains of moisture per lb. of dry air are plotted against inches drilled. It will be seen there is no direct relationship between humidity and work done. It may be stated, however, that when high humidities were associated with high dry- and wet-bulb temperatures the working efficiency was reduced.

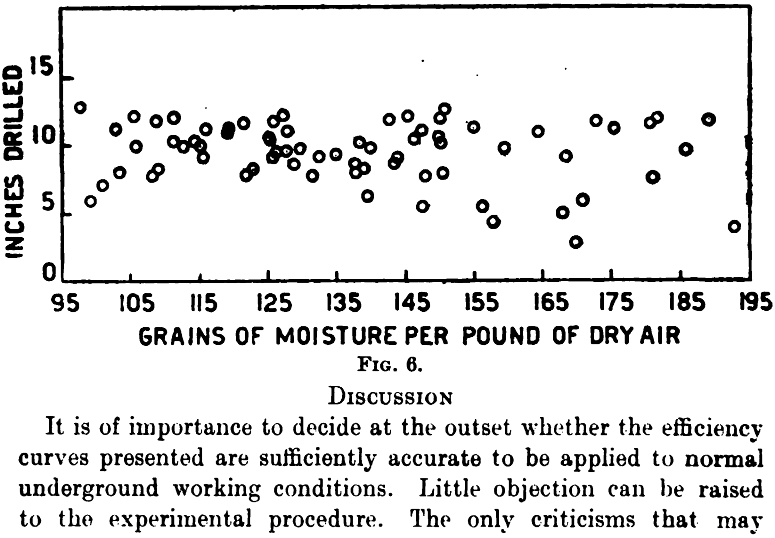

Fig. 6.

Discussion

It is of importance to decide at the outset whether the efficiency curves presented are sufficiently accurate to be applied to normal underground working conditions. Little objection can be raised to the experimental procedure. The only criticisms that may legitimately be made are : First, that the number of men engaged or the number of tests carried out (especially at certain ranges of temperature) were too few to give a fair average of work done under varying underground conditions ; and, secondly, that the men were only fully acclimatized up to wet-bulb temperatures of 93°F. The graphs show that in a number of tests the work done was either much below or above the average for a particular wet-bulb temperature, effective temperature, and wet kata reading, but this is not an uncommon finding in experiments introducing the human factor. A greater number of tests would undoubtedly have assisted in neutralizing this factor and more accurate curves obtained. Nevertheless, it is felt that a sufficient number of tests was performed to permit curves to be drawn showing for practical purposes the relationship between working efficiency and the wet-bulb temperature and effective temperature. The wet kata curve, however, is not considered satisfactory.

It is admitted that working efficiency under the most oppressive conditions would have been a little higher if the men had been acclimatized over a long period to wet-bulb temperatures above 93°F. It is considered unlikely, however, that the differences would alter the main outline of the curves to any appreciable degree.

The main facts brought out by the present investigation are :

(1) During a three-hour shift the output of work diminished hourly even under the most comfortable working conditions. This hourly decline in efficiency was increasingly manifest when the wet-bulb temperature rose above 83°F., the effective temperature above 85°, and the wet kata cooling power fell below 14. Ultimately the efficiency fell to a degree when little or no useful work could be performed in the third and even the second hour. This stage was reached when the efficiency fell below 25 per cent and was noted :

(*a*) After two hours' work when the wet-bulb temperature rose above 94°F., the effective temperature above 97°, and the wet kata fell below 3.

(*b*) After one hour when the wet-bulb temperature rose above 95·5°F., the effective temperature above 99°, and the wet kata fell below 2.

A serious falling off in efficiency (say 50 per cent) occurred, however, at considerably less oppressive conditions. This was noted :

(*a*) After two hours' work when the wet-bulb temperature rose above 91·5°F., the effective temperature above 94·5°, and the wet kata fell below 5.

(*b*) After one hour when the wet-bulb temperature rose above 93°F., the effective temperature above 96·5°, and the wet kata cooling power fell below 4.

During the first hour the output was not seriously impaired—i.e., fell below 50 per cent—until the wet-bulb temperature rose above 94·4°F., the effective temperature above 98·0°, and the wet kata fell below 1·8.

It may be assumed that during a normal six-hour working shift the efficiency would continue to fall with each successive hour and after the third hour a serious falling off in output would occur in conditions less oppressive than those found in the present investigation.

(2) In a complete three-hour shift the efficiency began to fall when the wet-bulb temperature rose above 88°F., the effective temperature above 85°, and the wet kata fell below 18·5. When conditions deteriorated the efficiency fell at first gradually and then more rapidly. If it is assumed that work was (*a*) moderately impaired when the efficiency was 75 per cent, (*b*) seriously impaired at 50 per cent, and (*c*) useless below 25 per cent, these stages were reached :

(*a*) When the wet-bulb temperature rose above 89·5°F., the effective temperature above 91·5°, and the wet kata fell below 7·5.

(*b*) When the wet-bulb temperature rose above 92·5°F., the effective temperature above 96·0°, and the wet kata fell below 4·0.

(*c*) When the wet-bulb temperature rose above 95°F., the effective temperature above 98·5°, and the wet kata fell below 2.

Again it may be assumed that during a normal six-hour working shift there would be a greater all-round reduction in efficiency under the conditions described.

These findings are at variance with those of other workers. Thus Haldane (1)* in 1905 believed that the limit of active work was at 78°F. wet-bulb temperature in still air, 85°F. in air moving at a velocity of 135 ft. per minute, and that at 80°F. the work of a miner decreased. This statement, however, conflicts with a later quotation by Haldane (2) (in 1929) from the Eighth and Ninth Reports on 'The Control of Atmospheric Conditions in Hot and Deep Mines' (by Prof. K. Neville Moss), in which he states :

> About 50 men were working in the places selected, the wet-bulb temperature being 86·5°F. and the dry-bulb temperature 100·5°F. The mean wet kata reading was 10·0. Their average output per man shift was actually greater than in cooler parts of the same pit, the coal being easier to get. . . . It was clear, at any rate, that even with as low a wet kata reading as 4·0 a quite ordinary amount of work could be done when, as in that case, the men worked without clothing and were thoroughly acclimatized. Since it has often been assumed that a wet kata reading several times higher is necessary this observation is especially significant.

This quotation is given in full, for it is 'especially significant' and emphasizes the inaccuracy of Haldane's earlier oft-quoted belief that at 85°F. wet-bulb temperature hard work was virtually impossible. In the present investigation the efficiency was hardly affected at 85°F. wet-bulb temperature, but fell to about 85 per cent at 86·5°F., and at a wet kata of 4·0 there was definite reduction to 60 per cent.

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

The results obtained by Bedford and Warner (3) in efficiency tests on British coal miners is shown in Table III, as arranged by Yaglou.

TABLE III

WORKING CAPACITY OF MINERS IN RELATION TO EFFECTIVE TEMPERATURE AND WET KATA COOLING POWER

Effective Temperature	*Wet Kata Cooling Power*	*Rest Pauses, Min. per Hour*	*Tub-filling Time, Min.*	*Relative Output, Per cent*
65·8	18·6	7·3	8·0	100
65·8	14·6	6·7	8·6	94
75·3	12·9	9·0	8·5	91
75·8	10·8	10·0	9·2	82
77·8	9·0	11·1	9·1	81
81·8	6·4	22·4	9·6	59

It will be seen that the relative output is 59 per cent at an effective temperature of 81·8° and wet kata of 6·4, whereas in Southern Indian labourers the efficiency remained at 100 per cent at an effective temperature of 81·8° and was about 70 per cent at a wet kata of 6·4. It will also be noted that the effective temperatures representative of the wet kata readings quoted differ widely from those found in the present investigation. Thus on the Kolar goldfield a wet kata cooling power of 6·0 was more likely to be represented by an effective temperature of 93° than 82°.

The explanation offered for these marked differences is the opinion given by Hinsley (4) that there is no simple connection between the effective temperature and wet kata cooling power. Hinsley points out that a wet kata of 12 in different underground conditions could be represented by an effective temperature ranging between 63° and 83°. This observation was confirmed in the present investigation, for wet kata readings of 7 to 8 were represented by effective temperatures ranging between 81° and 97°.

Ehrismann and Hasse (5) (quoted in a Review of Literature on 'Conditioning Air for Advancement of Health and Safety in Miners') report a fall in efficiency in German coal miners of 33 per cent to 69 per cent between wet kata cooling powers of 6·3 and 8·2. At these wet kata readings the efficiency in Southern Indian mine workers was 70 per cent and 80 per cent respectively. These authors also consider an effective temperature of 82° to be the limit for a full working period. At 82° on the Kolar goldfield efficiency was unimpaired.

Orenstein and Ireland (6) found that the working efficiency of native South African miners was reduced when the wet kata cooling power fell below 16·0 and at a wet kata of 5 the efficiency was about 55 per cent. In the present investigation the efficiency figures were somewhat higher for comparable conditions. Thus the efficiency began to fall at a wet kata of 13·5 and was 60 per cent at a wet kata of 5. It is interesting to note that the efficiency figures for native South African miners do not differ from those found in the present investigation to the extent noted in comparing efficiency tests on European miners.

Yaglou (7) found the upper limit of temperature efficiently endured with heavy manual work to be 80° effective temperature. Above 75° the output decreases gradually until 80° is reached, when the fall becomes very rapid. Hinsley agrees with Yaglou that for best conditions the effective temperature should not exceed 80°, but adds that a fall in output occurs when the effective temperature rises above 85° and at 90° and over little or no work can be done. It is agreed that the efficiency falls when the effective temperature rises above 85°, but in Southern Indian mine workers at 90° effective temperature the efficiency was as high as 80 per cent. The stage of ' little or no work ' was reached only when the effective temperature rose above 98·5°.

The attitude of the average mining engineer to the relative high efficiencies found in the present investigation has, we understand, changed during the past 25 years. In 1920, when it was stated in the First Report of the Committee on ' The Control of Atmospheric Conditions in Hot and Deep Mines ' that about 80°F. might be assumed as the upper limit for the performance of efficient work, the results would have been received with some surprise. It is now being realized, however, this estimate is too low for British coal mines. This is supported by the observation of Moss (8) already quoted and that of Lawton (9) who found that acclimatized men ' can work quite efficiently ' at wet-bulb temperatures of 88°F., provided there is adequate ventilation. It is interesting to note that the calculated effective temperatures for some of the working places with a wet-bulb temperature of 88°F. described in Lawton's paper is 90°. If these are the places referred to by Lawton in which men worked efficiently the upper limit of effective temperature for working efficiency in British coal miners given by Hinsley should be revised.

The reason for the relative high efficiency in the Southern Indian labourer under seemingly oppressive conditions is not difficult to understand. It is well known that conditions considered hot and oppressive in the average mine in Britain may be pleasant to the acclimatized worker in the tropics. The Southern Indian labourer on the Kolar goldfield is accustomed to surface atmospheric temperatures up to 100°F. dry-bulb and 75°F. wet-bulb temperatures. His tolerance to high temperatures is therefore much higher than that of the miner in a temperate climate and is further increased by acclimatization to high underground temperatures. Even the European underground workers on the Kolar goldfield do not find wet-bulb temperatures of 85°F. uncomfortable and provided there is adequate ventilation wet-bulb temperatures up to 90°F. are not considered unduly oppressive. Indeed the investigators were surprised when the tests showed that the efficiency began to fall at a wet-bulb temperature as low as 84°F.

The fact that underground conditions comfortable to a European need not necessarily be comfortable to an Indian is demonstrated in the results of the tests carried out in a main downcast level. A number of tests were performed in this place, the dry-bulb temperatures being 85° to 86°F., the wet-bulb temperatures 78° to 75°F., the effective temperatures 65° to 69°, wet kata cooling power 35 to 32, and air velocity 740 to 900 ft. per minute. In all tests the efficiency was appreciably lower than in tests performed in average working places. It may well be that the lowered efficiency was partly due to the discomfort occasioned by the high velocity, but it is also probable that temperature conditions were below the optimum for efficient work in Southern Indian labourers. This is supported by Penman's ([10]) observation that acclimatized men working in Indian coal mines at wet-bulb temperatures over 85°F. with very little air movement (wet kata about 5·7) did not like being transferred to cooler places.

Another and less obvious reason for the apparent high efficiency figures is the difference between the energy per hour expended by labourers in the tropics and in temperate climates. The Southern Indian labourer has sensibly adjusted his rate of work to that demanded by tropical conditions. His output per hour may be less, but energy is conserved which enables him to tolerate and continue working in hot underground atmospheres for longer periods. The 100 per cent efficiency figure decided upon in this investigation might be considered low for the average European miner, but it is certain that a decline in efficiency would occur earlier and at lower temperatures in a European than in an Indian.

Comparison between the Relative Value of the Wet-Bulb Temperature, Effective Temperature, and Wet Kata Cooling Power as Indices of Working Efficiency

The degree of comfort or discomfort of underground working conditions is commonly expressed in terms of wet-bulb temperature, wet kata cooling power, or effective temperature. The wet kata thermometer was introduced as an improvement on the wet-bulb thermometer, and later the effective temperature, which embraces the dry- and wet-bulb temperatures and air velocity, was evolved in America as the most satisfactory index of comfort conditions.

A deterioration in the comfort of working conditions beyond a certain point is bound to be associated with a decreasing ability of men to perform work. The ideal index of comfort conditions should therefore also be the ideal index for measuring working efficiency. Such an ideal should theoretically enable an efficiency curve to be drawn after a sufficient number of carefully-supervised tests has been performed under varying conditions (provided of course the men are in a comparable healthy state in all tests and the only variable factor is working conditions). The individual points on the graph should be grouped closely around the mean curve.

Applying this principle to the findings in the present investigation it appears from inspection of Figs. 2 to 4 that neither the wet-bulb temperature, effective temperature, nor wet kata cooling power is an ideal index of working efficiency. The graphs show that although it was possible to draw mean curves for the wet-bulb temperature, effective temperature, and wet kata cooling power, many points in all graphs were well above or below the line. The anomalous findings in some of these tests could undoubtedly be attributed to the human factor, but in others it is probable that degrees of comfort or discomfort were not correctly interpreted by the wet-bulb temperature, effective temperature, or wet kata cooling power.

A careful examination of the grouping of individual points around the mean curves demonstrates there is little to choose between the wet-bulb temperature and effective temperature groupings, while in the wet kata cooling power graphs the points are much more haphazard. On this basis it may be concluded that :

(*a*) The wet kata cooling power is a less accurate index of working efficiency than either the wet-bulb temperature or effective temperature.

(*b*) The wet-bulb temperature is as useful an index of working efficiency for conditions occurring in the mines on the Kolar gold-field as the effective temperature.

These findings are somewhat unexpected, for the effective temperature is now generally accepted as being the most accurate index of comfort conditions. In addition there has been a tendency in recent years to favour the wet kata cooling power rather than the wet-bulb temperature when describing comfort conditions. Further information was therefore sought by comparing graphically the wet-bulb temperatures, effective temperatures, and wet kata cooling power in each test. This is shown in Fig. 7, which also includes the dry-bulb temperatures, air velocity, and the number of inches drilled. The tests are arranged for comparison in order of wet-bulb temperature ascendency. The important facts brought out are summarized thus :

(1) The rise in wet-bulb temperature is associated with an almost comparable rise in effective temperature. Up to wet-bulb temperatures of 84°F. there are many fluctuations in the effective temperature curve due to marked variations in the air velocity and/or dry-bulb temperature. Beyond 85°F. wet-bulb temperature the fluctuations are less obvious, the effective temperatures being affected to a lesser extent by changes in air velocity and dry-bulb temperature.

(2) Excepting when conditions are oppressive there is no definite relationship between the wet kata cooling power and wet-bulb temperature or effective temperature. The wet kata curve closely follows the air velocity curve and is unduly influenced by changes in velocity at the expense of the dry- and wet-bulb temperatures.

(3) When the wet-bulb temperature is between 76° and 84°F. working efficiency is little affected by variations in the dry-bulb temperatures (83° to 110°F.) and air velocity (0 to 700 ft. per minute). It is at this range of wet-bulb temperature that the effective temperature expresses more accurately comfort conditions, but as Southern Indian mine workers are unaffected by these conditions there is little advantage to be gained in using the effective temperature rather than the wet-bulb temperature as an index of working efficiency.

(4) In the higher ranges of wet-bulb temperatures—i.e., above 86°F.—when working efficiency falls the worsening of conditions is shown almost as well by the wet-bulb temperature as by the effective temperature. There are variations in efficiency which show a closer relationship to the effective temperature, but there are also examples when a rise in effective temperature is not associated with the expected fall in output and vice versa.

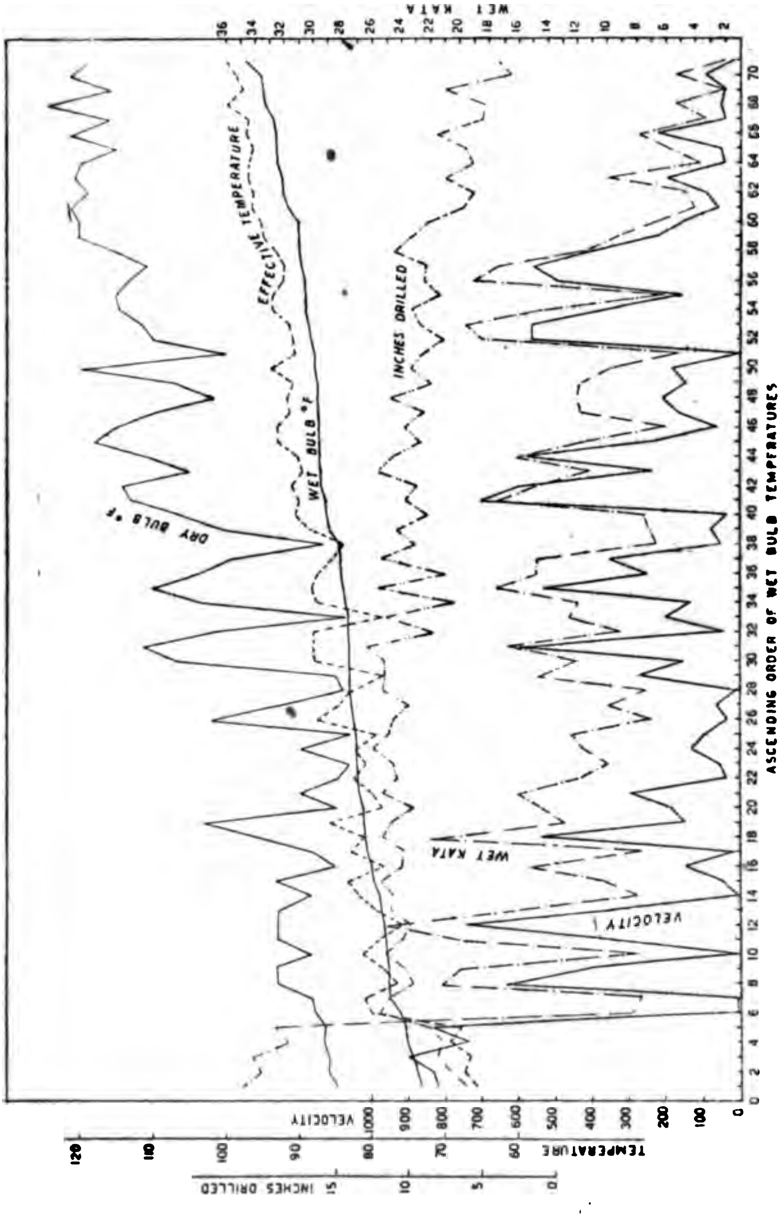

FIG. 7.—Graphical comparison, in order of wet-bulb ascendency, of dry-bulb, wet-bulb, and effective temperatures, inches drilled, air velocity and wet kata cooling power.

(5) The wet kata curve generally bears less relation to working efficiency than either the wet-bulb temperature or effective temperature.

It is therefore concluded that although comfort or discomfort conditions are best described in terms of effective temperature there is much to be said in favour of the use of the wet-bulb temperature as an index of working efficiency on the Kolar goldfield. There is every reason to support a wider use of the effective temperature in deep mining, for its calculation necessitates the determination of dry- and wet-bulb temperatures and air velocity. Thereby the attention of the mining engineer is kept focused on the important subject of ventilation.

There does not appear to be any justification in recommending the use of the wet kata thermometer as an aid to ventilation and working efficiency problems on the Kolar goldfield. The wet-bulb thermometer is familiar to all mining engineers and is very much easier to use than the wet kata thermometer. It is felt that until the mining engineer becomes effective-temperature minded he will be well served by the familiar wet-bulb temperature.

Notes on the Physiological Changes

(1) *Body Temperature.*—A rise in body temperature occurred even when working conditions were comfortable and became more marked when conditions deteriorated. The relationship of the rise to wet-bulb temperatures is shown in Table IV. The rise

Table IV

Relationship Between Rise in Body Temperature and Wet-Bulb Temperatures

Wet-Bulb Temperature °F.	*Rise in Body Temperature °F.*
Up to 84	0·2 — 1·2
85 — 89	0·6 — 1·4
90 — 94	1·2 — 2·8
Above 94	2·6 — 3·4

varied inconsistently with each individual and the maximum rise during the shift always occurred before the end of the second hour. This was almost invariably followed by a fall by the end of the third hour even when conditions were oppressive.

(2) *Pulse Rate.*—The inevitable rise in pulse rate occurred with increasingly oppressive conditions, but it varied so greatly with the individual (irrespective of underground conditions) that it is difficult to present the findings in a compact tabular form. The relationship between the rise in pulse rate and wet-bulb temperature is shown in Table V, but it is emphasized only rough estimates are given.

TABLE V

RELATIONSHIP BETWEEN RISE IN PULSE RATE AND WET-BULB TEMPERATURE

Wet-Bulb Temperature °F.	*Rise in Pulse Rates/Min.*
Up to 82	10 — 70
83 — 87	35 — 80
88 — 97	40 — 90

The difficulties in interpreting the findings will be appreciated by the following examples: In one test when the wet-bulb temperature was 78°F. the rise in pulse rate after one hour's work was 8 beats per minute in one worker and 55 in another. In other tests the rise in a certain individual was higher at 84°F. than at 94°F. Unlike the changes in body temperature, however, relatively high or low pulse rates were usually consistently found in the same individuals. The maximum rise in the pulse rate (as with the body temperature) occurred before the end of the second hour and by the end of the third there was an appreciable fall.

(3) *Blood Pressure.*—Unfortunately, owing to the shortage of staff, blood pressure recordings were carried out only in a few tests when conditions were oppressive. In all tests wet-bulb temperatures were above 93°F. (effective temperatures above 96°). In every case there was an invariable fall in the systolic blood pressure of 20 to 30 m.m. Hg., and recovery to normal did not occur until at least two hours after the end of the shift. The diastolic pressure varied : in some cases it was indeterminable and in others the fall was comparable to that of the systolic blood pressure. It is important to note that in spite of the fall in blood pressure no complaints were made by the workmen and even by direct questioning no untoward symptoms could be elicited.

(4) *Sweat Loss.*—The sweat loss in oz. per hour working shift is plotted against the wet-bulb temperature in Fig. 8. It will be

seen that although there is much variation the quantity of sweat lost increases steadily with a rise in wet-bulb temperature from an average of about eight oz. per hour at 74°F. to 30 oz. at 93°F. When conditions further deteriorate the sweat loss falls, presumably due to the fact that less energy is expended in work.

(5) *Water Consumption.*—This also varied greatly from an intake of nil at low wet-bulb temperatures to an average of 25 oz. per hour at the higher ranges of temperature. The water consumption was usually about six oz. per hour less than the sweat loss. In a few tests the fluid intake was increased to an amount approximating the estimated sweat loss. This was resented in all cases and when forced resulted in nausea.

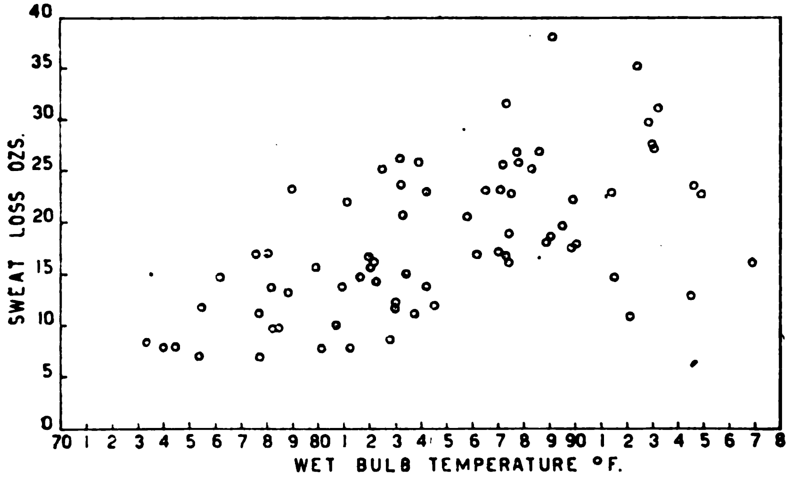

FIG. 8.—Relationship between sweat loss and wet-bulb temperature.

(6) *Plasma Chloride.*—A comparison between the plasma chloride estimations before and at the end of the shift in six tests when conditions were oppressive showed no significant fall in spite of a total sweat loss varying from 96 to 192 oz. It is necessary to add that salt water was not drunk during these tests.

(7) *Haemoglobin and Red Blood Cell Counts.*—The object in carrying out these tests was to investigate the possibility of haemo-concentration occurring as the result of the high sweat loss. The tests were done at the same time as the plasma chloride estimations and it was found there were no significant changes before and after the shift. There is therefore no evidence that haemo-concentration occurs under normal circumstances.

INTERPRETATION OF PHYSIOLOGICAL CHANGES

It is possible for workers in deep mines to become acclimatized over a period of years to conditions which would be intolerable to the unacclimatized person. A stage is reached, however, when it becomes dangerous for men to work in excessively oppressive conditions. It was therefore hoped that an investigation of the physiological changes would demonstrate useful indications of latent signs of distress associated with a serious falling off in working efficiency.

The only significant sign found was the consistent fall in blood pressure, certainly when the wet-bulb temperature rose above 93°F. and possibly at lower readings. It has been pointed out elsewhere ([11]) that heat collapse is brought about by a breakdown in the efficient functioning of the cardio-vascular system and a constant finding in cases of heat collapse is a fall in blood pressure. It is therefore reasonable to assume that when an underground labourer is working in conditions which are such that a fall in blood pressure is produced the addition of some predisposing factor is sufficient to precipitate collapse. If with the fall in blood pressure at a wet-bulb temperature of 93°F. (effective temperature of 96°) is coupled the serious falling off in efficiency during a three-hour shift, there is sufficient evidence to show that prolonged work under these conditions may be harmful to the health of the individual. In the absence of positive information 93°F. wet-bulb temperature is given as the absolute limit at which prolonged work can be carried out without undue strain on the acclimatized individual. When further observations have been made it is extremely likely that a significant fall in blood pressure will be found at wet-bulb temperatures between 90° and 93°F.

In this connection it is interesting to note that in the three tests when one or more men stopped working on account of fatigue before the end of the three-hour shift, the wet-bulb temperatures were 94·9°, 95°, and 96·9°F. and the effective temperatures 98·7°, 98·0°, and 100·1°. Other tests were, however, completed at similarly high wet-bulb and effective temperatures without undue fatigue.

No consistent and significant rise in body temperature was noted until wet-bulb temperatures rose above 94°F. The rise in pulse rate also appeared to be of little value in pointing to a critical limiting temperature. The highly variable findings prevented definite conclusions being drawn.

The results of the plasma chlorides are of interest, although unconnected with the present discussion. The fact that no reduction in plasma chloride occurred in spite of a heavy sweat loss and no salt intake during the working shift indicates that the body reserve of chloride is the important factor in maintaining an adequate plasma chloride level. Provided the salt intake for the days previous to the shift is sufficient there is little danger of a salt deficiency arising. This statement does not infer that it is not necessary to take salt during work; on the contrary, it is strongly advised.

Practical Applications

Although the findings in the present investigation have a specific application to mines of the Kolar goldfield their significance is of importance wherever deep-mining operations are carried out. In mines such as those of the Kolar field, where the reefs dip at a steep angle, the transition from comfortable to uncomfortable conditions takes place slowly over a period of years. Associated with the decline in comfort conditions is increased acclimatization of the mine worker, so that for a time no diminution in efficiency is experienced. Beyond a certain point, however, a decline in capacity to perform useful work occurs in spite of the increased acclimatization acquired, but such a decline is less than might be expected, because of the extraordinary ability of man to adapt himself to very high underground temperatures. As the process is insidious a reasonable degree of efficiency can still be maintained in conditions which are harmful to health. When conditions deteriorate further the output ultimately reaches levels which are uneconomical, even in fully acclimatized workers. These factors are significant for mines working at what might be termed the 'critical efficiency range' of wet-bulb temperature.

The mining engineer in deep mines is faced with serious problems when the efficiency of workers acclimatised to underground conditions begins to decline to uneconomic levels and especially when this is associated with possible harmful effects on the health of the mine worker.

The temperatures at which a fall in efficiency occurs may vary somewhat between mines in the tropics and those in temperate climates. In the present investigation it was found that during a three-hour shift a fall in efficiency became noticeable when the wet-bulb temperature rose above 83°F. and effective temperature above 85°. The loss in capacity became more serious at 90°F. wet-bulb temperature and 93° effective temperature. Ultimately when the wet-bulb temperature rose above 93°F. and effective temperatures above 96° little useful work could be performed after the second hour. In addition work was being carried out in conditions harmful to the health of the individual.

These findings refer to a three-hour shift, but in making practical deductions the results must be interpreted in terms of a six-hour shift, this being a fair average of the time a worker may be expected to spend at the face after allowing for meal time and travelling to and from surface. It is reasonable to suppose that an efficiency curve for a six-hour shift would be displaced to the left—that is to say, a decline in efficiency would commence at lower temperatures and in the higher range of temperatures the working efficiency would decline much more rapidly.

In a normal working shift no serious diminution of work is likely when wet-bulb temperatures are kept below 85°F. and effective temperature below 86°, but if temperatures are allowed to rise to 90°F. wet bulb, comparable to about 98° effective temperature, it is extremely unlikely that for a six-hour period the efficiency would be any higher than 60 per cent. When wet-bulb temperatures reach 93°F. and the effective temperatures 96° it is questionable whether any satisfactory work can be performed after the second hour and certainly not after the third hour. Thus workmen remaining in such atmospheres for periods longer than three hours can only do so by drastically reducing their hourly output of work. This means in effect that when temperatures rise above 90°F. wet bulb, although the time at work may be doubled or even trebled, the actual amount of work done remains the same.

It can be pointed out that these figures are based upon efficiency tests performed by men acclimatized to atmospheres up to wet-bulb temperatures of 93°F. There is no doubt that higher efficiencies than those found can be attained in wet-bulb temperatures above 93°F. by men who have become acclimatized to such temperatures over a period of months. Even so the efficiency is economically low and there is always the danger of heat collapse occurring at these very high temperatures.

Two of the most important factors which determine comfort conditions in deep mines and over which control can be exercised are humidity and air velocities. The steps which should therefore be taken to improve or maintain a desirable standard of comfort are: First, the control of moisture underground to prevent the air in its passage through the mine airways from absorbing additional moisture, and, secondly, the speeding up of air velocities in the working places by the use of auxiliary or booster fans. A monthly effective temperature survey of all working points is of great assistance in this respect, because the effective temperature necessitates the determination of the dry-bulb and wet-bulb temperatures and the air velocity. It also indicates what degree of improvement could be effected in comfort conditions.

There are, however, obvious practical and economic limitations to the control of air velocities and water underground. In most mines it is impossible to keep the air from contact with all wet places, but, wherever practicable, all gutters, sumps, etc., should be provided with covers, and all wet shafts or roadways lined with some suitable material to prevent the air from coming in contact with water and picking up additional moisture.

In dry and dusty mines with high temperatures the following factors limit the usefulness of air velocity as a cooling medium :

(1) The amount of dust and general discomfort in working conditions when air currents are increased beyond a certain speed.

(2) The impracticability under normal working conditions of providing every working place with sufficiently high air velocities.

(3) Temperatures reach a point when ventilating power costs rise out of all proportion to the additional comfort obtained.

When the control of humidity and adequate mechanical ventilation fails to prevent a continued decline in comfort conditions the use of conditioned air must be considered.

It has already been stated that efficiency falls when wet-bulb temperatures of 83°F. and effective temperatures of 85° are exceeded. How far conditions may be allowed to deteriorate beyond this point can only be determined by the economic and practical considerations peculiar to any individual mine, but when wet-bulb temperatures approach 90°F., if work is to be undertaken with any degree of efficiency and comfort to the workmen some form of air conditioning becomes an absolute necessity.

Summary and Conclusions

The working efficiency in Southern Indian mine workers under varying underground conditions has been investigated experimentally and the method used described in detail.

Seventy-one tests carried out by three pairs of hand drillers in granite have been described. Each test lasted three hours and at the end of each hour the depth of hole drilled was measured. The amount of work done was plotted against the wet-bulb temperature, effective temperature, and wet kata cooling power. Mean curves were drawn showing the working efficiency for the first, second, and third hours and for the complete three-hour shift.

The efficiency curves have been analysed and discussed and the salient features are :

(1) During the three-hour shift the output of work fell hourly. This hourly decline in efficiency was more marked when conditions became more oppressive. It was noted that there was a serious falling off in efficiency (say 50 per cent) :

(*a*) After two hours' work when the wet-bulb temperature rose above 91·5°F., the effective temperature above 94·5°, and the wet kata cooling power below 5.

(*b*) After one hour's work when the wet-bulb temperature rose above 93°F., the effective temperature above 96·5°, and the wet kata reading fell below 4.

Ultimately a stage was reached when little useful work could be performed (say 25 per cent efficiency) :

(*a*) After two hours' work when the wet-bulb temperature rose above 94°F., the effective temperature above 97°, and the wet kata reading fell below 3.

(*b*) After one hour's work when the wet-bulb temperature rose above 95·5°F., the effective temperature above 99°, and the wet kata reading fell to 2·0.

During the first hour work was not seriously impaired until the wet-bulb temperature rose above 94·5°F.

(2) In the complete three-hour shift the efficiency began to fall when the wet-bulb temperature rose above 83°F., the effective temperature above 85°, and the wet kata fell below 13·5. The fall became increasingly more marked when conditions deteriorated. The efficiency was moderately impaired—say, at 75 per cent efficiency—when the wet-bulb temperature rose above 89·5°F., the effective temperature above 91·5°, and the wet kata fell below 7·5 ; seriously impaired—say, 50 per cent efficiency—when the wet-bulb temperature rose above 92·5°F., the effective temperature above 96°, and the wet kata fell below 4 ; and little or no useful work could be done when the wet-bulb temperature rose above 95°F., the effective temperature above 98·5°, and the wet kata fell below 2.

It is pointed out that in a normal six-hour working shift the efficiency will be further reduced under the equivalent conditions. It was decided that at a wet-bulb temperature of 90°F. and effective temperature of 93° the efficiency would probably only be about 60 per cent. When wet-bulb temperature reaches 93°F. and effective temperature 96° it is questionable whether any useful work could be done after the second hour and certainly not after the third hour.

The findings in the present investigation are compared with those of other workers and the relative high efficiencies under seemingly oppressive conditions are attributed to the acclimatization of the Southern Indian labourer to high surface and underground temperatures. It is also suggested that the Southern Indian labourer, in common with other labourers in the tropics, conserves his energy by sensibly reducing his rate of work.

The relative merits of wet-bulb temperature, effective temperature, and wet kata cooling power as indices of working efficiency are compared. It was found that for local conditions the wet-bulb temperature was as useful an index of working efficiency as the effective temperature. The wet kata cooling power was generally less accurate an index than either the wet-bulb temperature or effective temperature. Possible reasons for these findings are given. It is further suggested that the effective temperature has not proved to be much superior to the wet-bulb temperature as an index of working efficiency, because the effective temperature scale drawn up by the originators does not reflect sufficiently the beneficial effects of air movement at higher ranges of temperature. As Critchley ([12]) has pointed out, the effective temperature is an empirical standard drawn up from the work of a relatively small number of Americans. It also does not take into account the acclimatization factor and its validity in the higher ranges of temperature and air velocities has been impugned. The findings in the present investigation tend to support the latter statement and a further series of tests at the highest temperatures may permit a more accurate chart to be drawn up. Nevertheless, under normal working conditions, the effective temperature is considered a more accurate index of comfort conditions than either the wet-bulb temperature or wet kata cooling power and its wider use in mining is recommended. However, the wet-bulb temperature is familiar to every mining engineer and until the average mining man becomes more conversant with the effective temperature the wet-bulb temperature may be used as a sufficiently accurate index of working efficiency and indeed comfort conditions.

The physiological changes are described and discussed. It was found that a fall in blood pressure occurred when the wet-bulb temperature rose above 93°F. and the effective temperature above 96°. It is considered that the blood pressure provides more useful information than either the body temperature or pulse rate, in regard to the ability of men to endure high underground temperatures. A fall in blood pressure should be looked upon as an ominous sign of the possible occurrence of heat collapse and indicates that further measures should be taken to improve working conditions.

The practical implications of the findings are discussed and it is stated that when, in spite of good mining practice and especially adequate mechanical ventilation and control of moisture, wet-bulb temperatures continue to rise above 85°F. and effective temperature above 87° air conditioning should be seriously considered. When wet-bulb temperature approaches 90° and effective temperature 93° air conditioning becomes a necessity.

Acknowledgment.—The authors wish to express their thanks to the Superintendent, Mysore Gold Mining Co., Ltd., and the Chief Medical Officer, Kolar goldfield, for their permission to carry out this investigation. They are highly indebted to Mr. L. H. McLaggan, who ably assisted them in collecting and classifying the data and drawing the graphs, and to Mr. F. G. Sharp, who supervised the tests and took all temperatures and velocity readings.

References

(1) Haldane, J. S. The influence of high air temperature. *J. Hyg.*, 1905, 5, 494.

(2) Haldane, J. S. Work of the Committee of the Institution of Mining Engineers on 'The Control of Atmospheric Conditions in Hot and Deep Mines'. *Trans. Instn. Min. Engrs.*, 1929, 77, 338.

(3) Yaglou, C. P. Abnormal air conditions in industry: their effects on workers and methods of control. *J. industr. Hyg.*, 1937, 19, 12.

(4) Hinsley, F. B. Some aspects of deep-mine ventilation (Thirtieth Report to the Committee on 'The Control of Atmospheric Conditions in Mines'). *Trans. Instn. Min. Engrs.*, 1944, 103, 567.

(5) Ehrismann, O., and Hasse, A. Permissible working time at high temperature and humidity of the air. *Arch. Gewerbepath. Gewerbehyg.*, 1938, 8, 611.

(6) Orenstein, A. J., and Ireland, H. J. Experimental observation upon the relation between atmospheric conditions and the production of fatigue in mine labourers. *J. industr. Hyg.*, 1922, 4, 70.

(7) Yaglou, C. P. The thermal index of atmospheric conditions and its application to sedentary and to industrial life. *J. industr. Hyg.*, 1926, 8, 5.

(8) Moss, K. N. The mechanical efficiency of the human body during work in high air temperatures (Ninth Report to the Committee on 'The Control of Atmospheric Conditions in Hot and Deep Mines'). *Trans. Instn. Min. Engrs.*, 1924–25, 68, 377.

(9) Lawton, B. R. Atmospheric conditions in deep mines (Twenty-fifth Report to the Committee on 'The Control of Atmospheric Conditions in Hot and Deep Mines'). *Trans. Instn. Min. Engrs.*, 1936–37, 93, 37.

(10) Penman, D. Atmospheric conditions in Indian coal-mines (Fifteenth Report to the Committee on 'The Control of Atmospheric Conditions in Hot and Deep Mines'). *Trans. Instn. Min. Engrs.*, 1928–29, 76, 340.

(11) Caplan, A. A critical analysis of collapse in underground workers on the Kolar Goldfield. *Bull. Instn. Min. Metall.*, 1943, No. 463.

(12) Critchley, M. Problems of naval warfare under climatic extremes. *Brit. med. J.*, Aug. 4, 1945, p. 146.

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The Geology and Opencast Mining of the Jurassic Ironstones of Great Britain.

By W. David Evans, *Associate*.

Introduction

In Great Britain the mining of bedded ironstones of Jurassic age has afforded a classic example of the development of mechanized opencast methods dating back to 1895—the year in which the first mechanical excavator was introduced to opencast working of ironstone in this country. This machine was a Wilson steam crane-navvy, carrying a 1½-cu. yd. bucket. It was used for stripping overburden from the Northampton Sand ironstone at Corby, Northants. Prior to this the bedded Jurassic ironstones were worked by hand in open-pits throughout the east Midlands. To-day, as a result of improvements in mechanized opencast mining, these bedded ironstones rank second to coal in our country's natural resources.

Geology of the Jurassic Ironstone Field

Beds of ironstone occur on several horizons in the Jurassic rocks throughout eastern England. Of these the Frodingham Ironstone (Lower Lias), the Cleveland Ironstone (Middle Lias), the Marlstone Ironstone (Middle Lias), and the Northampton Sand Ironstone (Inferior Oolite) have proved workable over large areas. The stratigraphical position of these ironstones is shown in Fig. 1, which also indicates the nature and approximate thicknesses of the overlying strata.

The Cleveland Ironstones.—Among the stratified iron-ores of Britain those of Cleveland, north Yorkshire, at one time took first place both in respect of output and industrial consequence.

These ores occur as interbedded deposits in the upper part of the Middle Lias, and are almost wholly obtained from a band known as the Main Seam. After some minor trials on the coast, the exploitation of the main seam on a large scale was begun in the year 1850 at its northern outcrop near Easton.(6)*

*Figures in parentheses refer to references given at the end of the paper.

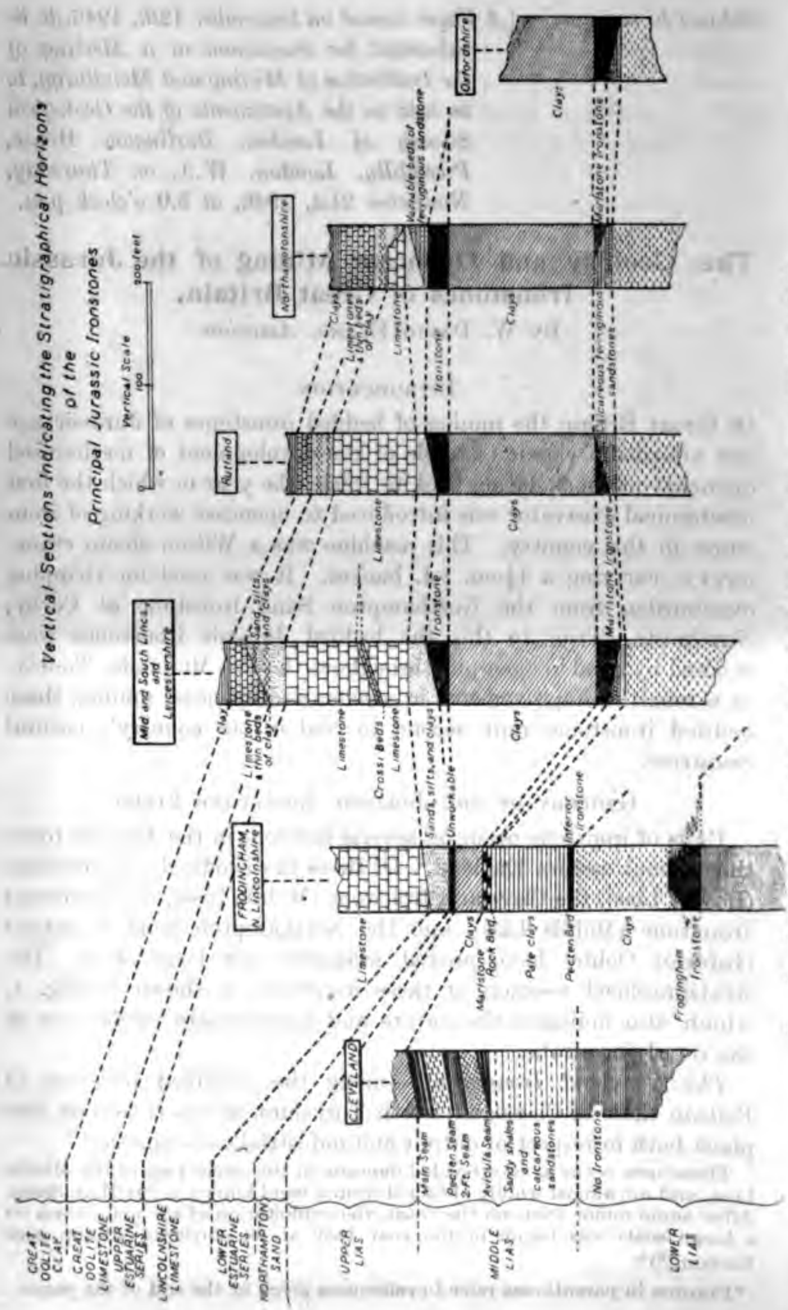

FIG. 1.—Comparative sections of the Jurassic ironstone field.

Henceforth the industry developed rapidly, and led to the institution of the iron and shipping industries of the Tees estuary and of Middlesborough.

The geological structure of the ore-field is relatively simple, the strata dipping gently to the east. Locally, however, high angles of dip in an opposite direction are met with, owing to folding, and occasionally the ore-beds are dislocated by faults. Of these, the principal ones are: (1) The Upsall Fault, running W.S.W. to the south of Easton Hills, which has a northerly downthrow of a maximum of 400 ft., but diminishing rapidly eastward, and (2) the Lockwood Beck Fault, which runs north-south, and has a downthrow east of about 240 ft. in places.

At the outset outcrop ironstone was worked in open pits, but the bulk of ore produced was obtained from many extensive mines, of which only eight are operating to-day, working on the bord-and-pillar system. These workings were principally in the Main Seam (Fig. 1), from which the ore averaged 27 to 30 per cent iron, 10 to 15 per cent silica, 8 to 18 per cent alumina, and 4 to 7 per cent lime.

Despite the fact that output from the Cleveland Field has waned in the past 25 years (Fig. 2), it is still an important source of iron ore. However, of considerable interest is the recently published estimate of reserves (1), which is quoted in Table I. Estimated

TABLE I

	Ore-Bed	*Per cent Iron*	*Tons*
Main Seam	in present mine-field.........	Slightly under 30 ...	60,000,000
	in abandoned mine-field...	„ „ 28 ...	60,000,000
	in inferior ground south of mined area...............	25 to 28 ...	150,000,000
Pecten Seam, probably mostly unworkable		About 26 ...	40,000,000
Two-Foot Seam, in northern area, if workable		30 ...	50,000,000
Avicula Seam, in southern area, if workable		25 ...	10,000,000
		Total tonnage	370,000,000

on the 1943 output of 1,756,000 tons this reserve indicates a life of just over 200 years. Assessing it on the proved reserves in the Main Seam it seems more probable that the Cleveland field has a life of up to 50 years.

The Frodingham Ironstone.—The Frodingham ironstone is restricted in occurrence to the Scunthorpe district of north Lincolnshire, where it occurs in the shales and clays of the Lower Lias formation (Fig. 1). It extends for over seven miles southwards from near the Humber, and forms a broad outcrop up to two miles wide which embraces an area of nearly eight and three-quarter square miles. Within this outcrop, from a point midway between the villages of Coleby and Thealby southward nearly to Bottesford, the ore is up to 32 ft. thick and is workable almost continuously by opencast methods. North and south of this area the ore deteriorates rapidly and, so far as is known, it nowhere recovers its quality and great thickness.

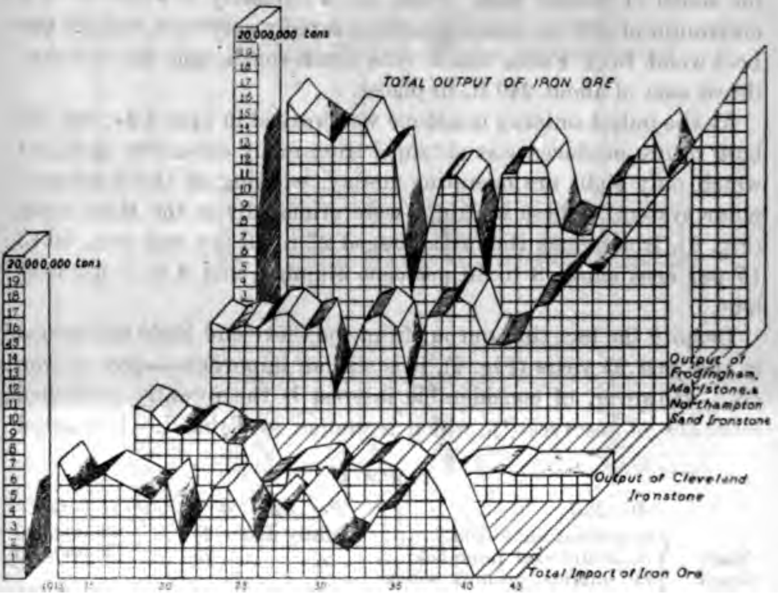

FIG. 2.—A graphical representation of imports and outputs of iron ore.

The broad outcrop of the Frodingham ironstone is due to the gentle eastward dip of the strata and the low relief of this part of north Lincolnshire. The dip rarely exceeds 1 in 100 and the ironstone is seldom dislocated by faults or affected by folds. Faults occur, but their throw rarely exceeds 10 ft. Nevertheless, their occurrence, even to this small degree, seriously hampers some of the workings. Similarly, the small folds which sometimes occur within the ironstone field obstruct development of the ironstone. The western parts of this broad outcrop of ironstone are sometimes concealed by glacial sands, gravels, and boulder clay, as well as more recent deposits of blown sand. The removal of this cover, although not free from attendant problems, has been effected with comparative ease, even in the early days of hand-worked pits.

Along the eastern margin of the outcrop the ore is concealed by blown sand and the shales, mudstones, clays and ferruginous limestones of the Middle and Upper Lias formations, which stratigraphically succeed the Frodingham ironstone. This latter group of deposits is naturally increasing in thickness as the ore-deposit is being worked down-dip; in many places it will soon attain a thickness beyond the capacity of the present-day excavators. Consequently the reserves of outcrop ironstone, and ore beneath shallow cover, are diminishing rapidly, and it is evident that open-pit workings will be replaced eventually by underground mines in the ground to the east of Scunthorpe. Already one such mine is operating in this district.

The unweathered ore is a dark green lime-rich ironstone, but over large areas, particularly at outcrop, it is almost completely oxidized to a soft brown limonitic ironstone with a much reduced lime content. In its unaltered state the iron is present as carbonates (siderite) and, to a lesser extent, as silicates (chamosite)—the former mineral usually occurring as ooliths and particles in the matrix, and the latter, whilst present as green ooliths in a few thin bands, usually occurs in the matrix. The carbonate content of the unaltered ore is often increased by the presence of fossil shells preserved in calcite and by the presence of thin bands of relatively pure limestone in the ore-bed.

Since percolating waters play a large part in the oxidation of the ironstone the ore tends to be preserved in an unaltered condition beneath the impervious clays of the Middle and Upper Lias. Complete preservation from oxidation is not wholly effected, since the pervious ironstone is usually saturated to considerable depths with waters derived from the excellent catchment area formed by the broad outcrop of the deposit in the west. Parts of the ore-bed respond differently to oxidizing agents, owing to an inherent variation in mineralogical content from the main part of the deposit. At the Frodingham Wing pit ([3]) the composition of the ironstone from top to bottom varies from 12·9 to 15·0 per cent moisture, 20·5 to 25·7 per cent iron, 1·4 to 11·8 per cent lime, 17·1 to 28·2 per cent insoluble matter, and 0·114 to 0·147 per cent sulphur. In addition the lateral variation throughout the same pit, within a distance of 1,100 yds., is from 8·30 to 10·1 per cent moisture, 21·4 to 23·3 per cent iron, 14·2 to 15·1 per cent lime, 13·3 to 17·4 per cent insoluble matter, and 0·540 to 0·718 per cent sulphur. Such vertical and lateral variations are partly due to selective oxidation but mainly to the great variation in the original lithology of this relatively shallow-water deposit of ironstone.

In the early days only the more completely oxidized parts of the Frodingham ironstone were smelted. This necessitated selective hand digging and prevented an early introduction of wholly-mechanized opencast methods. Usually the oxidized, and consequently richer, parts of the ironstone occurred at the top of the deposit, so that large areas of 'bottom stone' were left unmined and covered by dumps of hand-picked low-grade ironstone, rejected during the removal of the 'top bed'. In addition, a bed abnormally rich in sulphur occurs at a few places. It usually lies about 13 ft. from the top of the deposit and contains up to 3·975 per cent sulphur; it has to be removed by hand.

With improvement of blast-furnace technique it was found possible to smelt the hitherto-discarded lime-rich ironstones, which contained about 18 per cent iron, up to 30 per cent lime, and about 0·75 per cent sulphur. This was made possible by blending the ore with the siliceous Northampton ironstone—the latter acting as flux for the high-lime content of the Frodingham stone. Such developments immediately facilitated the introduction of mechanical excavators for digging the ore-bed, as well as for stripping overburden. This increased the output of ore from the pits, but led to further problems of grave concern to the furnacemen. Previously, the wholly-oxidized hand-loaded run-of-mine ore was composed of lumps rarely greater than 96 lb. in weight, consisting of 'boxstone' (or 'gingerbread') ironstone, with a very low proportion of fines below ⅜ in. The less-oxidized ore, which is now excavated and loaded mechanically, is much harder and has to be blasted to a greater extent than before, an operation which produces a high percentage of fines. The size of the lumps is now well over the old limiting figure of 96 lb. in weight.

An average figure to-day is about 11 lb. of explosive per 7 ton of stone, producing a maximum size of about 4 ft. by 3 ft. by 2 ft. in stone from under 25 to 30 ft. of cover.(3)

Owing to the variable nature of the component parts of the ore-bed and its thinly-bedded and closely-jointed structure it is rarely possible, even with a well-proportioned charge of explosive, to avoid the production of as much as 25 per cent fines in the run-of-mine ore. This problem has received attention from the Appleby Frodingham Ironstone Company and the results of their researches and the remedial measures adopted at the furnaces have been published in a comprehensive report to the Iron and Steel Institute.(3)

Considerable annual outputs of ore have been obtained from this somewhat restricted ironstone deposit. In 1939 the output was about 1,600,000 tons, but rose to 1,756,000 tons in 1943. Further, the run-of-mine ore was probably not in excess of 2,500 tons per foot-acre and the average thickness of the ironstone was about 20 ft. Thus, accepting the reserve figure published by the Kennet Committee (7) of 13,900 acres—i.e., 695,000,000 tons estimated on the basis quoted—it seems likely that the life of this field does not exceed 400 years. Of this estimate of reserves, however, probably only one-third is workable by opencast methods. Consequently, if these high outputs are to be maintained from this field, underground mining must be resorted to, in conjunction with the opencast mines, within the next 20 years or so.

The Marlstone Ironstone.—The Marlstone ironstone, as such, occupies a well-defined horizon at the top of the Middle Lias formation. In Lincolnshire it is about 7 to 12 ft. thick, where workable, and averages about 12 ft. in Leicestershire. Through Northamptonshire the ironstone is about 6 ft. thick, but increases to as much as 30 ft. in parts of Oxfordshire (Fig. 1). Throughout the main part of the field the outcrop of this ironstone is marked by a low escarpment and, owing to the gentle dip of the bed at about two to five degrees to the east, broad outcrops occur in many parts of Oxfordshire and Leicestershire. Locally the dip increases, or is reversed, by the occurrence of small folds and occasionally the ironstone is dislocated by small faults, some of which are due to the superficial movement of the beds on the underlying clays composing long gentle slopes or valley-sides. These rarely achieve the same prominence as they do in the Northampton Sand Ironstone field, and do not complicate the workings to the same extent (Fig. 3), a matter referred to later. Almost everywhere the Marlstone ironstone is free from complicated geological structures and thus is admirably suited to development by opencast methods.

Like the Frodingham ironstone the Marlstone stone is calcareous and, in its unoxidized state, greenish-grey in colour, often oolitic in texture, and sometimes shelly. At outcrop, and for a limited distance beneath the succeeding impervious shales and clays of the Upper Lias, it is usually thoroughly oxidized to a brown limonitic ironstone, and only in this state is it usually found to be of workable quality. In many places throughout the field it varies laterally into unworkable ferruginous shelly limestones and almost everywhere the ore is underlain by an almost equal thickness of calcareous sandstones and shelly limestones. These 'bottom beds' are sometimes quarried for building stone in conjunction with the development of the workable part of the Marlstone Rock Bed.

Where of workable quality the Marlstone ironstone varies from 20 to 28 per cent iron, 7 to 12 per cent silica, 4 to 12 per cent alumina, 1 to 15 per cent lime, and 10 to 25 per cent moisture. From experience it has been found that the ore-bed is almost always comparatively unoxidized, and hence unworkable, beneath 30 ft. to 40 ft. of the impervious cover of Upper Lias shales and clays. Where the ore-bed is waterlogged it is sometimes oxidized to a workable degree at slightly greater depths. Beneath over-burden largely composed of glacial gravels and boulder clay the ironstone is found in a workable condition at greater depths than 40 ft., since in such places the ore has been oxidized prior to the formation of the glacial deposits. For example, the deepest workings in the Marlstone ironstone occur at Eastwell, near Melton Mowbray, where the cover, which is over 40 ft. thick at the east end of the pit, is composed largely of boulder clay. It is also noteworthy that in this same district, as in other restricted parts of the field, the Marlstone ironstone contains iron, silica, and lime in such proportions as render it an almost self-fluxing ore. Elsewhere the greater part of the output of this ironstone is blended with the siliceous Northampton ironstone to somewhat the same degree as the Frodingham ore.

Originally, the Marlstone ironstone was quarried by hand, but now the ore and overburden are handled by mechanical excavators. As a consequence of the shallow depth at which the ore occurs over large parts of the field, and the remarkable simplicity of the geological structure of the area, large pits have been developed at many places, particularly in Oxfordshire. The largest outputs of this ore have been obtained from the Wroxton district, near Banbury, where the overburden rarely exceeds 10 ft. Extensive workings also occur in the Melton Mowbray district. At Harlaxton, Lincs., an opencast pit has been opened up recently in this iron-stone, which crops out from the base of the Lincoln Cliff. In conjunction with this, the same company is also working the Northampton ironstone, which crops out on the escarpment above. At Caythorpe the deposit is about 9 to 12 ft. thick, but further north, beyond the village of Leadenham, the deposit has not been found of workable thickness and quality.

The life of this field is somewhat comparable with that of the Frodingham ironstone field, but estimates at this stage will be seriously affected by the forthcoming publication of the results of the recent survey of the main parts of the Jurassic ironstone field by the Geological Survey. During this work new areas of the deposit, not accounted for in previously published estimates of reserves, have been found, whilst other areas, included in former figures, can now be shown to be unworkable.

The Northampton Ironstone.—This ore-bed occurs above the Upper Lias clays. In north Lincolnshire the deposit is unworkable, but at Greetwell, near Lincoln, some 9 to 12 ft. of ironstone has been mined underground, on the bord-and-pillar system, and in opencast pits fairly continuously up to 1938. This valuable deposit has now been exhausted. Further south towards Grantham the ore is variable in quality and thickness, but has been worked at a few places—such as Waddington (opencast), Coleby (underground), and Leadenham (opencast). South of Grantham there are extensive reserves of workable ore, which is often as much as 26 ft. thick. Throughout Rutland the deposit thins to 10 ft. in most parts, but in Northamptonshire, where the ore averages about 12 ft., as much as 20 ft. of ironstone has been worked in places. In the western and southern parts of this county, however, the ironstone deteriorates rapidly in quality and passes laterally into ferruginous sands, sandstones, and sandy limestones.

In its unaltered state the Northampton ironstone is greenish-grey in colour and contains a relatively high percentage of silica. Over large areas it has been oxidized to a brown limonitic ironstone, which is richer in iron but contains a proportionately lower lime-content and a higher concentration of silica. The ironstone worked at the present time averages about 28 to 35 per cent iron, 10 to 20 per cent silica, 4 to 12 per cent alumina, 1 to 12 per cent lime, and 8 to 22 per cent moisture.

As in the case of previously-mentioned deposits this ironstone has been oxidized to a considerable extent by percolating waters. The entry of this water is often governed by the permeability of the strata immediately overlying the ironstone. In addition it is affected by the circulation of underground waters collected at outcrop. Thus, in the choice of suitable sites for working ironstone such hydro-geological factors have to be considered. In places, where the ironstone is overlain by impervious clays and silts of the Lower Estuarine Series, it is often unoxidized at shallow depths. In other areas, where the ironstone is succeeded by sands and silts and loosely-jointed limestones, the ore is often thoroughly oxidized beneath considerable thicknesses of overburden. In general, the ironstone is less affected by oxidation as it is worked down dip in an easterly direction and a stage is reached where the green ironstone is of unworkable quality. In many places the partly-oxidized green ironstone is calcined in mounds, either within the confines of the opencast pit (Fig. 11), or at a point conveniently near a railway line.

Generally the Northampton ironstone dips gently towards the east, but in many places the dip is increased by small folds, and the bed dislocated by faults. Usually these geological structures are due to superficial movements known as 'cambering'.(5) The outcome of such cambering has been to produce fissures in the ironstone—known as 'gulls'—which are often filled with sands, clays, and silts derived from the overlying Lower Estuarine Series. Large cavities and open joints are also formed in the Lincolnshire limestone as a result of such movements (Fig. 3).

In addition to gulls these superficial movements give rise to the formation of closely-spaced dislocations known as 'dip-and-fault'

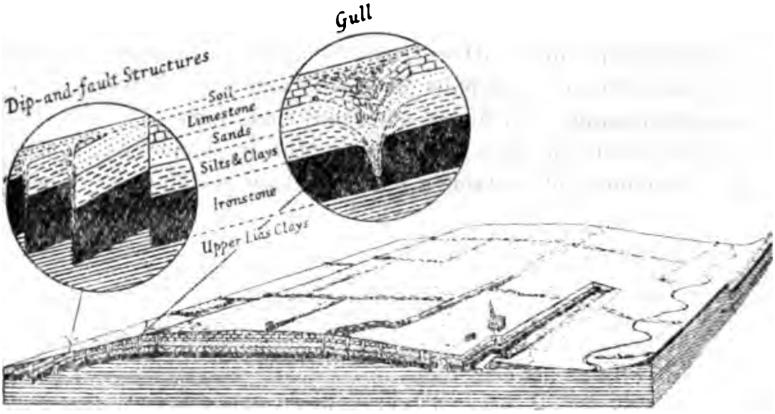

FIG. 3.—The relation of superficial disturbances to topography.

structures. These consist of a series of parallel steeply-inclined fractures between which the beds are tilted sharply downhill, causing an appreciable downthrow on the uphill side of the dislocations. A step-faulted arrangement of the beds is thus formed in which the downhill dip of the strata largely compensates the throw of the faults. Gulls are almost invariably developed along these lines of dislocation. The presence of these structures often obstructs the working of ironstone by opencast methods.

The estimates of reserves of Northampton ironstone already mentioned (7) cover 81,400 acres. Accepting this estimate it is evident that this extensive ore-bed (which averages 10 ft. thick) has a probable life of about 200 years if the 1943 output of 9,594,000 tons is to be maintained. A large part of these reserves of ironstone lies at depths beyond the range of present-day excavators and even now many of the opencast pits in the Northampton ironstone are rapidly approaching the limits of workability. Thus, there will be a strong tendency to replace opencast workings by underground mines (of which three are at present in production) if the Northampton ironstone is to maintain its present-day position as producer of more than half the total output from the Jurassic ironstone field (Fig. 2).

Geological Factors in Opencast Mining of Ironstone

From experience it is suggested that the following systematized approach to the planning and development of opencast sites results in a minimum expenditure of time and money in working ironstone and similar bedded deposits :

(1) Suitable sites are selected from the 6-in. geological maps.

(2) The selected site is investigated by examination of existing outcrops, followed by trial holes, or trenches, put down at points selected by the geologist to indicate the overall quality of the ironstone.

(3) If the deposit proves workable at this stage the site is further probed by a series of boreholes put down at carefully selected points to prove the maps of the geological survey, and to give additional information regarding the quality of the ironstone and the nature and thickness of the overburden. The level above ordnance datum should be taken of the top of each borehole.

(4) With this information *Isopachyte Plans*, showing the distribution of equal thicknesses of overburden, and *Contour Plans* showing the disposition of the ore-bed at depth should be constructed. These plans should be drawn to a scale equal to that to be employed in the construction of the lay-out of the opencast operation. The isopachyte plans will indicate the limits of workable thicknesses of overburden and will provide the basis for estimating the amount of discharged overburden involved in effecting the final restoration of the site. Contour plans are necessary for the top or bottom of the ore-bed, as the variations in gradient of the deposit frequently bear no resemblance to the topography of the site. Such plans will assist the engineer in formulating a scheme for draining the mine where necessary.

(5) Where the ironstone shows great variation in composition a plan showing the distribution of the principal chemical constituents of the ore-bed, obtained from the chemical analysis of the borehole and trial-hole samples, will assist in locating the workable parts of the deposit and estimating the available reserves of ore.

When this preliminary survey has been completed the engineer, in consultation with the geologist, is then in a position to formulate the lay-out and mode of development of the mine from the initial excavation to the final restoration of the land to agriculture. From the plans drawn up by the geologist it will be possible to select the most advantageous line for the initial gullet, the machinery necessary to deal with the type and thickness of overburden over the whole site, and to anticipate the obstacles imposed by geological disturbances. In addition, by adopting this orderly approach to the problem, it should always be possible to incorporate the restoration of the worked-out areas into the mining operation as a whole, and so effect it expeditiously.

Overburden.—In areas selected for opencast mining not only the thickness, but the composition, of the overburden must be considered, as upon this will depend the choice of excavators to be used, the depth to which the ore can be excavated, and the width of the excavation. Power shovels, which always excavate above the level at which they stand, are generally used for digging hard beds—such as, limestones, sandstones, and ironstone. Further, they must operate as far as is possible on a solid foundation, so that where the ironstone is of workable quality down to the level of the underlying Lias clays a thin layer of the ore has to be allowed to remain unworked as a foundation for the machine, since a shovel would undoubtedly become seriously bogged in the clays during wet weather.

The largest power shovel at present in use in the ironstone field for stripping limestone overburden weighs about 700 tons. It is mounted on four rail-bogies, and fitted with a bucket of 9- to 11-cu. yd. capacity, and discharges overburden at a distance of about 105 ft. This machine is also capable of dealing with as much as 55 ft. of overburden and excavates at a rate of about 300 yds. per hour. In some places the maximum thickness of overburden to be removed by power shovels has been seriously underestimated at the outset of the operation. This has resulted in either an expensive replacement of the machines by others of more adequate capacity, or with the equally expensive introduction of additional stripping machines of various types to deal with the overburden in excess of the capacity of the original machine (Fig. 4).

Sands, gravels, and clays can be removed with ease by a dragline excavator, which is also capable of digging loose beds of ironstone and limestone, as well as more compact beds if they have been well shattered by blasting beforehand. This machine has several advantages over the power shovel, but unfortunately its digging power, as yet, is not as great. The dragline, which normally digs to best advantage beneath the level at which it stands, is capable of discharging the excavated material at considerable distances away from the working face. Further, it possesses the ability to dig materials selectively. Thus it is able to remove and discharge soil and overburden separately, and a dextrous operator finds little difficulty in discharging them in a levelled condition on the worked-out areas. Draglines therefore are being used to an ever-increasing extent on sites where the restoration of the worked-out areas to agriculture forms part of the opencast operation (see Fig. 15, Plate I).

Dragline excavators are usually mounted on caterpillar tracks, but about five years ago 'Walking' draglines (Fig. 8 and Fig. 16, Plate II) were introduced into the field. The walking device replaces the usual caterpillar tracks and the machine is mounted on a hollow circular base, 20 to 30 ft. in diameter, which greatly increases the stability of the machine and enables it to carry a much longer boom than is normally fitted to caterpillar-mounted draglines. In addition the walking device makes it possible for the machine to move over soft ground, in which draglines were previously liable to become bogged. In this country the largest walking draglines in use are fitted with booms over 150 ft. long, but still larger machines, with a boom length of over 200 ft., are in use in the opencast coal workings of the U.S.A.

Tractor-drawn scrapers are capable of removing soft beds such as sands, shales, clays, and sometimes boulder clays (Fig. 4). The presence of limestone or sandstone in the overburden prevents them from completing the stripping operation, so that care has to be taken to ascertain whether such beds occur in the overburden if an opencast operation using tractor equipment is envisaged. Scrapers and bulldozers have been used extensively in recent years on the opencast coal sites of many parts of this country. To a much more limited degree they have also been employed in the ironstone field and it is probable they will find increasing application to this latter operation as the need for handling overburden in excess of the present equipment occurs, particularly where total restoration of the worked-out areas is enforced.

The Frodingham ironstone is normally succeeded by blue clays, gravels, and blown sands. In the past it was considered inadvisable to use dragline excavators standing on top of the overburden, as it was feared that the overburden would not withstand the load when working at critical angles of repose of the clays, etc., and

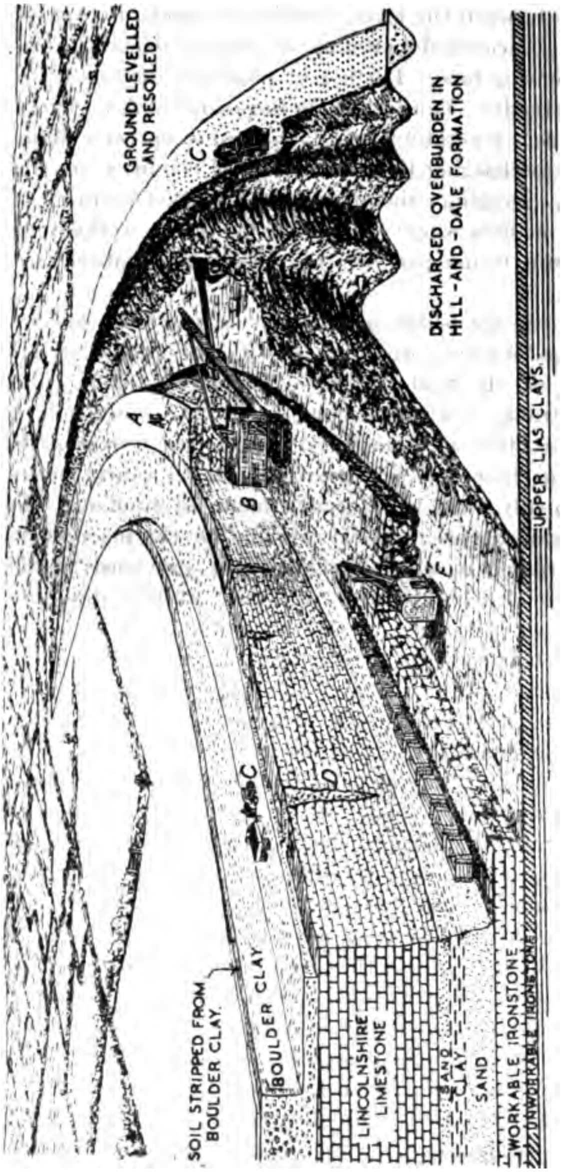

Fig. 4.

A : Drilling blast-holes for shattering limestone in advance of the power shovel (B) stripping overburden.

C : Tractor-drawn scraper removing boulder clay overburden in excess of the capacity of the power shovel (B), and depositing it in the 'dales', thus levelling the 'hill-and-dale' formed by the power shovel.

D : Cavities in the Lincolnshire limestone filled with boulder clay.

E : Rock-navvy loading ironstone into wagons running on tracks along the top of the bared ironstone.

that serious slips would occur. Such slips do occur in the somewhat similar cover-deposits of boulder clay which cover parts of the Corby, Northants., area. Thus, in the Scunthorpe district it was considered advisable to use long-jib power shovels working on a bared bench of ironstone, or endless-bucket excavators and belt-conveyors working from the top of the overburden, for stripping the Frodingham ironstone.

In late years this danger of collapse of overburden under load was averted by standing the dragline excavator on a bench cut at about 20 ft. from the top of the overburden. From this level the machine drags the upper part of the cover beds down towards itself, and discharges it, along with the material conjointly excavated from the underlying part of the cover. As a result of this modification in stripping technique the largest dragline excavator in the ironstone field operates in the Scunthorpe district baring Frodingham ironstone. This machine is fitted with a 2¼-cu. yd. bucket, a boom 155 ft. long, and is capable of discharging overburden at a distance of 150 ft. At the present time it is removing about 25 ft. of cover, but it is hoped to work to a depth of at least 50 ft. with the machine.

The Northampton ironstone is frequently bared by dragline excavators, but where the cover contains thick beds of hard Lincolnshire limestone, it has to be removed by long-jib power shovels. In some places smaller power shovels loading into skip transporters are used to maintain a wide pit. In a few places where the limestone has been sufficiently well-shattered by blasting beforehand it has proved possible to maintain a wide pit with large draglines of the walking type.

The gradient at which the overburden will stand without collapsing during the working of a site will depend upon the nature and thickness of the cover and the period of time that elapses between each traverse of the excavator. From experience solid beds of sandstone, limestone, and hard shales will stand at a gradient of 10 in 1, but if these beds are shattered by blasting, or loosened by earth-movements, forming joints and fissures, the slope is reduced to 7 in 1. When these beds are saturated with water, and subject to frost action, these gradients are greatly reduced. Soft beds—such as sands, gravels, and clays—will stand at a gradient of about 1 in 2. In large opencast pits, where the working face is deep and long, the stripping machine may only traverse the pit about once every three weeks or so. Under these conditions such estimates of gradient of the working face are reasonably accurate. Where the pit is smaller, so that the stripping machine traverses the working face more frequently, steeper gradients can be maintained, even in soft beds of clay, if these are not rendered plastic by seepages of ground water.

Geological Structure

The dip of the strata and the topography of the land largely determine the thickness of cover on ironstone within an area unaffected by faults. Thus the ideal is reached when the slope of the land approximates in direction and degree to the dip of the ore-bed. The presence of faults and folds naturally affect the amount of overburden present, and in addition they often seriously hinder the smooth working of an opencast pit, since mechanical excavators work to fullest advantage on level or evenly inclined surfaces.

The geological structure of the Frodingham ironstone is remarkably simple. The beds dip gently eastwards at rarely more than 2° and, apart from small folds, which usually extend from north to south, and faults which rarely downthrow the beds more than 10 ft., the ironstone preserves a very even surface over the greater part of the field. The geological structures affecting the Marlstone ironstone are equally simple, but in a few places this ore-bed is dislocated sharply as a result of superficial earth-movements. Thus, in both these fields the ironstone can be developed in pits of great length, throughout which the increase in the thickness of the overburden is small.

The geological structures affecting the Northampton ironstone are often complicated, and in many places the general eastward dip of the beds is not evident, since the ironstone often dips at considerable angles in various other directions as a result of the action of superficial earth-movements which have developed during valley formation (Fig. 8). These structures seriously obstruct the development of the opencast workings and in not a few cases have led to the abandonment of the site.

'Gulls' (Fig. 3) usually run parallel to the present land surface, or at right angles to the direction of maximum gradient of the ground. This fact is of considerable engineering value, as it influences the design of opencast pits in areas traversed by gulls. Apart from leading to the contamination of the already highly-siliceous ironstones, the presence of gulls in the ore causes considerable obstruction to the development of pits. In the past large areas of shallow ironstone have been abandoned owing to the presence of gulls filled with sands and clays, but now their mode of occurrence is known methods have been devised for working such areas.

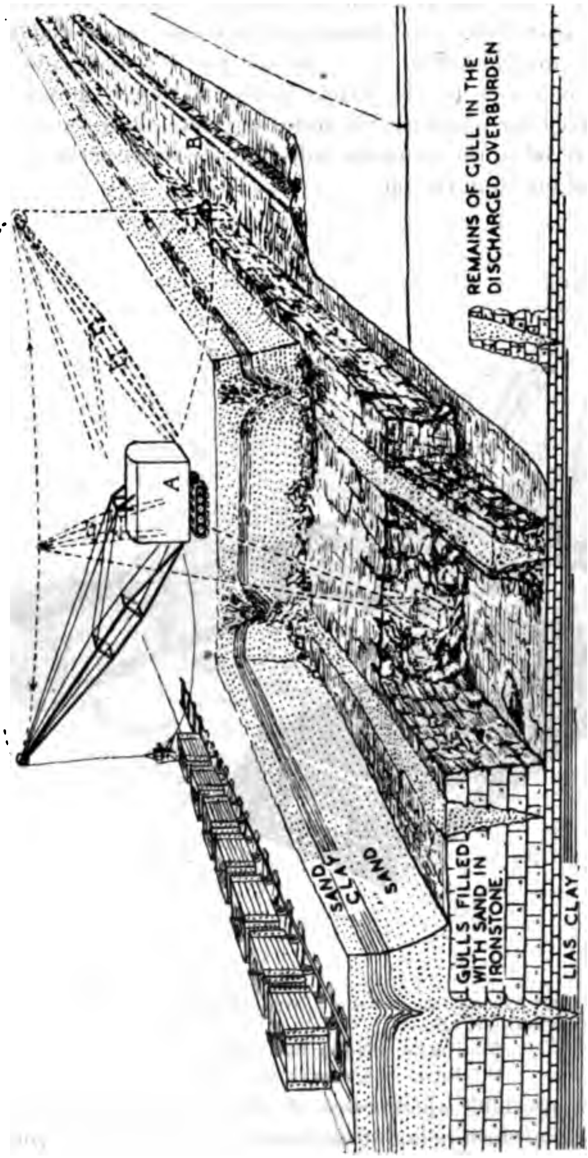

FIG. 5.—Dragline excavator (A) stripping and levelling overburden and removing and loading ironstone from between gulls (B).

In certain cases it has been found advantageous to open up the pit parallel with the direction of the gulls. Such a procedure is possible only when the ironstone is beneath shallow overburden, or in places where both overburden and ironstone can be removed by the same dragline (Fig. 5). As the pit develops, and the overburden increases to the extent where a power shovel has to be employed for removing the ironstone, these long barriers of siliceous material in the ironstone will seriously hamper the steady production of ore from the pit.

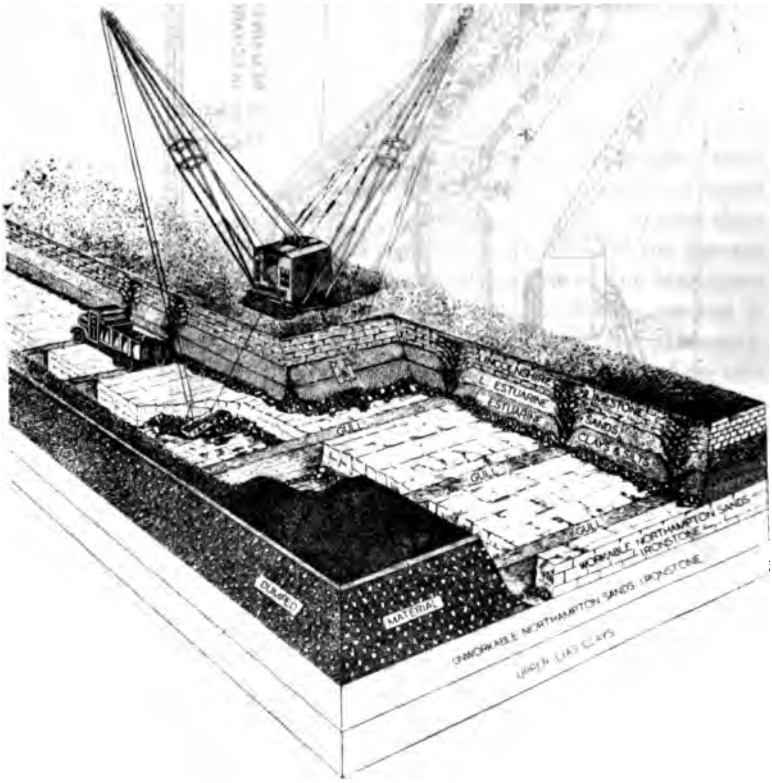

FIG. 6.—Dragline excavating ironstone from between gulls and restoring the worked-out areas.
A : Bucket digging ironstone.
B : Bucket digging overburden.

Beneath appreciable thicknesses of cover, and particularly in places where Lincolnshire limestone forms part of it, such a method is not too successful. Probably the best average mining conditions are obtained when the working face is aligned at right angles to the run of the gulls. Under shallow cover this operation differs slightly from the former case (compare Figs. 5 and 6), but it does not allow of the formation of a haulage way at the floor of the pit. Under deeper cover haulage problems are easily solved and great advantages are gained by running the pit at right angles to the line of the gulls (Fig. 7).

Where power shovels have to be employed an almost continuous output of ore can be maintained if this alignment is adopted. In this way the rock-digger excavating ironstone loads ore until that part of the gull exposed by the stripping machine is encountered. It pauses in removing ironstone and rips out the clays and sands discharging them clear of the haulage road, and then resumes its task of loading ore. Further, if considerable thicknesses of limestone occur in the overburden, it can be removed with considerably

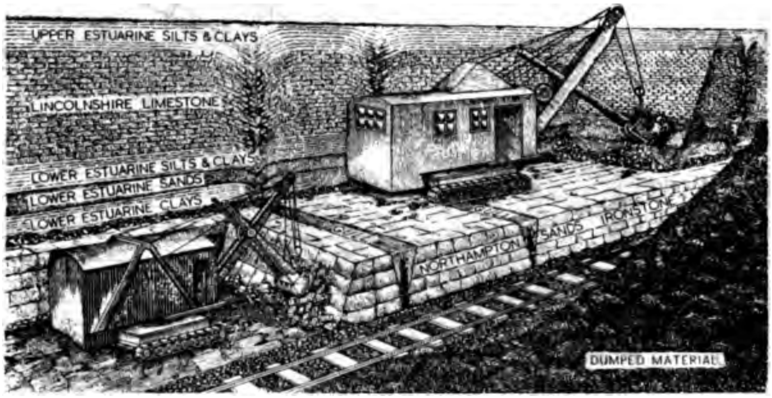

FIG. 7.—Power shovels excavating ironstone and overburden at right angles to the line of the gulls.

greater ease in this direction, since the beds have been loosened by the downhill cambering of the strata. In addition the shot-holes for blasting the limestones can be located so that they achieve the maximum effect by being placed in the unbroken limestone between the gulls.

The presence of dip-and-fault structures in an area to be opened up calls for close consideration. So far such structures have not been encountered in the Frodingham ironstone and it is unlikely that they will occur, as the ore-bed shows little sign of superficial movement on account of its occurrence in ground of low relief. Occasionally dip-and-fault structures occur in the Marlstone ironstone, but their effects are never as serious as those which affect the Northampton ironstone at many places. Two notable occurrences in the latter ore-bed are at Pitsford, near Brixworth, Northants.(5), and the disused workings at Wakerley, near Barrowden. In both places the workings are aligned at right angles to the strike of the faults. Such an alignment has certain advantages, one being that as the workings advance uphill into deeper cover they will cross at right angles the almost inevitable zone of ironstone traversed by gulls (Fig. 3). A disadvantage lies in the unevenness of the top of the ironstone, which can only be stripped successfully by a dragline excavator. Further, in order to maintain an even floor to the pit, the machine excavating ironstone must remove sections of uptilted strata underlying the workable ironstone (Fig. 8).

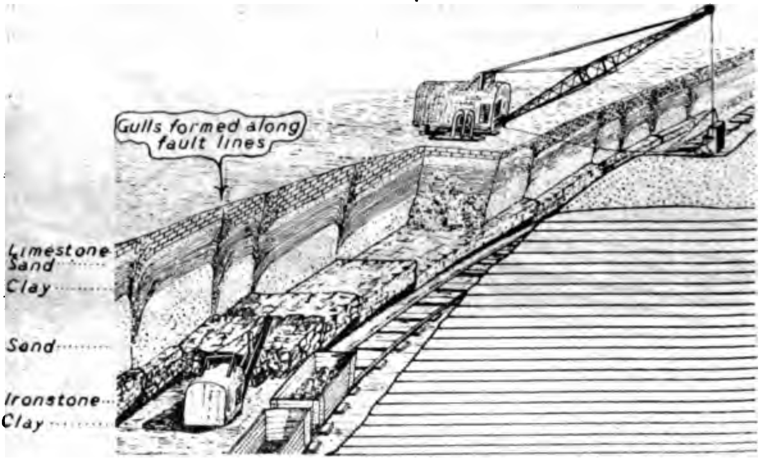

FIG. 8.—Walking dragline stripping overburden from ironstone dislocated by dip-and-fault structures, and a power shovel digging ore in a direction at right angles to the faults.

When the pit is aligned parallel to the dip-and-fault structures (Fig. 9) it is possible to maintain a more or less even floor to the workings. An additional advantage arises from being able to excavate both ironstone and overburden with greater ease, since these beds dip downhill and have been fractured by superficial movement. Further, the stripping of overburden is not necessarily confined to the dragline if the pit is aligned in this direction, for power shovels can operate from such dislocated benches of ironstone.

As such pits advance uphill the dip-and-fault structures diminish and the workings enter the zone where the ironstone is traversed by gulls (Fig. 3), which will extend parallel to the trend of the workings. This will offer no serious disadvantage in places where the overburden can be removed by a dragline, but where the overburden is heavy and power shovels are employed, the alignment of the workings will have to be re-orientated to traverse the gulls at right angles.

Valley Bulges.—These are also the result of superficial movement consequent upon valley formation. In their simplest form they consist of 'anticlinal uprises of the material composing the valley floor . . . as a simple or compound fold, or a series of discontinuous elongated domes'.(5) Valley bulges are responsible for areas of outcrop ironstone that have led mining engineers to undertake much fruitless exploration, as such occurrences of ore are always severely disturbed and of limited individual extent.

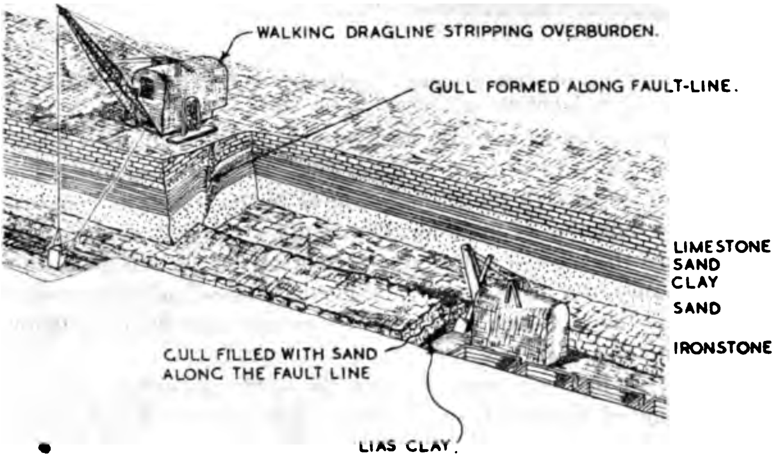

FIG. 9.—Walking dragline stripping overburden from ironstone dislocated by dip-and-fault structures in a pit lying parallel to the lines of dislocation.

Such areas should be avoided unless it has been established beyond doubt that the ironstone on either side of the valley is not at too great a depth, badly waterlogged, or severely affected by associated disturbances.

In addition to dislocations caused by superficial movements the ironstone deposits are traversed occasionally by normal faults. Where beds have been displaced by such structures, even to the small extent of 5 ft., they markedly influence the development of the site. In Fig. 10A the site has been opened up at right angles to two small faults. As a result difficulties are invoked. First, the digger stripping overburden has to leave a ramp of unexcavated overburden wherever it traverses the fault-line in order to move from one displaced part of the ironstone to the other. Similarly, the digger loading ore must be raised from one level to the other,

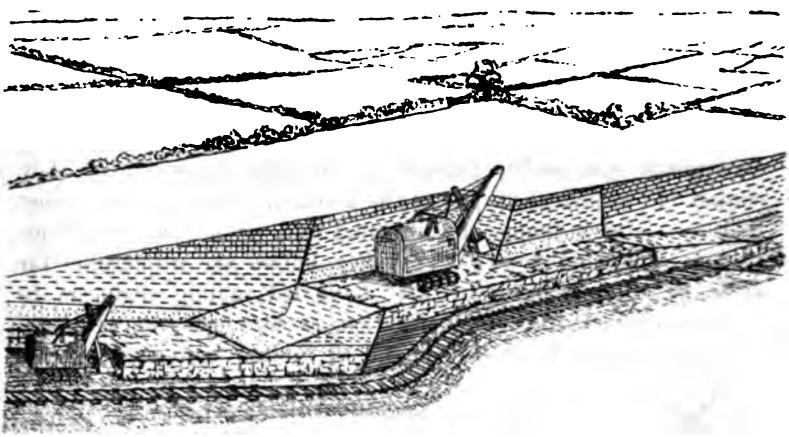

FIG. 10A.—Power shovels stripping overburden and excavating ironstone along a line at right angles to the faults.

or dropped down as the case may be, when it encounters a fault. As a result it will be forced to leave part of the ore-bed in the pit as it digs its way to the displaced level of the ironstone. Apart from the difficulty of maintaining continuous output, under such conditions the uneven nature of the floor of the pit will make the maintenance of a haulage system difficult. These difficulties recur each time the faults are encountered and interrupt the even tenor of the mining operation.

If the site had been designed with the workings running parallel to the line of the faults, or nearly so (Fig. 10B), such difficulties would not have occurred. On meeting the fault the workings are

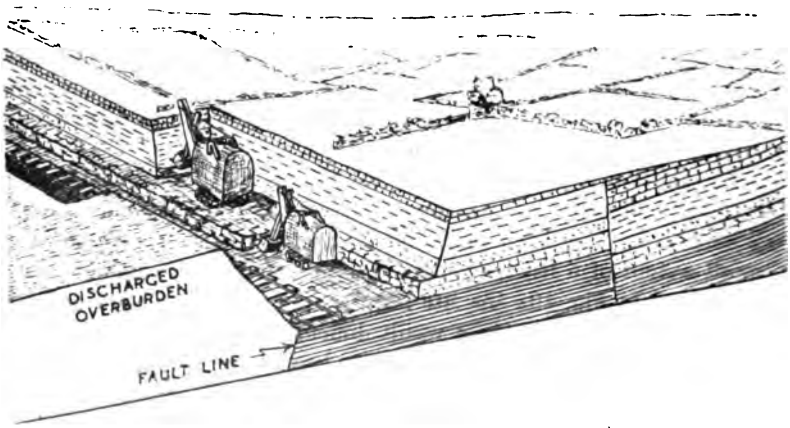

FIG. 10B.—Power shovels stripping overburden and excavating ironstone along a line parallel to the faults.

dropped or raised to the level of the displaced ironstone and the pit proceeds in a normal fashion.

The foregoing examples serve to illustrate the fundamental importance of a thorough geological examination of the site and the need for the production of comprehensive plans before the initial gullet is opened up. The ideal shape for the workings is semi-circular, with both ends open so that the ore-wagons can run out of the pit at either end. Such pits are not always possible to devise, but at Stainby the Stanton Ironworks Company has realized this ideal in an area where the cover on the ore bed, composed of unworkable ironstone, is thin (Fig. 11).* The dragline in

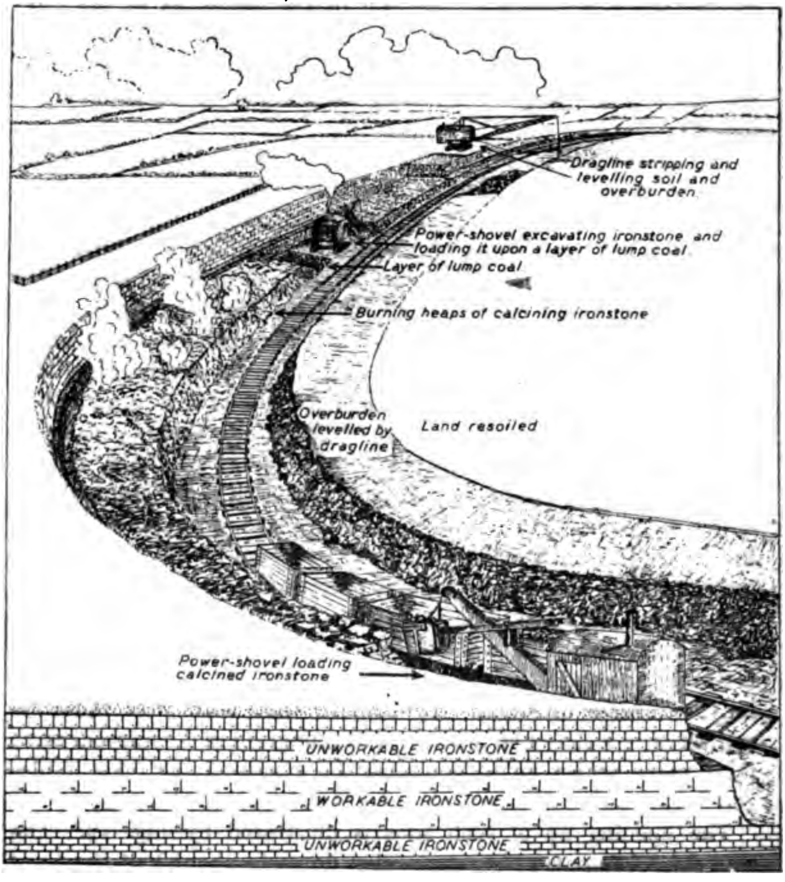

FIG. 11.

*An adaptation from 'The Quarries of Stanton Ironworks Company', by H. B. Hewlitt, 1935, Rochester.

the distance in Fig. 11 strips soil and overburden consecutively and discharges them in a levelled fashion on the worked-out ground. The soil is discharged on to a strip of overburden which has been previously levelled. Thus strip by strip the worked-out area is levelled and resoiled (see Fig. 15, Plate I).

The power shovel excavating ironstone removes the ore, then makes a half-circle turn and discharges it on to the layer of lump coal previously placed on the floor of the pit. At intervals small coal is added to the dump of ironstone formed behind the digger. At the other end of the pit the coal is ignited and the ore is progressively calcined as the fire burns forward along the heap. When calcination is complete at one end of the dump, as in the foreground, a power shovel loads the ore into wagons. The formation of such calcining heaps within the workings not only saves space, but reduces the number of times the ore has to be handled by diggers.

At Stainby the pit is concave-outwards in shape, but elsewhere it has been suggested that semi-circular pits which are convex-outwards in outline are more suitable.(⁸) However, the advantages are shared by both types of working, since they both provide a greater area for dumping overburden and a clear swing is afforded to each machine at either end of the pit. Thus the digger excavating ironstone is always able to work clear of the machine which strips overburden when it proceeds to work in the reverse direction. However, such ' open-end ' workings are not always practicable. For example, in the country east of Corby, Northants., many of the ironstone pits of the future will be forced to begin, not at outcrop, but in the heart of the deposit beneath considerable thicknesses of cover. This is due to the localized workable nature of the ore-bed. Thus, if the geological structure of the area is such that circularly shaped pits are not practicable, ' open- or fast-end ' box-cutting will have to be resorted to. Wherever practicable the ' fast-end ' type of pit will be avoided, since it is often difficult to maintain adequate space for the excavators to turn and excavate in the reverse direction. To achieve this, such sites must be subjected to a thorough geological examination and comprehensive mine plans drawn up to minimize the effects of the folds, faults and waterlogging on the maintenance of ' open-end ' workings.

The opening up of such sites is illustrated by Fig. 12. The shovel ' A ' removes the overburden down to about half the total thickness of cover on the ironstone. This gullet should be as wide as is practicable, but usually not less than 100 ft. The overburden is loaded into wagons, running on a temporary track, on one or other, or both sides, of the pit, and is transported to the point estimated as being the limit of the workings, to be used for restoration purposes. Shovel ‘ B ’, the largest of the machines, is used eventually to strip the full thickness of overburden.

At this stage it digs down to the level of the ironstone, but not for the whole width of the first gullet, and loads the overburden on to the platform formed by the remaining part of the gullet. Rock-navvy ‘ C ’ follows behind machine ‘ B ’ digging the exposed ironstone, but all the while leaving a retention bench on the down-dip side of the excavation.

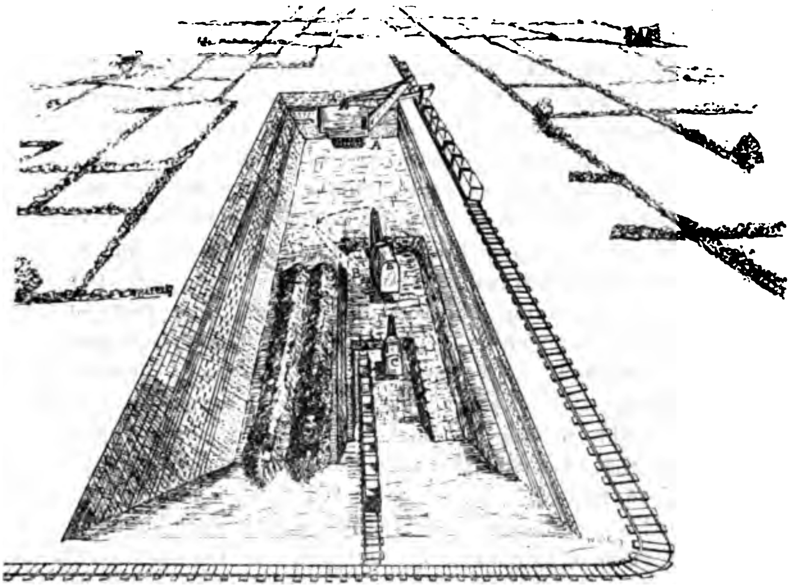

FIG. 12.—Rock-navvies opening up a site by box-cutting.

RESTORATION

In the past large areas of the ironstone field have been devastated by opencast mining, since in a number of places no attempt has been made to restore the land. Apart from the obvious wastage in valuable agricultural land when the overburden is left in rough ‘ hill-and-dale ’ formation, it forms a breeding ground for rabbits and other vermin and becomes a centre for the dissemination of weeds. While condemning this practice it should be borne in mind that in the days when this ground was being worked the power shovel was the only type of excavator available. The employment of hand-labour for this task would have been too heavy a charge to put on ironstone, the market value of which, even at the present time, is rarely more than five shillings per ton on rail. It is therefore necessary to incorporate the process of restoration into the mining operation as a whole (see Figs. 4, 5, 6, and 11), so that it can be discharged with the minimum expenditure of money and time.

Where power shovels are employed for excavating the overburden, they discharge it in 'hill-and-dale' formation (Fig. 4). The height of the hills is dependent on the type of excavator (or transporter) and the thickness and nature of the overburden. The more pronounced the hill-and-dale character becomes, the more expensive it is to level; therefore attempts should be made to vary the dumping points as frequently as possible so that material is continually being discharged into the 'dales'. Power shovels are incapable of taking part in the levelling of the discharged overburden and they are also unable to strip the soil prior to the excavation of the cover. Therefore, in all areas where they are employed care must be taken to remove, and store, the soil in a suitable place clear of the ground to be worked over. Where power shovels are indispensable it should not prove difficult to arrange for tractor-drawn scrapers to remove the soil strip by strip as the pit develops and then relay it over those parts of the worked-out areas which have been levelled by tractor equipment.

Where dragline excavators can be employed hill-and-dale formation can be avoided or reduced to a minimum. A dextrous operator is able to fling the bucket, at the point of discharge, in such a way that the material is spread over the previously excavated area (Fig. 15, Plate I). These machines are also capable of levelling each panel of overburden and layering it with soil obtained from the next piece of ground to be removed. In this way the final restoration of the land becomes an essential part of the mining operation and proceeds with the development of the pit.

Fig. 13 shows a working face (diagrammatically foreshortened) which has been developed in conjunction with the restoration of the land. The soil from Plot 1 was stripped and discharged into a heap located at Z. The overburden from Plot 1 was afterwards removed and dumped and levelled on the area 1A, but at the same time the excavator was removing the ironstone a little ahead of the discharged material (Fig. 6). The soil from Plot 2 was used to resoil the levelled overburden excavated from Plot 1, and the overburden in Plot 2 discharged into area 2A, which had by now been cleared of ironstone. In this way the dragline traversed the first strip to Plot 7, when it was placed at point X to resoil the overburden of Plot 7A with soil taken from Plot 8. The excavator

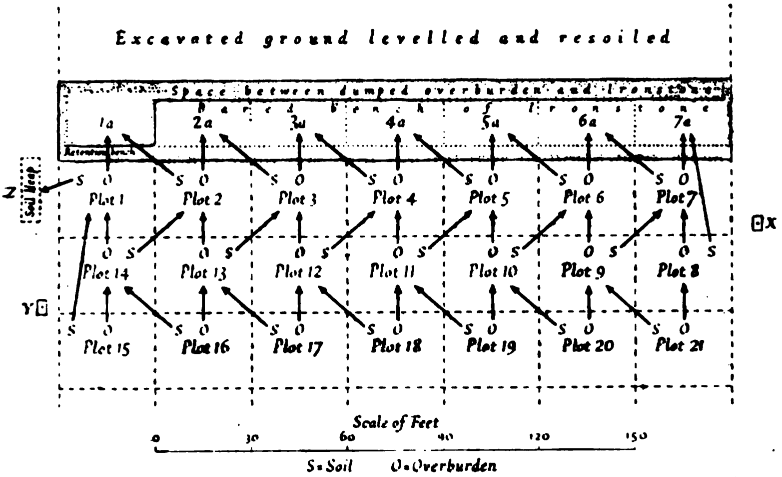

FIG. 13.

[*Reproduced by kind permission of the Royal Geographical Society.*]

then proceeded to re-traverse the pit in the opposite direction, digging ironstone, excavating, levelling and resoiling the overburden in the same way as before. The discharged overburden from the last plot of this strip was resoiled by placing the excavator at point Y to strip the soil from Plot 15. In this way the site was systematically exploited, and, at the end of the operation, the soil from Plot 1 was used for resoiling the last plot and the land was in a fit state to be returned to agriculture.

In pits where the stripping machine is unable to discharge the excavated overburden sufficiently distant from the bared ironstone,

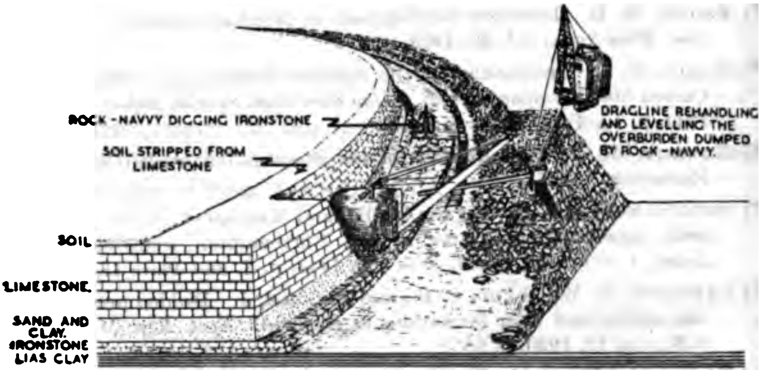

FIG. 14.

it is assisted either by transporters or by dragline excavators standing on the overburden dumped on the worked-out areas. Fig. 14 shows a dragline assisting a power shovel in piling up the overburden and so maintaining a good width to the pit; in addition this machine levels the overburden.

The incorporation of restoration into the development of opencast pits necessitates consideration of the quantity of material required to fill the final workings. From isopachyte plans, which show the variation in thickness of overburden, the position of the end-stage of the workings can be determined. The width and volume of the final workings can be estimated and sufficient material derived from stripping overburden can be transported to this point in sufficient quantity, not only to fill the final gullet, but to grade the worked-out area with the unworked ground.

Acknowledgments

The author's thanks are due to Mr. T. Eastwood, Assistant Director of H.M. Geological Survey, for facilities and encouragement during this investigation of the geological factors involved in the opencast mining of bedded deposits. To former colleagues of the Geological Survey and the mine managers of the ironstone companies the author is indebted for information and constructive criticism of new methods of planning and development devised during this survey of the ironstone field.

List of Selected References

(1) Anderson, W. Jurassic iron ores, Cleveland District. *Mem. Geol. Surv.* Wartime Pamphlet No. 23, 1942, p. 19, Table XVII.

(2) Beilby, R. B. Ironstone development in Northamptonshire. *Trans. Inst. Mine Surv.*, vol. 20, 1939, p. 3.

(3) Elliot, G. D. Ironmaking at the Appleby-Frodingham works of the United Steel Companies, Ltd. *Iron Steel Inst. Special Report* No. 30, 1944, p. 10, Table III, and p. 15.

(4) Hewlitt, H. B. The quarries of the Stanton Ironworks Company, Rochester. 1935.

(5) Hollingworth, S. E., Taylor, J. H., and Kellaway, G. A. Large-scale superficial structures in the Northampton ironstone field. *Quart. J. Geol. Soc., Lond.*, vol. 100, 1944, p. 3, and Plate II, Fig. 2.

(6) Lamplugh, G. W., Wedd, C. B., and Pringle, J. Bedded ores of the lias, oolites and later formations in England. *Spec. Rep. Min. Res. G.B.*, vol. 12, 1920, p. 9.

(7) Ministry of Health. Kennet Committee report on the restoration of land affected by iron ore working. H.M.S.O., 1939.

ADDITIONAL BIBLIOGRAPHY

BARNES, W. Excavating Machinery. Benn (London, 1928).

BARNES, W. The Dragline excavator. *J. Instn. Civ. Engrs.*, vol. 14, March, 1940, pp. 8–54.

BARNES, W. Methods of excavation work at home and abroad. Dugald Clerk Lecture, *J. Instn. Civ. Engrs.*, vol. 15, January, 1941, pp. 159–96.

BARNES, W. Excavating machinery in the ironstone field. *Engineer*, vol. 174, August 7–28, 1942.

EVANS, W. D. The opencast mining of ironstone and coal. *Geogr. J.*, vol. 104, 1944, Nos. 3 and 4.

HALLIMOND, A. F., ENNOS, F. R., and SUTCLIFFE, R. Iron ores: bedded ores of England and Wales: petrography and chemistry. *Spec. Rep. Min. Res. G.B.*, vol. 29, 1925.

Memorandum dealing with working of stratified ironstone of the Jurassic system of England. National Council of Associated Iron Ore Producers. (Copyright reserved).

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Phot. W. Barnes.

FIG. 15.—Caterpillar-mounted dragline excavating ironstone and stripping and levelling overburden.

Phot. W. Barnes.

FIG. 16.—Walking dragline and power shovels with transporter.

Subject to revision.] [*Paper issued on September* 12*th*, 1946, *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, Burlington House, Piccadilly, London, W.* 1, *on Thursday, November* 21*st*, 1946, *at* 5.0 *o'clock p.m.*

Notes on the Estimation of Tonnage and Grade of Some Chromite Dumps.

By N. W. Wilson, *Associate.*

Introduction

In the South African winter of 1944 the writer, as referee, estimated the tonnage and grade of 40 dumps of friable chromite ore. These dumps ranged in size from less than 200 long tons to 26,000 long tons. As little detailed information is available in technical literature about dump surveys and samplings, descriptions of the methods used and of the results achieved may be of some value.

Tonnage Estimates

Estimation of dump tonnages is in two stages: Of these the first is to determine the volume and the second is to find the average number of cubic feet of the undisturbed mineral as it lies in the dump that weigh one ton; in short, the tonnage factor. If the calculated tonnages are to check closely with the railway weights, as they should for dumps of a valuable mineral, factors concocted from theoretical ingredients or determined in artificial conditions are useless.

Determination of Volume.—As most of the dumps measured were irregularly shaped, level floors being exceptional, it was decided to contour them tacheometrically at intervals of 1 ft., measure the area of each contour (or horizontal section) with the planimeter, and compute the volume by either the prismoidal formula or the end area formula, whichever should be applicable. Tacheometry was done in the usual way, but to lessen the number of 'spots' needed the dump surfaces were smoothed out by pick and shovel before the survey began. 'Spots' were placed at changes of slope and along the peripheries of dumps. Contours were then interpolated between the 'spots' on plans to the scale of 1/100. 'Spots' over 100 ft. from the tacheometer were co-ordinated in order to reduce plotting errors. For co-ordination the slide rule and four-figure tables of natural sines and cosines were used. At the larger dumps theodolite and steel tape traverses were run to fix points from which the 'spots' might be observed. The elevations of these points were levelled.

The prismoidal formula applies except at the tops and bottoms of dumps. It is:

$$\tfrac{1}{6}(a_1 + 4M + a_2) \times 2h.$$

where, for the purposes of this paper:

a_1 = Area of upper contour in three adjacent contours.
M = " mid " " " " "
a_2 = " lower " " " " "
h = Contour interval (1 ft.).

The application of the end area formula need not be detailed here.

Calculations are straightforward and difficulties only arise at the top and bottom of a dump. At the top of a dump there will be 'spots' slightly higher than the topmost contour. If there is a pronounced peak the volume may be approximated to a pyramid and the formula $\tfrac{1}{3}Ah$ used, 'A' being the area of the topmost contour and 'h' the elevation of the peak above the contour. Alternatively, should there be a large flat area above the topmost contour, with no decided peak, a contour should be interpolated at an interval less than 1 ft. so that the volume may be calculated by the end area formula.

At the bottom of a dump with a sloping floor there will be a prismoid with sides that are not parallel. For rapid calculation this solid is divided into three prismoids, of which, in cross-section, one is three-sided and two are four-sided. The volume of the three-sided prism is computed by multiplying its mean plan area by half its vertical height:

$$\frac{a_1 + a_2}{2} \times \frac{h_1}{2}$$

To compute the volume of four-sided prism No. 1 its mean plan area is multiplied by its mean vertical height:

$$\frac{a_3 + a_4}{2} \times \frac{h_2 + h_3}{2}$$

Compared with the total volume of the dump the volumes of the solids calculated by these approximate formulae (the volume of the three-sided figure is about 3 per cent greater than it should be) are small and the error introduced is negligible. The volume of four-sided prism No. 2 is calculated by the end area formula.

Commonly the floors of dumps were regular enough or the dumps were sufficiently small to allow contours of the ground below the dumps to be interpolated between 'spots' on the dump peripheries. However, there were three dumps (C, D, and E) in which ground

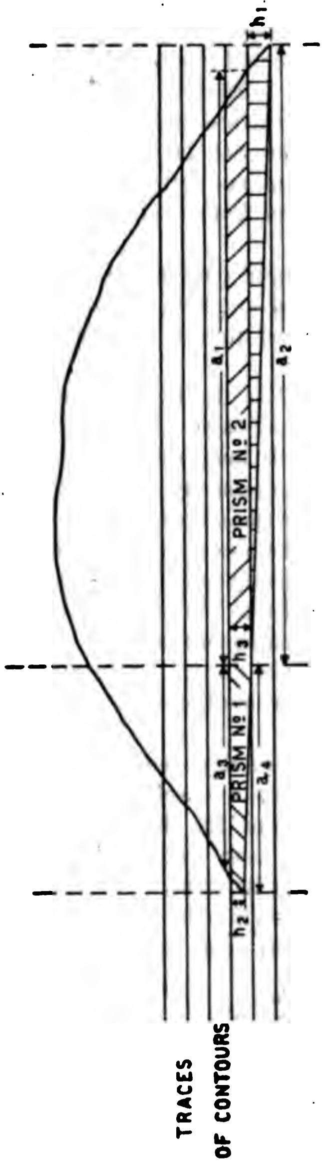

Fig. 1.—Cross section of dump showing prismoids at base.

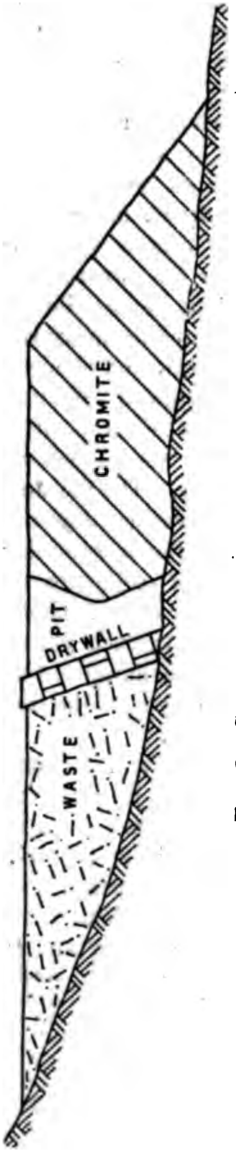

Fig. 2.—Cross section of dump (not to scale).

contours had to be defined by driving pipes vertically downwards from surveyed points until the ground was reached. Sloping buried drywalls built on the upper sides of the three dumps to separate them from waste complicated matters. To measure the slopes, pits were sunk in the chromite at equal intervals along the walls; the walls formed the foot-walls of these pits. At each pit the level of the intersection of the dump surface and the wall was fixed by a 'spot' so that the side of the wall facing the dump might be contoured. Both wall-contours and ground contours were drawn in colour to distinguish them from contours on the dump surfaces, which were in black.

How the owner's engineers check-surveyed dumps B, C, D, and E might be described here. The method is quick, accurate and well suited for measuring long prism-shaped dumps. It eliminates draughting and planimetric errors.

A rectangle with its long sides parallel to the long axis of the dump is laid out with theodolite and steel tape. Nails marking stations are aligned precisely along both long sides at regular intervals. The theodolite is then set up successively at each station so that vertical angles and slope distances (measured with the steel tape to the nearest tenth of a foot) may be observed along vertical planes of section passing through opposite pairs of stations (one on either side of the dump) at right angles to the long axis of the dump. In each plane of section a measurement is made to the foot of the dump and then to each change of slope. Extra sections are surveyed at the ends of the dump or where the surface is much deformed.

Vertical and horizontal components of the slope lengths are then calculated logarithmically and plotted as a check on a rough cross-section. To simplify the calculations the dump (and the cross-sections) is divided longitudinally into two parts by an imaginary vertical plane passing through its crest. The areas of the two portions into which each cross-section is split are calculated separately and then added together. An example follows:

Part Lamina No.	*Area No.*	*Working*	*Area, Sq. ft.*
S280	1	32·5 × 4·14	134·55
		½ × 7·0 × 4·14	14·49
	2	22·8 × 2·82	64·30
		½ × 9·7 × 2·82	13·68
	3	13·7 × 3·59	49·18
		½ × 9·1 × 3·59	16·33
	4	½ × 13·7 × 7·98	54·66
N280	5	5·3 × 20·83	110·40
	6	½ × 32·8 × 20·92	343·09
			800·68

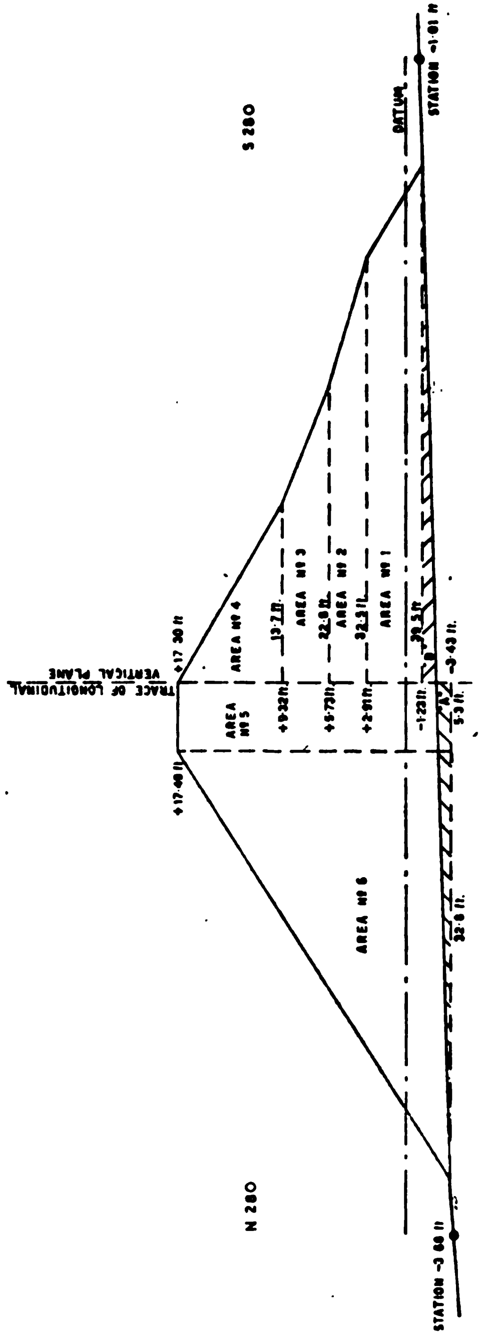

FIG. 3.—Calculation of area of lamina.

This quick but approximate calculation may, *when necessary*, be corrected to absolute accuracy by subtracting from the north part of the lamina the area of triangle A, or:

$$\tfrac{1}{2} \times 38{\cdot}1 \times 38{\cdot}1 \times \frac{2{\cdot}2}{77{\cdot}6}$$

and adding to the south part of the lamina the area of the triangle B, or:

$$\tfrac{1}{2} \times 39{\cdot}5 \times 39{\cdot}5 \times \frac{2{\cdot}2}{77{\cdot}6}$$

which is equivalent to adding:

$$\tfrac{1}{2} \times (39{\cdot}5^2 - 38{\cdot}1^2) \times \frac{2{\cdot}2}{77{\cdot}6}$$

or: $1{\cdot}5$ sq. ft.

The correct total area of lamina 280 is thus 802·2 sq. ft. From the areas of the laminae and the horizontal distance between them the volume is worked out by the prismoidal formula.

Evaluation of Tonnage Factor

The perfect method of getting the tonnage factor of a material would be to make a hole in it, weigh the material removed from the hole and determine the volume it occupied in place by filling the hole with a measured quantity of water. Unfortunately, for dumps, which are usually porous, this method must be modified. The writer's modification was to use maize instead of water. Maize does not pack appreciably and does not seep away through porosities; on the other hand, the grains are small enough to fill cavities in the walls of the hole into which they are poured. For this purpose, therefore, maize has the advantages of a liquid without its disadvantages.

In detail, the tonnage factor was found by:

(1) Weighing π cu. ft. of maize (a measure with a radius and depth of 1 ft. is easy and cheap to make). This weight varies slightly for maize from different sources.

(2) Digging a pit six or more feet square at the top to a depth which depended upon the depth of the dump. Usually several pits of differing depths were dug in each dump to ascertain if the degree to which the mineral was compacted changed appreciably.

(3) Digging in the bottom of each pit a hole from 1 to 3 cu. ft. in volume. All the chromite taken from this hole was sacked immediately to prevent drying. The hole was then filled with a weighed quantity of maize. Lastly, the chromite that had been sacked was weighed.

Knowing the average weight of 1 cu. ft. of the maize that was used the volume of the hole from which the chromite had been taken could be calculated and thence the weight of chromite per cubic foot and the number of cubic feet per long ton could be deduced. Some tonnage factors are given in Table I.

TABLE I

LONG TONNAGE FACTORS

mp o.	*Long Tonnage*	*Depth of Pit (Ft.)*	*Volume of Sample (Cu. ft.)*	*Long Tonnage Factor*	*Mean Long Tonnage Factor*	*Long Tonnage Factor Adopted*	*Remarks*
A	6,553	4	2·07	11·0	—	—	Dump composed of chromite from several sources, dumped at intervals of some months with different degrees of compactness. Height of dump, 14 ft.
	—	5	2·42	13·0	—	—	
	—	4	2·18	12·0	—	—	
	—	6	3·00	10·8	—	—	
	—	6·5	2·66	11·2	—	—	
	—	6	2·42	11·6	11·6	11·6	
G	565	6	2·90	11·9	11·9	11·9	Dumps composed of chromite from a single source.
H	557	6	1·98	11·8	11·8	11·9	
I	639	6	2·46	12·1	12·1	11·9	Maximum height of dumps, 8 ft.
B	26,041	4	1·74	11·1	—	—	Dump was composed of chromite from a single source. Tonnage factors of the order of 11 represent a shell beneath the dump crest which had been compacted by lorry traffic. A weighted mean was used, but was unsatisfactory. Height of dump, 21 ft.
	—	5	2·36	11·6	—	—	
	—	6	2·15	12·0	—	—	
	—	4	2·06	11·1	—	—	
	—	4	2·20	12·0	—	—	
	—	9	2·61	12·0	—	—	
	—	4	2·65	12·1	—	—	
	—	5	1·83	11·0	—	—	
	—	10	1·83	12·2	—	—	
	—	4	2·43	11·0	11·6	12·0	
C	7,573	4	2·08	11·5	—	—	Chromite from same source as that composing Dump B. Height of dump C, 13 ft.
	—	8	1·18	11·5	—	—	
	—	4	1·69	12·1	11·7	11·7	
D	193	4	1·84	11·7	11·7	11·7	Height of dump D, 8 ft.
E	13,182	4	2·01	11·7	—	—	Height of dump E, 17 ft.
	—	4	1·87	12·7	—	—	
	—	6	1·47	12·1	12·2	12·2	

To deal with thoroughly dry ore other means would have been devised, but luckily when freshly dug the sides of excavations were damp and stood well. Most dumps consisted of fines mixed with lumps up to 10 in. in major diameter. In compensation for these lumps, with their higher density, holes were made as big as time and the tendency of the sides to crumble would permit. The number of pits dug in a dump depended upon :

(1) Its tonnage,
(2) how many sources the ore came from, and
(3) costs.

Without reason to the contrary the arithmetic mean of the results from the pits was accepted as the tonnage factor for the dump.

At dump B, which had been tipped from lorries running along its crest, the tonnage factor was 11·0 in a crust of mineral 4 to 5 ft. thick below the lorry tracks. Beneath this crust, at depths of 8 to 9 ft., the factor rose to 12. Lower still, it must have fallen again to less than 11 judging by the average factor calculated from the cubic footage of the dump and the railway weight. Immediately before the dump was measured a small percentage of hard and lumpy ore was sorted out of one half of it. The friable ore that remained was then shovelled back into position. These operations caused the density to vary unpredictably and increased the difficulty of selecting a representative tonnage factor.

As a check upon the displacement tonnage factors, factors were twice determined for loose and tamped ore by weighing: First, a measure filled with loose ore, and, secondly, a measure filled with tamped ore, the volume of both measures being π cu. ft. The results are set out in Table II; it is interesting to compare them with the average displacement factors.

TABLE II

Dump No.	*Mean Displacement Tonnage Factor*	*Tonnage Factor, Loose Ore*	*Tonnage Factor, Tamped Ore*	*Mean of Columns 3 and 4*
(1)	(2)	(3)	(4)	(5)
1	12·5	14·95	11·85	13·4
2	11·7	13·5	10·7	12·1

At the first dump the ore in the measure was tamped after the measure had been filled. At the second dump successive 3-in. layers of ore were tamped as they were filled into the measure.

Discussion of Results of Measurements

In surveying the volumes of dumps C, D, and E (Tables III and IV) the owner's engineers were deliberately, and for their purposes, rightly conservative. Their figures for these dumps may therefore to some extent be discounted. At dumps A, B, and G, and H and I, where no reserves are known to have been made in the surveys or the calculations, the divergencies of the two sets of

TABLE III

RESULTS ACHIEVED

1	2	3		4	5	6	7	8	9	10	11
	Wilson's Figures					*Owner's Figure*					
Dump No.	*Volume (Cu. ft.)*	*Computed Long Tonnage Factor*		*Computed Weight (Long tons)*	*Total Computed Weight (Long tons)*	*Volume (Cu. ft.)*	*Mean Volume (Cu. ft.)*	*Percentage Divergence of Column 2 from Column 7*	*Railway Weight (Long tons)*	*Column 2 divided by Column 9*	*Column 5 as Percentage of Column 9*
		Factor Adopted	*Mean Factor*								
A	76,013	11·6	—	6,553	6,553	79,900	77,950	− 2·5	6,915·5	11·0	94·8
G	6,722	11·9	11·9	565	—	—	—	—	—	—	—
H	6,626	11·9	11·8	557	—	—	—	—	—	—	—
I	7,600	11·9	12·1	639	—	—	—	—	—	—	—
	20,948	11·9	—	1,761	1,761	—	—	—	1,795	11·7	98·1
					8,314				8,710·5		95·4
B	312,494	12·0	11·6	26,041	—	309,035	310,764	+ 0·6	28,356	11·0	91·8
C	88,605	11·7	11·7	7,573	—	82,060	—	—	—	—	—
D	2,259	11·7	11·7	193	—	4,550	—	—	—	—	—
E	160,818	12·2	12·2	13,182	—	155,612	—	—	—	—	—
	564,176	—	—	46,989	—	551,257	557,716	+ 1·2	—	—	—

TABLE IV

COMPUTED TONNAGE FACTORS COMPARED WITH PROBABLE TRUE TONNAGE FACTORS

Dump No.	Computed Long Tonnage Factor		Probable True Value of Long Tonnage Factor	Percentage Divergence of Computed Long Tonnage Factor ADOPTED from Probable True Value	Percentage Divergence of MEAN Computed Long Tonnage Factor from Probable True Value	Remarks
	Factor Adopted	*Mean Factor*				
A	11·6	11·6	11·0	+ 5·5	+ 5·5	Maximum height of dump, 14 ft.
G, H, and I	11·9	11·9	11·7	+ 1·7	+ 1·7	" " " 8 ft.
B	12·0	11·6	11·0	+ 9·1	+ 5·5	" " " 21 ft.

figures from the arithmetic mean of the volumes do not exceed 2·5 per cent and for dump B, which was reasonably regular, are only 0·6 per cent.

Table IV shows that the average computed long tonnage factors were too big at all those dumps of which the probable true factors are known. As factors for chromite from the middles of the two higher dumps (A and B) near ground level could not be included in the figures upon which the averages were based, the differences between the computed long tonnage factors and the probable true values at these dumps may perhaps be attributed to compaction.

From the admittedly scanty statistics given the writer deduces: First, that the volumes even of irregular dumps may be fixed up to nine times more accurately than their tonnage factors; secondly, that in choosing tonnage factors engineers should be biased in favour of the smallest figures obtained.

Sampling

The specifications called for a chemical grade of chromite, which, to avoid penalty, had to contain not more than 5 per cent SiO_2 or less than 44 per cent Cr_2O_3. At a meeting with the chromite producers it was agreed that one sample should represent from 500 to 1,000 tons of ore. In practice, as most of the dumps were small and came from a number of sources at least, one sample had to be taken from each dump. The weight of ore represented by a sample thus became, on the average, only 450 tons.

Weights of samples taken ranged from 1,000 lb. to 2,000 lb., according to the maximum size of the lumps and the depth of the dump. Each sample was composed of chromite taken from vertical grooves in not less than two pits, the weight taken in each pit depending upon the depth of the groove. Pits were sunk to ground level and were spaced equidistantly, an arrangement that departed little from the ideal spacing according to equal volumes, except in wedge-shaped dumps on sloping ground. Had wedge-shaped dumps with marginal values been discovered the intention was to survey the positions of the pits that had been sunk and weight the samples from these pits proportionally to the volumes they stood for.

Assays of samples taken from different parts of the same dump seldom varied much. Thus in a 270-ton dump (suspected to be below grade) assays of samples taken from three different pits were as given in Table V. Again, in a dump weighing 1,080 tons, samples from two separate sets of pits gave the assays shown in Table VI. However, both these dumps were at a railway siding

TABLE V

Sample No.	*Cr_2O_3, per cent*	*SiO_2, per cent*
1	42·40	5·66
2	43·00	5·56
3	42·75	5·34

TABLE VI

Sample No.	*Cr_2O_3, per cent*	*SiO_2, per cent*
1	44·35	1·58
2	44·60	2·06

and consisted of ore that had been transported by lorry and shovelled into place. During loading and off-loading some mixing would inevitably have occurred. As the dumps were flat lumps did not segregate much at the bottom.

At some of the mines the dumps were still in their original positions at the mouths of the incline shafts from which the ore was hoisted. These dumps had heights up to 9 ft. and had been tipped from side-tipping cars. The tops were thus narrow and the bases broad and conditions favoured an increase in the proportion of lumps downwards. Like those at the sidings, these dumps were sampled by pitting. Theoretically each groove should have been sectionalized, so that the assays of the sections might be weighted by the widths of dump they represented. Instead of this long and costly procedure (an assay for chromium costs 15s.) only a single sample was taken from each groove, but, as a check, grabs of fines and lumps were taken respectively from the tops and bottoms of dumps. Had the grade of the ore in the dumps been below the lowest penalty-free grade according to the pit sampling, and should the assays of the grabs of fine and lumpy ore have differed considerably, the system of sampling desirable on theoretical grounds might have been adopted. A comparison of the results of pit and grab sampling is given in Table VII. Each of these samples was taken from two pits. The assays of grab samples of fines and

TABLE VII

Pit Sample No.	Cr_2O_3, *per cent*	SiO_2, *per cent*
1	45·84	4·60
2	45·20	4·86
Mean	45·52	4·73

lumps from this dump were as given in Table VIII. If the grade of the ore from the pit samples had been marginal in this example the big difference between the assays of the fines and lumps would have been a warning to do more pitting and, possibly, to sectionalize and weight groove samples.

TABLE VIII

Grab Sample No.	Cr_2O_3, *per cent*	SiO_2, *per cent*
1	44·95	5·92 (Fines)
2	48·90	1·30 (Lumps)
Mean	46·92	3·61

To reduce samples to reasonable bulk for transport, they were:

(1) Screened through a 1-in. screen, the oversize being crushed to pass 1-in. mesh and mixed with the undersize.

(2) Coned and quartered; for quartering, as no mixing cone could be got, the ore was poured from shovels on to a cross 18 in. high and 30 in. in diameter made of ¼-in. steel plate. One opposite pair of quadrants tended to be fuller of ore at any given time than the intervening pair. This tendency was counteracted by rotating the shovellers round the ring into which the sample was drawn out. The customary practice of accepting ore from one opposite pair of quadrants as the sample was adopted.

(3) Screened through a ¼-in. screen, the oversize being crushed to pass ¼-in. mesh and mixed with the undersize.

(4) Split successively in a Jones riffler to a weight of about 50 lb.

Each split followed a thorough mixing. Both the last splits were kept, one as an original and one as a duplicate.

All crushing was by pestle and mortar. As lumps were friable and disintegrated easily the time taken to crush the *plus* 1-in. part of a sample to *minus* 1-in. was seldom more than 90 minutes. Dry chromite ore runs easily and has an angle of repose of less than 35°, consequently the quadrants of the sample had to be separated by a cross during quartering. For a sampling floor sheets of ¼-in. steel plate were laid down. A foundation of crushed chromite diminished contamination of samples with dust and soil.

An arbitrary figure of 1½ per cent for moisture content of the ore in the dumps was agreed upon between buyers and sellers; this obviated moisture determinations.

Acknowledgment.—The writer is grateful to the firm for which the estimates were made and its clients for permission to publish this paper and for the figures with which they have supplied him. Mr. Donald Gill's constructive criticisms of the rough draft of the paper were very helpful and were much appreciated.

*** *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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[redacted]

THE INSTITUTION OF MINING AND METALLURGY

FIRST ORDINARY GENERAL MEETING

OF THE

FIFTY-SIXTH SESSION

Held in the Rooms of the Geological Society, Burlington House, Piccadilly, London, W. 1,

ON

Thursday, June 20th, 1946.

Mr. G. F. LAYCOCK, *President*. in the Chair.

DISCUSSION

ON

Sandfilling at Mufulira.

By A. C. TURTON.

The President said the paper submitted for discussion that evening was on 'Sandfilling at Mufulira'. It had been a long time since they had heard anything about sandfilling methods at these meetings. The last occasion had been in a paper by A. A. Jones submitted in February, 1932, and dealing with sandfilling methods at the Hodbarrow hæmatite mines in South Cumberland.

Although sandfilling was not very widely used in metalliferous mining, it was, no doubt, a very effective method of supporting heavy ground in certain special cases, but it was rather limited in its application. The method had its critics and was not by any means universally approved. The paper was most practical, full of valuable operating details, and would be much appreciated by all those who were engaged in work of this nature. The author was not able to be present to introduce his paper in person but Mr. Rutter had kindly undertaken to do that.

Mr. D. W. Rutter expressed his regret that the author was unable to be present to introduce his own paper. Mufulira was a comparatively young mine in that stoping commenced only in 1934; the mine had been shut down from December, 1931, to 1933, when unwatering was started. At first mining operations were confined to the orebodies lying above the 460-ft. haulage level and the sub-level open stoping method of mining was introduced. As lower levels were opened up it was decided to mine the thick

B/C orebodies by a caving method—in order, among other things, to improve extraction. The change-over in mining methods could not be made, however, until the water table in the hanging-wall beds had been lowered to a point below the cave line. Therefore an intensive drainage programme was put in hand. The importance of this work would be appreciated when it was realized that about 16,000,000 gallons of water were pumped from the mine every 24 hours. At the same time deeper mining gave rise to increased ground pressures and the need for hanging-wall support. It was fortunate that classified mill tailings proved eminently suitable as a filling medium, because that happened to be the only cheap material available. The drainage problem had been overcome to such an extent that it was only necessary in many cases to fill stopes sufficiently to prevent air blasts. The author had pointed out that the hazard to life was considerable when an air blast was driven through the mine workings by a sudden collapse of hanging-wall over a series of stopes.

Under the heading ' Mining System ' the author had referred to the reversal of the sequence of stoping. Experience with sand-filling had shown the proper sequence to be C-B-A. In addition the C orebody had a greater thickness than either A or B, and it was a simpler matter to keep the stope faces of all three orebodies in step when the C orebody was mined first. The fact that the three orebodies could be mined in the sequence C-B-A proved the success of the sandfilling operation in practice. Naturally, when the sub-level caving method of mining came into operation the sequence would revert to A-B/C.

There was insufficient material available to fill stopes completely with sand, although in some cases possibly the level of the fill could be raised if necessary with the aid of stand pipes. The amount of fill indicated in Fig. 8 had so far proved to be all that was necessary to obtain the desired results.

The author discussed the development of timber desliming tanks, which had given very satisfactory service. Batteries of steel cones had been discarded owing to the time and expense involved in dismantling and re-erecting them. The automatic sand discharge valve was also discarded because of occasional stoppages. The simple hand-operated discharge door illustrated in Fig. 6 worked very smoothly and, as a native attendant was, in any case, required at each battery, operating costs were not increased thereby. Sand was fed to underground distributing points situated in the foot-wall of the C orebody through churn drill holes lined with concrete pipe. These holes were drilled at 500-ft. intervals.

It was interesting to note that the tailings thickener discharge had first to be diluted with water to reduce solids from 56 per cent to 34 per cent in order to obtain a satisfactory separation of sand and slime. The author stated that the most important requirement of a sandfilling material was that it must leach rapidly. The presence of slime reduced the leaching rate and too much slime resulted in failure of the bulkheads through hydrostatic pressure. All bulkheads were now constructed of concrete reinforced with old wire rope. Sandfilling followed so closely on stoping that timber bulkheads did not stand up to blasting in adjoining stopes, even with concussion barricades. The author preferred timber bulkheads, however, for use in moving ground that was sufficiently remote to be unaffected by blasting operations.

The great superiority of rubber-lined pipe for transporting abrasive material was once again demonstrated. Unfortunately, as this pipe had become unobtainable during the war a substitute had had to be used. It was found that a pipe supplied in standard lengths and fitted with welded flanges had a life of four to five months, against one month with ordinary pipe and threaded flanges.

The novel compressed-air non-return valve shown in Fig. 11 had proved most satisfactory. It was easily made and very efficient in operation. By means of compressed air sand could be transported long distances horizontally and, at the same time, wear was distributed evenly round the bore of the pipe. This even distribution of wear was important, as it avoided the necessity of turning the pipe at frequent intervals. In order to provide smooth and continuous operation an independent air compressor had been installed to ensure a constant supply of high-pressure air.

The actual stope filling was an intermittent process. A period of three days was allowed between successive layers of fill to enable the sand to leach and consolidate. The author observed that six hours after filling was stopped the surface of the sand was compact and hard and the leaching ceased about two months after a stope was filled. Members had probably noticed the error in the formula given on p. 23. The value of ' s ' was 28, so that the denominator of the second fraction should be 80 *minus* 28. This gave a corrected value for ' S ' of 158 tons per hour, equivalent to 94,800 tons per month.

The author had concluded with some practical hints that would no doubt be of interest to those faced with a similar problem.

Mr. James Whitehouse then opened the discussion on the paper by congratulating the author on his paper, which he thought was a record of a job well done. He had no criticisms to make and no questions to ask, but that was not surprising, because the technique of sandfilling had advanced through the years and the author had carried it a stage further.

As he read the paper he had recalled that in 1910 it was their late President who was in charge of sandfilling operations in South Africa, he having been to Germany to study practice there, and it had seemed a long step backwards to those times. Practice at Mufulira in some respects followed that which they had used at the New Modderfontein mine, particularly as regards the desliming and dewatering of the sand at the top of the borehole with cones. Mufulira conditions were eminently suitable for sandfilling, but, of course, they entailed, as no doubt the manager had realised, considerable risk, and he noted that they had not been without the unavoidable breakaways that sometimes accompanied sandfilling.

He recalled that at the City Deep mine an area in a worked-out part of the mine that had been sandfilled some years before unfortunately became waterlogged. Water had got into the sand from the hanging-wall, which resulted in the bursting of the barricade, and the sand broke away with a sudden rush, gathered up the loose rock that was in its path, and finally finished up by travelling down an oreshoot which, unfortunately, opened at the bottom on to a station at the vertical shaft. By this time it had gathered so much material that it went right through the station, and 60-lb. rails forming the grizzly were precipitated into the bottom of the shaft as though they were pieces of matchwood. Had the accident happened an hour earlier there would have been some hundreds of people on that station, but fortunately it only caught one native who was operating the door at the bottom of the oreshoot ; it cut him into small pieces. Consequently the speaker thought that the use of concrete bulkheads at Mufulira was amply justified ; nothing else would have been a safe practice.

The author had given them some very valuable information on the percentage of solids permissible for the distribution of sand and also the amount of compressed air required for boosting in the mine. The use of compressed air for this purpose was rather a novel idea and it was interesting that with its use the wear on one side of the pipe was very much diminished.

With regard to the cost of sandfilling at Mufulira, the figure of about 6d. a ton was highly satisfactory. It was about half the cost, as he remembered it, of sandfilling on the Witwatersrand, but, of course, there were two explanations for that: First, the amount of preparatory work necessary was not very great, and,

[redacted]

secondly, the amount of sand filled in one locality was very considerable. In conclusion, he wished to congratulate all those concerned on the excellent work that had been done.

Mr. L. Tucker* said he greatly appreciated the invitation to attend the meeting. As regards the detailed work in connection with sandfilling there was not very much to add. The author had covered all the operating details and the paper had been ably introduced by Mr. Rutter and commented upon by Mr. Whitehouse. He would like to say how important sandfilling was at Mufulira. In opening the mine it was necessary to choose a stoping method. Mufulira had flat dipping massive orebodies and certain beds in the overlying rocks were heavily waterlogged. Those beds were so close to the orebody that any extensive caving of the hanging-wall before they were drained would have endangered the mine. Therefore, the question of hanging-wall support became of immediate importance. Sub-level stoping was adopted as being most suitable for the upper levels.

Mill tailing was found to be the best and most economical support available. It was recognized that, due to its compressibility, sandfill was not an absolute support and that with increasing depth larger and larger pillars would become necessary. Sandfilling provided sufficient support, however, to allow reasonable extraction of ore from the upper levels and for drainage of the hanging-wall beds, without which the caving method now being introduced would have been impossible.

Sir Lewis Fermor hoped they would not be surprised that a geologist should wish to take part in the discussion on this paper. But sandfilling was a problem of very great practical interest to the geologist because of its effect upon the conservation of mineral resources. In India, hydraulic-stowing, or sand-stowing as they called it, had come very much to the fore in recent years in connection with the working of the coalfields of Bengal and Bihar.

The Indian coalfields contained many successive seams of variable value, and the companies who worked them naturally wished to work the best seams first. It was, therefore, important that in working any one seam the overlying ground should not be allowed to collapse. By the ordinary method used in most of the mines—pillar and stall—about 40 per cent of the coal was recovered in driving the galleries, leaving 60 per cent in pillars. Then, if the market price of coal fell, the problem was how to win those pillars at a profit if the financial policy of the company had not been

* Mine Manager.

to debit a part of the cost of future extraction of pillars against the cost of driving galleries. In such cases a colliery company often preferred to drive fresh galleries and leave the pillars alone.

After some years one coalfield in particular (Jharia) had become rather like a honeycomb and there was a great danger of fires : in times of high prices there was a tendency to rob the pillars, causing collapses, and then fires broke out. In consequence, the Government of India, now many years ago, took the advice of a mining specialist from England, and then, on receipt of his report, appointed a Commission, which made suitable recommendations. The resulting proposals of the Government of India were circulated amongst those who were interested ; everyone had kicked at the particular recommendation that hit him, making difficult action by the Government of India. The Geological Survey had, however, succeeded in saving the situation and prolonging the action.

Large-scale new topographical maps were made of the two coalfields in question (Jharia and Raniganj) and new geological surveys were made to estimate the quantity of coal available. Experiments in coal washing were made to find out whether it was possible to improve the quality of low-grade coal at an economic price. The answer was in the negative and therefore it was important to conserve the high-grade coal. Eventually the Government had appointed another Commission, and Government had introduced measures to encourage sand-stowing, if not to make it compulsory. Co-operative schemes had been introduced and a levy imposed on the output of coal to help pay for such schemes.

Before Government could be cajoled into all this it had been necessary to make sure that there was a sufficient supply of sand. So he had deputed one of his officers, Mr. G. V. Hobson, a member of the Institution, to make a rough general survey of the Damodar valley area and report whether it was likely that the annual replenishments of sand during the monsoon would be sufficient to make up any losses caused by withdrawals for sand-stowing, supposing this were generally adopted. Mr. Hobson's report had been to the effect that they need have no fear ; the annual replenishments would be at least adequate to make up for any possible extractions of sand. With that assurance it had been possible to press for adequate measures for the conservation of coal.

Prior to this some of the leading companies had already themselves introduced measures of sand-stowing, so that the process was already known as applied to coal, and it was only really an economic question of whether the companies could pay for the work ; the smaller companies had been helped by co-operative schemes.

He had noticed in the paper that Mufulira mine supplied its own stowing material, at least in part, and that was a good thing. But, of course, a coal mine was not like that, because the whole of the coal extracted was going to be used: one did not put it through a dressing plant, recover some 4 per cent and throw 96 per cent away, and have that available, after it had been cleaned of slimes, to fill the voids. So the two problems did differ; but, on the other hand, when the coal industry took sand out of the river they did not have to deslime it; it was already good clean sand available for use.

The paper was one of very great importance, and he felt surprised that the mining members of the Institution had not taken more interest and filled the room in order to join in the discussion.

Mr. R. H. Craven thought the way in which the author had dealt with his subject was most informative and the description of the research work connected with the system of sandfilling of the stopes at the Mufulira mine was extremely interesting, the detail being so complete. An important point which he had failed to gather in reading the paper, however, although it might have escaped his observation, was any reference to the nature of the ore. If this was a sulphide, and he was inclined to think that it must be a chalcopyrite, as the author had mentioned that the upper zone had been leached over a height of some 460 ft., the water carrying the sand in the form of a slurry must have presented particular difficulties in pumping it to the surface again.

If the ore was a sulphide it would be interesting to know what proportion of the consumption of the metal pipes, carrying the slurry into the stopes, was due to abrasion and how much to electrolytic action on the iron. Many years before, about 1910, this system of filling with sand was adopted at a mine in Tuscany. The firm of Carlo Marchi acquired a concession to mine sulphide ores adjoining the Gavorrano mine belonging to the Montecatine Company, which had suffered from various serious falls of ground in the course of mining operations. To avoid similar difficulties in the new mine and to support the walls of the deposit, filling with sand after the ore was broken appeared to give the greatest advantages.

The system of mining adopted by Marchi called for stoping from the bottom upwards, as was done in the Mufulira mine. Shafts were designed to go down to the deepest point at which the pyritic ores existed in the new concession and when these reached below the lowest workings of the Gavorrano mine it was found that they had to pump not only the water from their own mine but also that which filtered through from the adjoining property. Eventually the new mine owner decided to bear the double burden of pumping for the time being, knowing that, as the working progressed upwards, their difficulties would cease and the Montecatine Company would have the doubtful pleasure of pumping for both mines. The Gavorrano mine was working downwards and the Marchi mine upwards, and the time was not long before a common level was reached. The Montecatine Company eventually decided to put in an adit level of some eleven kilometres in length, rather than pump the water to the surface. He wondered whether similar problems might not occur in the district around the Mufulira mine.

Another interesting thought emerged in connection with the tests made in ascertaining the doming properties of the leached sands. In filling stopes in mines where copper sulphides were the mineral produced, a certain amount of copper and iron was found in solution in the acidulated water percolating through the mineral and rock. This liquid found its way into the open spaces in the rock filling used to support the walls of the stopes after the ore had been broken. In mines with which he had been connected, the filling used became cemented together quite firmly after a few months and after long periods of, say, ten years or more, when it became necessary to open passage-ways through fillings, blasting had to be resorted to as the mass had become as hard as concrete. Was it possible that the good doming results shown by the tests made were due somewhat to the cementing properties of the iron and copper sulphate in solution in the water having assisted in the binding together of the sand?

The cost of pumping the water from the stopes leached from a ton of filling was given as 0·015s. Was that inclusive of discharging the water at the surface from the 900-ft. level? The percentage of solids was given as 68, thus 32 per cent or one-third of a ton of water had to be raised 900 ft. at a cost of something under a farthing. If his figures were correct, this appeared to call for very efficient pumping equipment, details of which would be very interesting.

The application of the properties of sand in connection with mining was of considerable importance and further developments would doubtless increase the sphere of its utility in other operations connected with ore production.

Mr. Tucker replied that it was a sulphide orebody. There was no evidence of cementing in the filled stopes which had been entered subsequently. The sand was still damp and could be cut with a shovel. When so cut it would stand in a vertical face several feet high. That was one point which had emerged from the discussion—the possible danger to workings lower down. Naturally this aspect had been watched very closely. After the sand had been in the stope for some time and the excess moisture had been drained off, the consistency was such that if one of the side barricades was removed the sand would not run. It was quite safe to walk on the sand as soon as the surface moisture had drained off. That moisture drained off in a few hours, while the excess moisture drained from the filled stope in, say, three months. There had been sand-filled stopes adjacent to open stopes when the intervening pillar had crushed. In these cases there had been no tendency for the sand to run out.

Another small point was the mention of the possibility of water seeping in through the hanging-wall and wetting the sand, thereby allowing it to become soft enough to flow out again. The water in the actual orebody was drained out ahead of the time before stoping took place. There was a great deal of water well back of the hanging-wall, behind impervious strata. This water constituted the main drainage problem, but there was no percolation through the impervious strata into the sand-filled stopes.

Mr. W. H. Wilson said that he had been struck by the fact that the sandfilling in Mr. Craven's mine had become cemented and needed explosives to move it, while at Mufulira consolidation only had taken place. The diffidence of certain mines to the use of sandfilling might be overcome if the sand subsequently became cemented and rocklike. Soil mechanics had, during recent years, become a most important branch of civil engineering, and Mr. Wilson thought that research into the causes of the cementation of sands might have been undertaken by such a firm as Soil Mechanics, Ltd. One wondered what constituent—chemical or slurry—might be needed to cause cementation.

Lt.-Col. J. B. Simpson said that he had seen sandfilling used in deep-level rill stopes, where the sand was sent down in tubs from surface and tipped into the stopes together with waste-rock from development. This was to give a more compact support than would have been the case with waste-rock alone. When the fill was within 6 ft. of the stope face, a further 50 tubs of sand were used to form a mat. Being damp, this tended to hang up at the top of the rill and had to be scraped down. Until the sand dried off the stope was uncomfortably humid, being in a hot dry mine. This humidity would not affect working conditions at Mufulira, but might in other mines differently placed.

Other members taking part in the discussion included Mr. R. S. Botsford, Professor W. R. Jones, Mr. E. J. Pryor, Professor S. J. Truscott, and Mr. T. Muir Warden.

The President said that although he had seen mining operations in a good many parts of the world, he had only once come across sandfilling methods being employed, so it was evidently not in very common use. He had been most interested to hear from Mr. Tucker that sandfilling was only a temporary measure at Mufulira and that they proposed to go back to caving methods in due course.

He said he was sure members would wish to pass a hearty vote of thanks to the author for his most interesting paper, to Mr. Rutter for introducing it, and also to Mr. Tucker who had given them such interesting additional information. The vote was carried by acclamation.

AUTHOR'S REPLY TO DISCUSSION*

ON

The San Telmo Orebody, Spain.

By J. C. Allan, *Member.*

Mr. J. C. Allan: I am gratified that the paper has produced such an interesting discussion and have to express my best thanks to Dr. David Williams for introducing it in my absence. Dr. Williams, in his comments, considered that the theory was based on an inadequate amount of fact, but I feel that the justification for this has been shown by the fact that the majority of the discussion was based on the theoretical part of the paper rather than on the facts as presented. Had the data available been more complete I might not have had to venture into the dangerous field of theoretical geology.

The lack of recorded data is such that I cannot give Dr. Williams concrete answers to a number of the points he raises. The apex of the mass, however, showed abundant characteristic black chalcocite and secondary copper minerals, as distinct from the primary chalcopyrite occurring in certain zones of the mass down to below the 11th Floor. Dr. Williams appears to make two striking admissions:

(1) That under the influence of pressure the sulphides would tend to suffer plastic flow.

(2) That such plastic flow might exhibit cross-cutting relationships towards the other sulphides in the form of intrusive veins.

These admissions, from so able a protagonist of the replacement hypothesis, seem to me to surrender the fort. If, under certain conditions, plastic flow of sulphides can take place, it would appear to be only a question of degree for such conditions to produce enough mobility for the sulphides to travel some considerable distance before reaching their present situation. My reading of the evidence is that an important proportion of the San Telmo mass arrived at its present situation with substantially the same concentration of sulphides in which it is found to-day. That it did so either as a hydropyritic matte, as sulphides under conditions of plastic flow, or as sulphides in a dispersed solid phase, to which Dr. Williams referred later in his remarks, is not a point that I feel competent to discuss and does not affect the main thesis.

Bull. 478, Jan., 1946.

This view seems to me to be confirmed by the extremely interesting contribution by Mr. P. E. Whelan. If I read his remarks correctly, if the rhythmic precipitation occurs under quiescent conditions it infers that the substances precipitated were there in substantially their present concentration when precipitation took place. While in the body of my paper I have suggested the possibility of a colloidal state, the main point I was endeavouring to make was that the material arrived at its present situation in a concentrated form, as distinct from a replacement by dilute hydro-thermal solutions. I would like to thank Mr. Whelan for his interesting remarks, which are a valuable contribution to the study of this problem.

Mr. G. Thomas, in stating that no other type of replacement except hydro-thermal has been recognized, does not, I think, explain the occurrence of the concentric structure of the specimens submitted by the President.

I was glad to have Mr. Richardson's confirmation of the way in which complex ore could be followed through clean-cut contacts —such as are shown in Plate I, Figs. 3 and 4, of the paper—to ore showing marked pyrite complex banding.

Both Mr. Varvill and Dr. Smith refer to the manner in which some of these pyritic masses cut out in depth and I have always felt, with Dr. Smith, that this feature fits in far better with an injection theory than with any possible form of replacement.

In conclusion I would like to record my thanks to Dr. W. R. Jones, without whose support and advice the paper would not have been written ; to Colonel J. Cross Brown for the details of the map in Fig. 1, and to Mr. Andrew Pearson who helped in the preparation of an early draft.

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CONTENTS.

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NOVEMBER 14TH, 1946.

LIST OF CONTENTS.

[NOTE.—*The Report of Discussion at the Second Ordinary General Meeting of the Session, Further Contributed Remarks and Authors' Replies to Discussion, and Report of Discussion in Johannesburg on Papers previously submitted, are also attached hereto, together with the following new Paper :*

The Panasqueira Mines, Portugal: Wolfram Mining and Milling ; Labour Organization.

By J. C. ALLAN, *Member*, G. A. SMITH, *Member*, and R. I. LEWIS, *Associate*.]

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Committee on Mining in Great Britain.

NOTICE OF GENERAL MEETINGS.

The THIRD ORDINARY GENERAL MEETING of the Fifty-sixth 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, NOVEMBER 21st, 1946, at 5 o'clock p.m.

The following Paper, of which a copy is attached, will be submitted for discussion :

The Panasqueira Mines, Portugal: Wolfram Mining and Milling ; Labour Organization.

By J. C. ALLAN, *Member*, G. A. SMITH, *Member*, and R. I. LEWIS, *Associate.*

The FOURTH ORDINARY GENERAL MEETING of the Session will be held, also in the Apartments of the Geological Society, on THURSDAY, DECEMBER 19th, 1946, at 5 o'clock p.m., when the following Papers, both of which were published with the September, 1946, *Bulletin*, will be submitted for discussion :

The Geology and Opencast Mining of the Jurassic Ironstones of Great Britain.

By W. DAVID EVANS, *Associate.*

Notes on the Estimation of Tonnage and Grade of Some Chromite Dumps.

By N. W. WILSON, *Associate.*

(*Note.*—Since the publication of these two Papers the arrangements for their discussion have been altered and they will, as announced above, be presented at the December General Meeting and not in November as printed at the head of the Papers.)

Light refreshments will be provided at 4.15 p.m. for members and friends attending the Meetings.

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 Council desire to remind members that in addition to the more comprehensive types of papers for discussion at General Meetings, they welcome for publication in the *Bulletin* short notes recording data or describing technical experience, which may be of general interest and value. Such notes are governed by the same rules in regard to acceptance as ordinary papers, but would be open for discussion by correspondence.

FIFTY-SIXTH SESSION : 1946–1947.
DATES OF SUBSEQUENT MEETINGS.

The following dates have been provisionally fixed for subsequent General Meetings of the Institution during the Session 1946–47 :

Thursday, January 16th, 1947.	Thursday, April 17th, 1947.
,, February 20th, 1947.	,, May 15th, 1947.
,, March 20th, 1947.	

Preliminary Announcement.

CONFERENCE ON SILICOSIS, PNEUMOKONIOSIS AND DUST SUPPRESSION.

Arrangements are now being made jointly by the Institution of Mining and Metallurgy and the Institution of Mining Engineers for a Conference on Silicosis, Pneumokoniosis and Dust Suppression in Mines, to be held in London on **Wednesday and Thursday, 16th and 17th April, 1947.**

The Conference will be divided into sessions consisting broadly of papers on (*a*) Dust Sampling and Evaluation, (*b*) Pathology of Pneumokoniosis and Silicosis, and (*c*) Dust Suppression. Papers are being prepared by authorities in these subjects, and a full list will be published early in 1947.

R.S.M. (OLD STUDENTS) ASSOCIATION.

The following notice is published at the request of the Association :

Will any past student of the Royal School of Mines who has not received a form asking for his particulars for insertion in the forthcoming Register of Old Students please communicate at once with the Secretary, Dr. J. H. Watson, Royal Mint, E.C. 3. Will any Old Student who has received such a form and has not yet filled in and returned it please do so as soon as possible.

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 September 12th, 1946 :

To Membership—

Elkan, Edward Felix (*Kuala Lumpur, Malaya*).
Spalding, Jack (*London*).

To Associateship—

Banfield, George Edmund Arthur (*London*).
Dover, Hubert Nicholas Basil (*Digboi, Upper Assam*).
Foley, Timothy Anthony (*Konongo, Gold Coast*).
Hamilton, Peter Hulbert Glenn (*Batu Gajah, Malaya*).
Lamb, Arthur James (*Maraisburg, Transvaal*).
Landcastle, Ralph (*Birmingham, Warwickshire*).
McCulloch, Gerald William (*Newcastle-on-Tyne, Northumberland*).
Massmann, Francis Grenville (*Barnsley, Yorkshire*).
Nater, Rudolph Charles (*Santa Rosa de Copan, Honduras*).
Passmore, Linton James Bartle (*Kisumu, Kenya*).
Rees, John David (*Holywell, Flintshire*).
Rowe, Edward James (*Prestea, Gold Coast*).
Smith, Maurice (*Bulawayo, Southern Rhodesia*).
Wilson, Tom Revell (*Lobitos, Peru*).

The following have applied for admission into the Institution since September 12th, 1946 :

To Membership—

Cocks, William Trevor Blamey (*Sheffield, Yorkshire*).

CANDIDATES FOR ADMISSION—*continued.*

To Associateship—

Borradaile, Arthur Basil Collingwood (*Gatooma, Southern Rhodesia*).
Paton, James Heweit (*Cumnock, Ayrshire*).
Rice, Herbert Ralph (*Raichur, India*).
Severs, Robert Braithwaite (*Marikuppam, South India*).
Strong, Brian Eustace (*Pennard, Glamorganshire*).
Treloar, William Henry Nash (*Penhalonga, Southern Rhodesia*).
Trythall, William John (*Accra, Gold Coast*).

To Studentship—

Bartlett, Brian (*Reading, Berkshire*).
Bath, Kenneth Arthur Edmund (*Gillingham, Kent*).
Cave, Colin Campbell (*London*).
Champness, Peter Anthony (*Ewell, Surrey*).
Davidson, David Hay (*Sevenoaks, Kent*).
Dayson, William Anthony Prosser (*Birmingham, Warwickshire*).
Dennis, John Gordon (*London*).
Dizioglu, Mehmet Yusuf (*London*).
Domzalski, Wojciech (*London*).
Gibson, David (*Bexhill-on-Sea, Sussex*).
Grondijs, Alfred Allen (*London*).
Grun, Bronislaw (*London*).
Khan, Abdul Mannan (*Croydon, Surrey*).
Kilic, A. Enver (*London*).
Knapp, John Henry (*Camborne, Cornwall*).
Linzell, Peter George (*Coulsdon, Surrey*).
Parkes, David Martyn (*Braintree, Essex*).
Pearse, Geoffrey Edwin (*Croydon, Surrey*).
Roberts, Adrian John (*Matlock, Derbyshire*).
Semmens, John William (*Southall, Middlesex*).
Simmons, Donald John (*London*).
Standers, John Rudolph (*Selukwe, Southern Rhodesia*).
Taylor, Arnold (*London*).
Watts, Martin (*Dover, Kent*).
Weekes, Peter Thomas (*Camborne, Cornwall*).
Williams, Jack (*Hayle, Cornwall*).
Wilson, John Corner (*Leeds, Yorkshire*).
Worthy, Stanley Raymond (*Camborne, Cornwall*).

TRANSFERS AND ELECTIONS.

The following have been transferred (subject to confirmation in accordance with the conditions of the By-Laws) since September 12th, 1946 :

To Membership—

Chappel, John Traer (*Ipoh, Malaya*).
Dannatt, Cecil William (*London*).
Hawes, James Thomas (*La Carolina, Spain*).
Manglis, Costa Pericles (*Nicosia, Cyprus*).
Wardrop, John George Lessels (*Keswick, Cumberland*).

TRANSFERS AND ELECTIONS—*continued.*

To Associateship—

Andrew, Robert Bruce (*Dunedin, New Zealand*).
Birkbeck, James Martin (*Royal Engineers*).
Blight, Charles Garfield (*Rooiberg, Transvaal*).
Clark, William Lionel Gladwell (*Johannesburg, Transvaal*).
Dunstan, Arthur Stanley (*Jos, Northern Nigeria*).
Fitch, Frederick Harry (*Batu Gajah, Malaya*).
Walker, Stanley Edmond (*Selukwe, Southern Rhodesia*).
Wallace, Reginald Catherwood (*Dunnottar, Transvaal*).

The following have been elected (subject to confirmation in accordance with the conditions of the By-Laws) since September 12th, 1946 :

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Giegerich, Joseph R. (*Kimberley, British Columbia*).
Gooday, Wilfred Emile (*Johannesburg, Transvaal*) (reinstatement).
Kirkpatrick, William Stafford (*Trail, British Columbia*).

To Associateship—

Arthur, John Albury (*West Rand, Transvaal*).
Barker, John (*Gwelo, Southern Rhodesia*).
Bhatt, Gunvantrai Vrajalal (*Lohardaga, India*).
Brodie, Robert Philip (*Hove, Sussex*) (reinstatement).
Francis, Edward Carey (*Barakin Ladi, Northern Nigeria*) (reinstatement).
Gobert, Mancil Joseph (*Selkirk, Manitoba, Canada*).
Krishnaswamy, Erapally (*Champion Reefs, South India*).
Napier, Alexander Nicol (*Aboso, Gold Coast*).
Osborn, George Howard (*Aylesbury, Buckinghamshire*).

To Studentship—

Stuckey, Ronald Clive (*London*).
Tremlett, Christopher Percy (*Newquay, Cornwall*).

NEWS OF MEMBERS.

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

Mr. E. C. Airth, *Associate*, has left East Rand Proprietary Mines, Ltd., to join the staff of Blyvooruitzicht Gold Mining Co., Ltd.

Mr. G. Keith Allen, *Member*, has left England on a visit to West Africa.

Mr. D. G. Armstrong, *Associate*, has left England for India to take up the appointment of chief metallurgist at Hutti gold mines.

Mr. R. B. Allwright, *Associate*, has arrived in England from Nigeria.

Mr. Charles A. Banks, *Member*, has been sworn in as Lieutenant-Governor of British Columbia.

Mr. L. H. Bartlett, *Member*, has left England on a visit to India and expects to return in six months' time.

Mr. J. S. Benson, *Associate*, is leaving England to join the staff of London and African Mining Trust, Ltd., in Nigeria.

Mr. A. J. Bensusan, *Member*, expects to leave Western Australia early this month to visit his son, Mr. A. M. Bensusan, *Associate*, who has taken over a gold mine in Southern Rhodesia under a Government scheme for ex-servicemen. He may come to England in the summer of 1947.

NEWS OF MEMBERS—*continued.*

Mr. P. Best, *Associate*, has returned to the Kolar Gold Field after furlough in England.

Mr. R. A. L. Black, *Student*, has arrived at Johannesburg, where he is an official learner at City Deep mine.

Col. H. H. W. Boyes, *M.C.*, *Member*, has returned to Northern Nigeria.

Mr. J. K. Broadhurst, *Associate*, has left England to rejoin the staff of Anglo Oriental (Malaya), Ltd., Kuala Lumpur.

Mr. W. A. Burke, *Associate*, is now in England on leave from Nigeria.

Mr. J. T. Chappel, *Member*, has left England on his return to Malaya.

The Hon. E. W. M. Cokayne, *Student*, is now working for Central Patricia Gold Mines, Ltd., Ontario.

Mr. C. Courtis, *Associate*, has rejoined the staff of Consolidated African Selection Trust, Ltd., Gold Coast, after demobilization.

Mr. C. A. Creelman, *Associate*, has resigned his post of underground superintendent of New Occidental Gold Mines, N.L., Cobar, N.S.W., to join the staff of Emperor Gold Mining Co., Ltd., Fiji.

Mr. A. Ctvrtecka, *Associate*, has arrived in Colombia to take up a position with Frontino Gold Mines, Ltd.

Mr. E. B. Davies, *Associate*, has returned to Malaya on the staff of Anglo Oriental (Malaya), Ltd., Perak.

Mr. K. H. Davison, *Associate*, has left England to begin work with Cia de Petroleo Shell de Colombia.

Mr. E. W. Dawson, *Associate*, has left Non-Ferrous Minerals Development, Ltd., to join Mianrai Teoranta, Eire.

Mr. E. L. Day, *Associate*, has left England on a visit to the Gold Coast for Imperial Chemical Industries, Ltd.

Mr. J. E. Denyer, *Associate*, has been released from the Royal Engineers and has returned to Mawchi Mines, Ltd., Burma.

Mr. F. T. C. Doughty, *Associate*, has rejoined Cyanamid Products, Ltd., after demobilization.

Mr. T. M. Dover, *Associate*, has been appointed opencast mines manager at the Scunthorpe Mines Section of United Steel Companies, Ltd.

Mr. S. B. C. Edwards, *Associate*, has joined the staff of the Directorate of Opencast Coal Production, Ministry of Fuel and Power.

Mr. Douglas F. Foster, *Associate*, has left England to take up a post with Messrs. James Barwell (South Africa), Ltd., Johannesburg.

Mr. W. F. Garwood, *Associate*, has resigned from the Southern Rhodesian Government Mines Department.

Mr. Donald Gill, *M.C.*, *Member*, is returning to England from Egypt.

Mr. W. C. Grummitt, *Associate*, has returned home from abroad.

Mr. R. M. Hannah, *Student*, has joined the Indian Copper Corporation, Ltd., as works metallurgist.

Mr. R. J. Harvey, *Associate*, has retired from his post with the Sulphide Corporation, Ltd., after 36 years' service, and has settled at Lake Macquarie, N.S.W.

Mr. G. H. Hoby, *Associate*, has been appointed chief surveyor of the Witwatersrand Nigel Gold Mining Co., Ltd.

Mr. V. T. Hockin, *Associate*, has been appointed chief inspector of mines, Tanganyika Territory, with retrospective effect from January, 1945.

NEWS OF MEMBERS—*continued.*

Mr. H. Hocking, *Associate*, has resumed his appointment as mill manager, Pahang Consolidated Co., Ltd., Malaya.

Mr. R. R. H. Horsley, *Associate*, formerly of the Royal Engineers, attached Indian Engineers, has been awarded the M B.E.

Mr. J. R. Hosking, *Student*, has left England to join the staff of Consolidated African Selection Trust, Ltd., Sierra Leone.

Mr. J. O. Howells, *Member*, is in England from British Columbia and will soon be leaving for Denmark and Sweden.

Mr. W. Hutchin, *Associate*, is now in England on leave from the Gold Coast.

Mr. J. H. Jackson, *Associate*, has left England for Rhodesia, to join Roan Antelope Copper Mines, Ltd.

Mr. D. E. S. King, *Associate*, is returning to England from Malaya.

Mr. E. C. Knuckey, *Member*, has been appointed manager and chief mining engineer of Chrestien Mica Industries, Ltd., Bihar, India.

Mr. H. A. Kursell, *Member*, has been appointed consulting engineer to the Mining Department of the American Smelting and Refining Co.

Mr. N. Landau, *Member*, has left Rezende Mines, Ltd., on his appointment as superintendent, Guinea Fowl Miners' Training School, Southern Rhodesia.

Mr. R. Landcastle, *Student*, has left England to take up a post as inspector of mines, Tanganyika Territory.

Mr. R. J. Lemmon, *Member*, has left England for the Gold Coast.

Mr. A. M. McMillan, *Associate*, has arrived in England from the Gold Coast.

Mr. C. P. McMillin, *Associate*, has returned to the Gold Coast.

Mr. J. C. Mance, *Member*, has been released from his employment with the Ministry of Supply.

Mr. W. Mankovsky, *Associate*, has left England to take up the appointment in Northern Nigeria of geologist to the London and African Mining Trust, Ltd.

Mr. L. C. Millett, *Associate*, has returned to the Gold Coast after furlough in England.

Mr. J. I. Milne, *Associate*, has left England on a six months' visit to Burma.

Mr. R. H. Mitchell, *Associate*, expects to arrive in England from Rhodesia about the end of November.

Mr. P. E. Molyneux, *Associate*, is now in Rhodesia.

Mr. G. C. Morgan, *Associate*, having been demobilized, is now on the Gold Coast.

Dr. N. E. Odell, *Associate*, has been lecturing in India and expects to return to England this month.

Mr. L. E. T. Parker, *Associate*, has left England to join the staff of Anglo-Iranian Oil Co., Ltd., at Abadan, South Iran.

Mr. I. L. Patterson, *Associate*, has arrived in Malaya from England.

Mr. H. J. D. Penhale, *Member*, has returned to Fernando Po from leave in England.

Mr. J. Penhale, *Associate*, has returned to Sierra Leone from England.

NEWS OF MEMBERS—*continued.*

Mr. W. Pulfrey, *Associate*, has arrived in England on leave from Kenya.

Mr. C. H. Richards, *Member*, has left England on his return to Tanganyika Territory.

Mr. J. A. Royce-Evans, *Student*, has been released by Gold Coast Main Reef, Ltd., to rejoin Gold Coast Banket Areas, Ltd.

Mr. F. G. Sharp, *Associate*, is on his way to India, after leave in England, to return to Mysore Gold Mining Co., Ltd.

Dr. A. C. Skerl, *Associate*, at present chief geologist to Quebec Gold Mining Corporation, B.C., has also been appointed consulting geologist to Cariboo Gold Quartz Mining Co., Ltd.

Mr. R. Smythe, *Associate*, has left England to rejoin the staff of Nundydroog Mines, Ltd., Mysore.

Mr. W. V. C. Stephens, *Associate*, has left Leeds and is now working in London.

Mr. G. Stiles, *Associate*, has been appointed manager of the Sheba Section, New Consort mine, Barberton, E. Transvaal.

Mr. H. L. Stocken, *Associate*, has arrived in England on leave from India.

Mr. A. I. Sussmann, *Associate*, who was demobilized in 1945, is now at Spaarwater, Transvaal.

Mr. P. A. W. Thuell, *Student*, has joined the staff of Messrs. Osborne & Chappel, Ipoh, Malaya.

Mr. J. W. C. Treeby, *Associate*, has returned to Malaya to resume practice at Penang.

Mr. E. H. Tregoning, *Associate*, is returning to Burma from England after furlough to resume his position with the Burma Corporation, Ltd.

Mr. A. J. Walton, *Member*, is sailing for South Africa early this month after his visit to England.

Mr. E. J. Wayland, *Member*, has arrived in England on leave from South Africa.

Mr. J. S. Williams, *Associate*, is now in England on leave from India.

The following Members of the Institution have been elected Honorary Life Members of the Chemical, Metallurgical and Mining Society of South Africa: Dr. J. V. N. Dorr, Mr. T. K. Prentice, Brigadier R. S. G. Stokes, *C.B.E.*, *D.S.O.*, *M.C.*, Professor S. J. Truscott, and Mr. J. Allan Woodburn.

ADDRESSES WANTED.

E. A. Banning.	A. W. Eyre.	P. H. G. Owen.
E. B. Currie.	A. McCall.	E. R. Robinson.
L. W. Elsum.	J. H. Marshall.	W. E. Storey.

BOOK REVIEWS.

Examination, Boring, and Valuation of Alluvial and Kindred Ore Deposits. By H. L. H. Harrison. London: Mining Publications, Ltd., 1946. x, 264 p., illus. 30s.

This useful textbook fully bears out the author's claims in his preface, namely ' to assist the untrained man, to arouse interest in what some may

BOOK REVIEWS—*continued.*

consider a monotonous task, and to contribute towards the standardization of methods'. Its publication appears at an opportune time when there is an accumulation of many years' arrears in prospecting for gold and tin alluvials and at the same time a grave shortage of young engineers trained to this class of work. Whilst including a useful summary of the characteristics of economic minerals found in surface deposits, the author only touches lightly on the aspects of their geological occurrence, and rightly so, for the subject is far too large to be dealt with properly in a book of this size. He concentrates mostly on detailed descriptions of the technique of boring, pitting and treatment of samples. On these subjects he gives a wealth of detail rarely, if ever, published previously, and the book is clearly the work of a man who has had long experience of actual field work. The subject of the valuation of bauxite, manganese and surface deposits of a kindred nature is also discussed.

The book is fully illustrated with figures and plates of equipment and methods, and has as well a comprehensive number of appendices and tables to cover most of an engineer's requirements. The photographic plates, whilst not up to pre-war standards, are clearer than in many war-time publications, whilst the printing is clear and the line diagrams excellent.

It is certainly a book which should be included in the library of every engineer engaged in this class of work, and its usefulness to the man in the field is enhanced by its portability.

W. W. VARVILL.

Principles of Field and Mining Geology. By J. D. FORRESTER. New York: John Wiley & Sons, Inc., 1946. 647 p., 316 figs. $7, 42s.

This instructive book makes an opportune appearance at a time of vital need for mineral exploration and the discovery of new orebodies. It is written on the premise that the reader is already familiar with the elementary principles of geology and mineralogy, and the treatment is primarily concerned with the field technique of mining geology.

Part I deals with the recognition and correlation of geological phenomena such as faults, cleavage, and folded structures, and includes a sketchy description of the nature of economic mineral deposits. Part II gives a clear account of general field-survey practice and procedure, including methods of determining dip and strike, measuring distances, and conducting Brunton compass and alidade-plane table surveys. The third part is of special concern to the mining geologist, for it discusses underground mapping, the technique of sampling, drilling methods, and the application of geophysical means of exploration, though the English reader may find little interest in the section dealing with Federal mining statutes in the United States. Part IV deals with the interpretation and use of field data, including the preparation of geological maps and sections, the analysis of structural controls of ore deposition, the determination of minable ore limits, the treatment of sample-assay analyses, the computation of ore reserves, and the submission of formal geological reports on individual mines and prospects.

In attempting to cover such a wide field it is not surprising that certain aspects of the subject receive rather scant attention, but one might reasonably have expected more detail on methods of mapping and recording

BOOK REVIEWS—*continued.*

surface geology. Nevertheless, the book is a notable addition to the few available volumes dealing with the field technique of mining geology. It is well written and organized, illustrated with clear diagrams, and can be highly recommended especially to anyone actively interested in the search for mineral deposits and in their appraisal.

DAVID WILLIAMS.

OBITUARY.

Charles Thomas Andrews died at Middlesex Hospital, London, on September 3rd, 1946, at the age of 62. He began his career in 1902 as a pupil at the London laboratory of Mr. Edward Riley, where he was employed on the assaying of minerals and metals, and in 1906 was appointed head chemist in charge of the laboratory attached to Carlos Wigg's manganese and iron ore mines at Minas, Brazil. He held this position for four years, also restarting and managing the local foundry, and then was made mine captain and assistant to the mine manager. In 1913 Mr. Andrews took up employment with Asia Minor Concessions, Ltd., as mining assistant engaged on the development of various deposits in Turkey, and after a few months he went to Spain, prospecting for Diana Mines, Ltd., Jaen. During the 1914–1918 war he served at Gallipoli with the Royal Naval Division Engineers, and with commissioned rank in 174 and 252 Tunnelling Companies, R.E., in France. He returned to England in 1918 and rejoined Messrs. Riley, Harbord and Law as assayer and mineral chemist, where he remained for twelve years. In 1931 and 1932 Mr. Andrews worked in Eire for the British (Non-Ferrous) Mining Corporation, Ltd., and in 1933 went to West Africa and was for three years assistant engineer to the African Manganese Co., Ltd., on the production and assaying staff. He then took up the position of metallurgist with Ustipraca Mining Co., Ltd., in Yugoslavia, and returned to England early in 1939. In recent years failing health prevented him from practising his profession.

Mr. Andrews was elected to Associateship of the Institution in 1910.

Charles Lawrence Butlin died in Johannesburg on August 29th, 1946, at the age of 67. He was a student at the Camborne School of Mines from 1896 to 1900, and in December, 1900, he took up the appointment, which he held for three years, of engineer-in-charge of a manganese mine in India controlled by the Central Provinces Prospecting Syndicate. In 1904 he went to South Africa as sampler to the New Heriot Gold Mining Co., Ltd., and later joined the staff of Ferreira Gold Mining Co., Ltd. In May, 1910, he took up the post of mine overseer and acting manager to Modderfontein B Gold Mines, Ltd., and four years later was appointed manager. He served in this capacity from November, 1914, to December, 1935, leaving to become consulting engineer (mining) to Central Mining and Investment Corporation, Ltd., in Johannesburg. He retired from this position at the end of 1945 and had continued to live in Johannesburg until his death. He was elected a Member of the Institution in 1937.

Gerald Noel Carroll died on 15th August, 1944, at the age of 53. He was educated at Bedford Grammar School and went to Germany in 1904 to Neuenheim College, Heidelberg, and was at Heidelberg University from

OBITUARY—*continued.*

1907 to 1908. He spent the following year in Hanover, and in 1910 began his mining career at Knights Deep, Ltd., Germiston, Transvaal, where he remained nine years, during the last two of which he held the position of mine captain. He was manager of asbestos mines at Pietersburg from 1919 to 1922, and during the latter year acted as efficiency officer to the Witwatersrand Gold Mining Co. in Germiston. He became manager successively of Spes Bona Gold Mining Co., and Geldenhuis Tributors, Johannesburg, and from May, 1924, was employed for six months as geologist to Messrs. Rosehaugh & Co., of Dar-es-Salaam, Tanganyika Territory. In August, 1925, Mr. Carroll took up the appointment of manager in charge of field operations with Northern Platinum Exploration, Ltd., in the Transvaal, and left their employment at the end of 1926 to join the staff of Luipaardsvlei Estate and Gold Mining Co., Ltd., at Krugersdorp as underground manager and later as sectional manager and acting general manager. He resigned early in 1928 owing to ill health, but for six months from the end of the year held the position of resident engineer at the Alaska copper mine in Southern Rhodesia. He then worked in Northern Rhodesia as geologist to Rhodesian Congo Border Concession, Ltd., until 1931, and from 1932 to 1933 was employed by Consolidated Main Reef Mines and Estate, Ltd. After holding the managership of Babrasco gold mines at Klerksdorp for a few months he set up in practice as consulting mining engineer in Johannesburg. He was elected to Associateship of the Institution in 1935.

Hubert Cartwright died on July 21st, 1946. He received his professional training at the Royal School of Mines from 1890 to 1894, and obtained an Associateship in Metallurgy. He first went to Southern Rhodesia, where he remained for the whole of his professional career, in 1895 as assistant mining engineer to Goldfields of Matabeleland, Ltd. From 1903 to 1904 he was mine manager of Eldorado mine in Lomagundi, and a year later became mine manager of Mandora mine and subsequently of Simoona mine, Mazoe District. For three years from 1906 Mr. Cartwright worked for British United Great Bear and Dalny mines in the Hartley District, and in 1909 and 1910 was prospecting in the Shamva District. He then was engaged for two years as a mining contractor before taking up the position in 1913 of shift boss at the Owl mine. He served as a gunner in the H.A.C. from 1917 to 1918, but resumed his career in 1919 as mining engineer and manager of Lomah (Rhodesia) Mining Co., Ltd., which position he held for eight years. In 1927 he was appointed chrome contractor to the Rhodesia Vanadium Co., Ltd., and from 1929 managed Mapeke Asbestos Mines, Ltd. In recent years he worked mines on his own account.

Mr. Cartwright was elected to Associateship of the Institution in 1906.

Ernest Gerald Dustow, of the South African Air Force, is reported to have been killed while on active service in North Africa. He was educated at Redruth County School from 1928 to 1935, and received his professional training at Camborne School of Mines from 1935 to 1938. On completing his course he went to the Transvaal as official learner at Brakpan Mines, Ltd., and during his two years there worked on mine sampling, valuation, and surveying. He joined the South African Air Force on July 1st, 1940,

OBITUARY—*continued.*

and was subsequently reported missing. It is believed that he was shot down over Tobruk. Mr. Dustow was elected a Student of the Institution in 1938.

Willem Johannes Dirk Kloezeman died in hospital about July, 1945, at the age of 54, after internment by the Japanese in Java. Of Dutch parentage, he was apprenticed in 1912 to Messrs. Werf-Conrad, Ltd., of Haarlem, Holland, but left in March, 1915, to join the Singkep Tin Mining Co., Ltd., at Singkep, Malacca Straits. In August, 1918, he took up the appointment of assistant manager at Tanjong (Rambutan), Ltd., for Messrs. Osborne & Chappel, and in 1919 transferred to the post of division manager at Kinta Tin Mines, Ltd., Gopeng. After a few months he was appointed Malayan representative of Messrs. Werf-Conrad, Ltd. He stayed with this firm for seven years and at the same time carried on various mining engineering activities. He acted as consulting mining engineer from 1921 to 1924 for Dr. W. A. Rogers, was managing partner to Foothills Tin Syndicate from 1923 to 1924, and consulting engineer for Sungei Rias Tin, Ltd., and Sungei Hujan Tin, Ltd. In March, 1926, Mr. Kloezeman joined the firm of Lindeteves-Stovis, Inc., engineers and importers, and was in charge of their mining and engineering activities in the Federated Malay States and Siam. Four years later he took over the management of Kay-Yew (Kinta Valley) Tin Mines, Ltd., at Menglembu, Perak, F.M.S., which was later reconstructed as Kay Tin Mines (Kinta), Ltd., of which he was general manager. He held this position for many years and continued to work in Malaya until the Japanese invasion. No news had been received of him until it was learned that he had been in a Japanese internment camp in Java and had died from exhaustion.

Mr. Kloezeman was elected to Associateship of the Institution in 1933.

Professor James Park, doyen of the mining profession in New Zealand, died on July 29th, 1946, at Oamaru, at the age of 89. He was born in Aberdeen and received his professional training at the Royal School of Mines between 1872 and 1874. He went immediately afterwards to New Zealand but, finding no suitable employment there, joined the ranks of the Gordon Highlanders in India. He returned to New Zealand after a very brief army career, during which he was wounded, and was appointed to the Geological Survey Department as assistant geologist in 1878. Twelve years later he took up the position of Director of Thames School of Mines and Superintendent of the Government Experimental Metallurgical Works. He relinquished this appointment to act as consulting engineer to various companies from 1896, but in 1901 was appointed Professor of Mining and Mining Geology at Otago University. He was later made Dean of the Faculty of Mining and Economic Geology, and on his retirement in 1932 was appointed Professor Emeritus. Professor Park was the author of many papers and textbooks, including three papers published in the *Transactions* of the Institution—'Notes on the action of cyanogen on gold' (vol. 6, 1897–8), 'Notes on the coalfields of New Zealand' (vol. 8, 1899–1900), and 'On the course of border-segregation in some igneous magmas' (vol. 14, 1904–5). He was elected a Member of the Institution in 1896, and in 1931 was elected to Honorary Membership in recognition of his great service to mining education.

OBITUARY—*continued.*

Professor G. J. Williams writes: Professor Park belonged to the old school before the days of specialization, and his interests were exceptionally diverse. He became a leader in scientific societies in New Zealand, and his scientific status coupled with an unusually sound judgment caused him to be recognized as one of the leading mining engineers and mining geologists of his day, and he was called upon to report on mining properties as far afield as New Caledonia, Canada, and Spain.

Professor Park's retentive memory, his sense of humour, and his never-failing optimism made him a delightful conversationalist and an agreeable companion. He loved to entertain his students and to recount to them stories of his youth: he kept young by his association with the younger generation. He has a large family and his youngest son later became world famous: he is Air Chief Marshal Sir Keith Park, who is so well known for his part in the Battle of Britain, as the air defender of Malta, and later as Allied Air Commander-in-Chief, South-East Asia Command. It was as if a life of exceptional fullness and usefulness were crowned by the visit of Sir Keith and Lady Park to him shortly before he passed away. The anticipation of this visit had sustained him through many months of illness.

Professor Park's portrait is hung in the Scottish National Gallery at Edinburgh among those of other distinguished Scotsmen.

The Council regret to report the death of **Walter Broadbridge,** *Member,* on October 23rd, 1946; **Arthur Gibb,** *Associate,* on February 13th, 1942; **Anthony Charles Bovill Malcolm-Sim,** *Associate,* on October 30th, 1946; and **John Waters Sutherland,** *Member,* on September 26th, 1946. Obituary notices will appear in a later *Bulletin.*

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

The address of the Library is now 424 Salisbury House, London, E.C. 2, and books (excluding periodicals) may be borrowed by members on personal application or by post.

American Institute of Mining and Metallurgical Engineers: Transactions. Vol. 164, Geophysics, 1945. N.Y.: The Institute. 1946. 426 p.

Australia. Council for Scientific and Industrial Research: 19th Annual Report for the Year ended 30th June, 1945. Canberra: Govt. Printer. 1945. 164 p. 7s.

British Standards Institution: Yearbook. London: The Institution. 1946. 296 p. 2s.

Colliery Explosions and Recovery Work. A Textbook for Candidates for the First-Class Managers' and Under-Managers' Certificates, 2nd ed., by J. W. Whitaker and H. L. Willett. London: Pitman. 1946. 12s. 6d. (*Presented by the Publishers.*)

LIST OF ADDITIONS TO THE JOINT LIBRARY—*continued.*

EXAMINATION, BORING AND VALUATION OF ALLUVIAL AND KINDRED ORE DEPOSITS, by H. L. H. Harrison. London: Mining Publications. 1946. x, 264 p. 30s.

EXPERIMENTS ON COKING PRACTICE: THE EFFECT OF TYPE OF OVEN UPON THE PROPERTIES OF THE COKE. Department of Scientific and Industrial Research, Fuel Research. London: H.M.S.O. 1946. 37 p. 25 plates. 4s. (*Presented by the Director of Fuel Research.*)

GEOCHEMICAL SURVEY OF THE UNDERGROUND WATER SUPPLIES OF THE UNION OF SOUTH AFRICA, by G. W. Bond. South Africa Geological Survey Memoir No. 41. Pretoria: Geological Survey. 1946. 216 p., maps. 10s.

GEOLOGY OF THE MESSINA COPPER MINES AND SURROUNDING COUNTRY, by P. G. Söhnge. South Africa Geological Survey Memoir No. 40. Pretoria: Geological Survey. 1945. 280 p., maps. £1 1s.

HANDBOOK OF PROCEDURE AND EVIDENCE IN ARBITRATION, by W. T. Creswell and Norman P. Greig. London: Eyre and Spottiswood. 1946. 175 p. 10s. 6d.

INSTITUTION OF MECHANICAL ENGINEERS: PROCEEDINGS, WAR EMERGENCY ISSUES 1–12. VOL. 153. London: The Institution. 1945. 517 p.

INTERNATIONAL GLOSSARY OF ACADEMIC TERMS. Paris: International Federation of University Women. 1939. 755 p. 10s. 6d.

INTRINSIC SAFETY OF ELECTRICAL APPARATUS; THE MINIMUM IGNITING CURRENT IN RELATION TO CIRCUIT CONSTANTS, by G. Allsop, E. M. Guénault, and A. D. I. Nicol. Safety in Mines Research Board Paper No. 104. London: H.M.S.O. 1946. 15 p. 6d.

IRON AND STEEL INSTITUTE: JOURNAL. VOL. 151. London: The Institute. 1945. 570+161+44 p.

LAW OF TRADE UNIONS, 2nd ed., by H. Samuels. London: Stevens. 1946. 96 p. 6s. (*Presented by the Publishers.*)

MATERIALS HANDLING MANUAL. VOL. 2. M. M. Williamson and G. W. Williamson, *eds.* London: Elek. 1945. 488 p., illus. £1 10s.

ONTARIO DEPARTMENT OF MINES: ANNUAL REPORT. VOL. 52, PT. 1, 1943. Toronto: Govt. Printer. 1946. 277 p., tables.

PRATIQUE MINIERE BRITANNIQUE, 1939 à 1945, by N. E. Webster. (Paper presented at the Conference given by the Société Royale Belge des Ingénieurs et des Industriels, 18 May 1946.) 9 p. (*Presented to the Institution of Mining Engineers by the Author.*)

PRINCIPLES OF FIELD AND MINING GEOLOGY, by James Donald Forrester. N.Y.: Wiley. 1946. 647 p. £2 2s.

LIST OF ADDITIONS TO THE JOINT LIBRARY—*continued*.

Report on the Restoration Problem in the Ironstone Industry in the Midlands. Summary of Findings and Recommendations. Ministry of Town and Country Planning. London: H.M.S.O. 1946. 17 p. 3d.

Reports of the British and Combined Intelligence Objectives Sub-Committees (B.I.O.S. and C.I.O.S.), Field Information Agency, Technical (F.I.A.T.), and Joint Intelligence Objectives Agency (J.I.O.A.). Various reports have been presented by the Department of Scientific and Industrial Research and are listed under the Index of Recent Articles.

Statistical Digest, 1945. Ministry of Fuel and Power. London: H.M.S.O. Oct., 1946. 114 p. 2s.

Surveying of Trunk Conveying in the Nottinghamshire and Derbyshire Coalfields. The Midland Counties Colliery Owners' Association. Derby: The Association. 1946. 53 p. (*Presented to the Institution of Mining Engineers by the Association.*)

Union of South Africa. Report upon the Work of the Miners' Phthisis Medical Bureau for the Three Years ended 31st July, 1944. Pretoria: Govt. Printer. 1946. 52 p. 7s. 6d.

MAPS.

Argentine. Camara Argentina de mineria. Principal minerals and rocks for industrial purposes. Scale: 210 km. = 1 in. Buenos Aires: Industria Minera. 1946.

Bourlamaque (South West), Abitibi County, Quebec. Geological Survey Preliminary Map 46–17. Scale: 1 in. = 1,000 ft. Ottawa: Bureau of Geology and Topography. 1946.

Callum Creek, Alberta. Geological Survey Preliminary Map 46–10. Scale: 2 in. = 1 ml. Ottawa: Bureau of Geology and Topography. 1946.

Manitoba. Geological map of Manitoba. Map 850A. Scale: 1 in. = 20 ml. Ottawa: Bureau of Geology and Topography. 1946.

Manitoba. Mineral map of Manitoba. Map 851A. Scale: 1 in. = 20 ml. Ottawa: Bureau of Geology and Topography. 1946.

Ottawa—Cornwall, Ontario and Quebec. Geological Survey Map 852A. Scale: 1 in. = 4 ml. Ottawa: Bureau of Geology and Topography. 1946.

Strait of Canso, Nova Scotia. Geological Survey Preliminary Map 46–12A and Paper 46–12. Scale: 2 in. = 1 ml. Ottawa: Bureau of Geology and Topography. 1946.

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.

INDEX OF RECENT ARTICLES—*continued.*

INDEX OF RECENT ARTICLES—*continued.*

INDEX OF RECENT ARTICLES—*continued.*

ECONOMICS—*continued.*

HEALTH OF MINERS — SILICOSIS — PREVENTION—ALUMINIUM THERAPY —MEXICO.—Terapia de aluminio en Parral, Mexico.—*Bol. Soc. Nac. Min. Peru*, Lima, Vol. 2, April, 1946, pp. 9–16.

LABOUR — WAGES — TRANSVAAL. — Mine wages negotiations.—*S. Afr. Min. Engng. J.*, J'burg., Vol. 57, Pt. 2, Sept. 7, 1946, pp. 5–7. 6d.

MINE INSPECTION—OPENCAST MINES —U.S.A.—Inspection standards for strip mines (coal and lignite) revised October, 1945. —*U.S. Bur. Min. Inform. Circ.* No. 7350, Wash., D.C., March, 1946, 32 p.

PRODUCTION — COAL — AUSTRALIA. — Australia's coal industry: attempt to improve production.—*S. Afr. Min. Engng. J.*, J'burg., Vol. 57, Pt. 1, July 20, 1946, pp. 525–7. 6d.

PRODUCTION — COAL — FRANCE.—The rising French coal output. W. R. Gordon.—*Colliery Guard.*, Lond., Vol. 173, Aug. 16, 1946, pp. 213–15. 1s.

PRODUCTION — GOLD — TRANSVAAL. —The gold industry's position.—*S. Afr. Min. Engng. J.*, J'burg., Vol. 57, Pt. 1, Aug. 3, 1946, pp. 561–3. 6d.

TECHNICAL PAPERS.—The preparation of technical papers. W. E. Clegg.—*J. Instn. Engrs. Aust.*, Sydney, Vol. 18, June, 1946, pp. 135–39. 7s.

GEOLOGY AND ORE DEPOSITS.

GEOLOGY — ALASKA — COPPER RIVER REGION — SPIRIT MT.—Nickel-copper prospect near Spirit Mountain Copper river region, Alaska. Jack Kingston and Don J. Miller.—*Bull. U.S. Geol. Surv.* 943–C, Wash., D.C., 1945, 9 p. 10 cents.

GEOLOGY—DIP AND STRIKE—DIP DETERMINATION.—A cotangent ruler for simplifying the graphic solution of problems in structural geology. Walter S. White.—*Econ. Geol.*, Lancaster, Pa., Vol. 41, Aug., 1946, pp. 539–45. $1.

GEOLOGY — ECUADOR — GUAYAQUIL ESTUARY.—The geology of the Guayaquil estuary, Ecuador. George Sheppard.—*J. Inst. Petrol.*, Lond., Vol. 32, Aug., 1946, pp. 492–514. 7s. 6d.

GEOLOGY—FAULTS.—The problem of faults.—*Iron Coal Tr. Rev.*, Lond., Vol. 153, Aug. 16, 1946, pp. 265–67. 9d.

GEOLOGY—MALAYA.—The relationship of the geological survey to the mining industry of Malaya. E. S. Willbourn.—*Imp. Inst. Bull.*, Lond., Vol. 44, April–June, 1946, pp. 128–48. 2s. 6d.

GEOLOGY—MALAYA.—The stratigraphy and structure of the arenaceous formation of the Main Range Foothills, F.M.S. J. A. Richardson.—*Geol. Mag.*, Lond., Vol. 83, Sept.–Oct. 1946, pp. 217–29. 5s.

GEOLOGY AND ORE DEPOSITS—*continued.*

GEOLOGY — MEXICO — GUERRERO — HUITZUCO.—Quicksilver-antimony deposits of Huitzuco, Guerrero, Mexico. James F. McAllister and David Hernandez Ortiz.—*Bull. U.S. Geol. Surv.* 946–B, Wash., D.C., 1945, 23 p., maps. 75 cents.

GEOLOGY — NORTHWEST TERRITORIES — ELDORADO. — The Eldorado enterprise. By members of the staff.—*Trans. Canad. Inst. Min. Metall.*, Vol. 49, 1946, pp. 423–38 (*Canad. Min. Metall. Bull.*, Montreal, No. 413, Sept., 1946. $1).

GEOLOGY—NOVA SCOTIA—STRAIT OF CANSO.—Strait of Canso map area, Inverness, Richmond, Guysborough, and Antigonish Counties, Nova Scotia. (Summary account.) S. A. Ferguson.—*Canad. Geol. Surv. Pap.* 46–12, Ottawa, 1946, 23 p.

GEOLOGY — OREGON. — Chromite-bearing sands of the southern part of the coast of Oregon. Allan B. Griggs.—*Bull. U.S. Geol. Surv.* 945–E, Wash., D.C., 1945, 37 p., maps. 55 cents.

GEOLOGY—POLAND (EAST).—The geological structure of East Poland and West Russia: a summary of recent discoveries. Zbigniew Sujkowski.—*Quart. J. Geol. Soc.*, Lond., Vol. 102, July 31, 1946, pp. 189–201. 10s.

GEOLOGY — QUEENSLAND — CLONCURRY AREA.—Geological sketch map, Cloncurry area.—*Qd. Govt. Min. J.*, Brisbane, Vol. 47, May, 1946, pp. 144–46. 6d.

GEOLOGY—RUSSIA (WEST).—The geological structure of East Poland and West Russia: a summary of recent discoveries. Zbigniew Sujkowski.—*Quart. J. Geol. Soc.*, Lond., Vol. 102, July 31, 1946, pp. 189–201. 10s.

GEOLOGY—VEIN CARBONATES.—Microscopic character of vein carbonates. Frank F. Grout.—*Econ. Geol.*, Lancaster, Pa., Vol. 41, Aug., 1946, pp. 475–502. $1.

MINERAL RESOURCES — CHINA. — Mineral resources of China. Vei Chow Juan. —*Econ. Geol.*, Lancaster, Pa., Vol. 41, June-July, Pt. 2, 1946, pp. 399–474. $1.

MINERAL RESOURCES—FRENCH EMPIRE.—Mineral resources of the French Empire: Pts. 2 and 3 (*concluded*). Philip F. Dyer.—*Min. J.*, Lond., Vol. 227, 1946; Aug. 17, pp. 649–51; Oct. 5, pp. 790–91. 8d.

MINERAL RESOURCES — U.S.A. — Ore-reserve viewpoints. S. G. Lasky.—*Min. Metall.*, N.Y., Vol. 27, Sept., 1946, pp. 466–68. 50 cents.

MINERAL RESOURCES—U.S.A.—Whence our future ore reserves? George M. Fowler.—*Min. Congr. J.*, Wash., D.C., Vol. 32, May, 1946, pp. 34–5. 20 cents.

INDEX OF RECENT ARTICLES—*continued.*

INDEX OF RECENT ARTICLES—*continued.*

IRON AND STEEL—*continued.*

SMELTING — BLAST FURNACES — CHILLED HEARTHS.—Opening a chilled blast-furnace hearth. R. P. Towndrow.—*Iron Coal Tr. Rev.*, Lond., Vol. 153, Aug. 30, 1946, pp. 349–52. 9d.

LEAD AND ZINC.

GENERAL.—The great lead and zinc mines. Walter Renton Ingalls.—*Min. Metall.*, N.Y., Vol. 27, Sept., 1946, pp. 469–73. 50 cents.

ORE DRESSING.—The dressing of simple lead and zinc ores. (Flow sheets.) F. B. Michell.—*Mine & Quarry Engng.*, Lond., Vol. 12, Oct., 1946, pp. 105–14. 1s.

UTAH—BEAVER CO.—WAH WAH MINE. —Examination of the Wah Wah lead-zinc mine, Beaver county, Utah. Robert L. Jones and W. Clifford Dunham.—*U.S. Bur. Min. Rep. Invest.* No. 3853, Wash., D.C., April, 1946, 14 p.

ZINC — COLORADO—SUMMIT CO.—BIG FOUR MINE.—Exploration of the Big Four zinc-silver mine, Summit county, Colo. R. B. McCulloch and W. P. Huleatt.—*U.S. Bur. Min. Rep. Invest.* No. 3884, Wash., D.C., April, 1946, 7 p.

ZINC—TENNESSEE.—An open-pit zinc mine in North Eastern Tennessee. William Black and Ernest L. Ohle.—*Engng. Min. J.*, N.Y., Vol. 147, Aug., 1946, pp. 91–3. 50 cents.

ZINC—TENNESSEE.—Exploration of Stiner and Bunch Hollow zinc properties, Powell River area, Union and Clairborn counties, Tennessee. Richard L. Sayrs.—*U.S. Bur. Min. Rep. Invest.* No. 3856, Wash., D.C., April, 1946, 4 p.

LIGHT METALS.

ALUMINIUM—ALUMINA.—Production of alumina; new advances and developments: Pt. 2. D. D. Howat.—*Mine & Quarry Engng.*, Lond., Vol. 12, Sept., 1946, pp. 73–9. 1s.

BERYLLIUM—GERMANY.—The beryllium industries of Germany and Italy (1939–1945).—F.I.A.T. Final Report No. 522, 1946, 102 p.

BERYLLIUM—ITALY. — The beryllium industries of Germany and Italy (1939–1945).—F.I.A.T Final Report No. 522, 1946, 102 p.

BERYLLIUM — ONTARIO — LYNDOCH TOWNSHIP.—The Lyndoch beryl deposit. D. C. McLaren.—*Canad. Min. J.*, Gardenvale, P.Q., Vol. 67, Sept., 1946, pp. 857–62. 25 cents.

LIGHT METALS — GERMANY. — German aluminium and magnesium industries.—F.I.A.T. Final Report No. 501, Nov., 1945, 128 p.

LIGHT METALS—*continued.*

MAGNESIUM—METALLURGY — LEACHING.—The bicarbonate process for the production of magnesium oxide. H. A. Doerner, W. Floyd Holbrook, and Otis W. Fortner.—*U.S. Bur. Min. Tech. Pap.* 684, Wash., D.C., 1946, 48 p. 10 cents.

METALLURGY AND CYANIDING.

METALLURGY — GERMANY — German metallurgical laboratories for ferrous metals with special reference to the K.W. institute for iron research.—B.I.O.S. Final Report No. 676, Item No. 21, 1945, 101 p.

METALLURGY — GERMANY (N.W.). — Copper, lead, zinc, tin, and antimony smelting and refining in North Western Germany.—F.I.A.T. Final Report No. 239, Oct., 1945, 47 p.

METALLURGY — POWDER METALLURGY.—Métallurgie des poudres. Robert Girschig.—*Mem. Soc. Ing. Civ. Fr.*, Paris, Jan.-April, 1946, pp. 1–28.

METALLURGY — POWDER METALLURGY.—Powder metallurgy. J. E. Newson.—*Proc. Instn. Mech. Engrs.*, Lond., Vol. 154, No. 2, 1946, pp. 208–18.

METALLURGY — POWDER METALLURGY.—Principes et possibilités de la métallurgie des poudres. Monsieur Zeicher.—*Rev. Univ. Min.*, Liége, 9th series, Vol. 2, No. 6, 1946, pp. 234–43. 60 frs.

METALLURGY — REFINING — ELECTROLYTIC.—Production of metals by electrolytic refining. F. A. Forward.—*Deco Trefoil*, Denver, July, 1946, 4 p.

MILLING AND CONCENTRATION.

CLASSIFICATION.—Screening and classification in grinding circuits. S. Gray and P. M. Elliott.—*Trans. Canad. Inst. Min. Metall.*, Vol. 49, 1946, pp. 466–73 (*Canad. Min. Metall. Bull.*, Montreal, No. 413, Sept., 1946. $1).

COMMINUTION — CRUSHING. — Small crushing plants.—*Min. Industr. Mag. S. Afr.*, J'burg., Vol. 34, July, 1946, pp. 441–5. 1s.

COMMINUTION — CRUSHING — CRUSHING STATIONS.—Underground crushing station at New Occidental.—*Chem. Engng. Min. Rev.*, Melb., Vol. 38, May 10, 1946, pp. 265–8. 1s.

COMMINUTION — CRUSHING — STONE. —Researches on stone crushing with particular regard to the shape of the product. Demetrius Rösslein (Abridged translation by F. A. Shergold).—*Quarry Mgrs.' J.*, Lond., Vol. 30, Oct., 1946, pp. 207–22. 1s.

CONCENTRATION — JIGGING — HARZ JIG.—Corundum concentration in the field. Tests in use of Harz jig. J. Levin.—*S. Afr. Min. J.*, J'burg., Vol. 57, Pt. 1, July 27, 1946, pp. 549–55. 6d.

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INDEX OF RECENT ARTICLES—*continued.*

MINING (General)—*continued.*

SURVEYING — AERIAL — MAGNETIC.—Controlling plane position in aerial magnetic surveying. Homer Jensen and J. R. Balsley, Jr.—*Engng. Min. J.*, N.Y., Vol. 147, Aug., 1946, pp. 94–5. 50 cents.

SURVEYING — THEODOLITES. — The choice of a mining theodolite. O. F. Wooding. —*Univ. Durh. King's Coll. Min. Soc. J.*, July, 1946, pp. 36–50. 5s.

TRANSPORTATION — NORTHWEST TERRITORIES. — Transportation in the north. A. M. Berry.—*W. Miner*, Vancouver, B.C., Vol. 19, Aug., 1946, pp. 43–6. 25 cents.

TUNNELLING — BLASTING. — Tunnel drilling and blasting. J. W. Fidoe.—*Compr. Air Hydr. Engng.*, Lond., Vol. 11, Sept., 1946, pp. 125–27. 1s.

TUNNELLING — LINING — COLORADO —ALVA B. ADAMS TUNNEL.—Colorado and Big Thompson Rivers Scheme: Pt. 4, Lining the Alva B. Adams tunnel.—*Engineer*, Lond., Vol. 182, Sept. 20, 1946, pp. 247–8. 1s.

VENTILATION.—Recent developments in mine ventilation. M. T. Adamtchik.—*Univ. Durh. King's Coll. Min. Soc. J.*, July, 1946, pp. 14–35. 5s.

VENTILATION—AIR CONDITIONING.—Mining and equipment problems: Pt. 5, Ventilation and air conditioning.—*Gold Min. Rec.*, Lond., Vol. 15, Aug., 1946, pp.200–01. 1s.

VENTILATION — AIR CONDITIONING.—Ventilation and air conditioning: Pt. 5.—*Gold Min. Rec.*, Lond., Vol. 15, Sept., 1946, pp. 226–7. 1s.

VENTILATION — AIR FLOW — ANEMOMETERS.—Tarage et utilisation des anémomètres à ailettes. André Jorissen and Pierre Ledent.—*Rev. Univ. Min.*, Liége, 9th series, Vol. 2, No. 7, 1946, pp. 249–64. 60 frs.

VENTILATION — DEEP MINES. — An experimental investigation of the effects of high temperatures on the efficiency of workers in deep mines. Anthony Caplan and J. K. Lindsay.—*Inst. Min. Metall. Bull.* No. 480, Lond., Sept., 1946, 30 p. 1s. 9d.

VENTILATION — DUST — CONCENTRATION.—Increased ventilation—its effect on dust concentration in mine air. A. S. Richardson, J. W. Warren, and W. C. Williamson.—*Min. Congr. J.*, Wash., D.C., Vol. 32, Sept., 1946, pp. 40–1. 20 cents.

VENTILATION — SHAFTS — DESIGN. — Determination of most economical airshaft size. Raymond Mancha.—(*A.I.M.E. Tech. Pub.* No. 1983) *Coal Tech.*, N.Y., Vol. 1, May, 1946, 6 p. $1.60.

VENTILATION — SURVEYING. — Ventilation surveying technique: Pt. A.—The measurement of sectional areas underground by a photographic method. Pt. B.—A two-liquid differential manometer for the measurement of pressure-drops along underground roadways. Edward L. J. Potts.—*Trans. Instn. Min. Engrs.*, Lond., Vol. 105, Pt. 11, Aug., 1946, pp. 631–50. 6s.

MINING (General)—*continued.*

VENTILATION — SYSTEMS — DESIGN.—Basic design factors for ventilation plant: Pt. 1. 'Industrialist'.—*Industr. Heat. Engr.*, Lond., Vol. 8, Oct., 1946, pp. 144–47. 1s.

WATER SUPPLY — WITWATERSRAND (FAR WEST).—Far West Rand water scheme. —*S. Afr. Min. Engng. J.*, J'burg., Vol. 57, Pt. 2, Sept. 7, 1946, p. 9. 6d.

WELDING.—Electric arc welding in colliery practice. J. W. Pirie.—*Min. Elect. Mech. Eng.*, Manchr., Vol. 27, Sept., 1946, pp. 105–11. 2s. 6d.

YUGOSLAVIA — KAPAONIK. — Balkan base-metal enterprise—Trepca Mines, Ltd.—*Engng. Min. J.*, N.Y., Vol. 147, Sept., 1946, pp. 54–60. 50 cents.

MINOR METALS.

ABRASIVES — CORUNDUM — CONCENTRATION.—Corundum concentration in the field. Tests in use of Harz jig. J. Levin.—*S. Afr. Min. Engng. J.*, J'burg., Vol. 57, Pt. 1, July 27, 1946, pp. 549–55. 6d.

ANTIMONY.—Quicksilver-antimony deposits of Huitzuco, Guerrero, Mexico. James F. McAllister and David Hernandez Ortiz.—*Bull. U.S. Geol. Surv.* 946–B., Wash., D.C., 1945, 23 p., maps. 75 cents.

ANTIMONY — METALLURGY — SMELTING—GERMANY.—Antimony smelting industry.—B.I.O.S. Final Report No. 328, Item No. 21, Nov., 1945, 9 p.

BARIUM — BARYTES — SWAZILAND. — Barytes in Swaziland. A. T. M. Mehliss.—*S. Afr. Min. Engng. J.*, J'burg., Vol. 57, Pt. 1, July 6, 1946, pp. 465–7. 6d.

CERIUM.—Cerium and thorium. E. F. Roots.—*W. Miner*, Vancouver, B.C., Vol. 19, Aug., 1946, pp. 50–6. 25 cents.

CERIUM—GERMANY.—The cerium industry in German territory, including reports on radium and mesothorium.—B.I.O.S. Final Report No. 400, Item No. 21, Sept., 1945, 73 p.

CHROMIUM—DUMP SURVEYING.—Notes on the estimation of tonnage and grade of some chromite dumps. N. W. Wilson.—*Instn. Min. Metall. Bull.*, Lond., No. 480, Sept., 1946, 14 p. 1s. 6d.

CHROMIUM—OREGON.—Chromite-bearing sands of the southern part of the coast of Oregon. Allan B. Griggs.—*Bull. U.S. Geol. Surv.* 945–E, Wash., D.C., 1945, 37 p., maps. 55 cents.

MANGANESE — GEORGIA — WILKES CO.—Exploration of the Piedmont manganese belt, McCormick county, South Carolina, and Wilkes county, Georgia. William A. Beck.—*U.S. Bur. Min. Rep. Invest.* No. 3858, Wash., D.C., April, 1946, 4 p.

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MINOR METALS—continued.

MANGANESE — MINING METHODS — OPENCAST—VIRGINIA.—Open pit mining for manganese. D. E. Eilertsen and J. W. Pains.—*Min. Congr. J.*, Wash., D.C., Vol. 32, Sept., 1946, pp. 28–39. 20 cents.

MANGANESE — SOUTH AFRICA. — The Union's manganese resources.—*S. Afr. Min. Engng. J.*, J'burg., Vol. 57, Pt. 1, July 27, 1946, pp. 537–9. 6d.

MANGANESE — SOUTH CAROLINA — McCORMICK CO.—Exploration of the Piedmont manganese belt, McCormick county, South Carolina, and Wilkes county, Georgia. William A. Beck.—*U.S. Bur. Min. Rep. Invest.* No. 3858, Wash., D.C., April, 1946, 5 p.

MERCURY. — Quicksilver-antimony deposits of Huitzuco, Guerrero, Mexico. James F. McAllister and David Hernandez Ortiz.—*Bull. U.S. Geol. Surv.* 946–B, Wash., D.C., 1945, 23 p., maps. 75 cents.

NICKEL.—Nickel's place in the world.—*S. Afr. Min. Engng. J.*, J'burg., Vol. 57, Pt. 1, Aug. 17, 1946, pp. 685–86. 6d.

NICKEL — METALLURGY — SMELTING. —Nickel blast-smelting still offers advantages. C. C. Downie.—*Engng. Min. J.*, N.Y., Vol. 147, Aug., 1946, pp. 88–90. 50 cents.

PLATINUM — GERMANY.—The platinum metals industry in Germany.—B.I.O.S. Final Report No. 441, Item No. 21, Nov., 1945, 68 p.

RHENIUM — GERMANY. — Rhenium. — F.I.A.T. Final Report No. 697, 1946, 4 p.

TANTALUM — WESTERN AUSTRALIA —WODGINA.—Tantalite in Western Australia.—*Min. Mag.*, Lond., Vol. 75, Oct., 1946, pp. 217–18. 1s.

THORIUM.—Cerium and thorium. E. F. Roots.—*W. Miner*, Vancouver, B.C., Vol. 19, Aug., 1946, pp. 50–6. 25 cents.

TUNGSTEN—FRENCH MOROCCO.—Tin and tungsten deposits of French Morocco. E. T. Heck.—*Econ. Geol.*, Lancaster, Pa., Vol. 41, June–July, Pt. 1, 1946, pp. 383–9. $1.

TUNGSTEN — VIRGINIA — MECKLENBURG CO.—Exploration of the Mecklenburg county, Va.: Tungsten area. Ben E. Argyle. —*U.S. Bur. Min. Rep. Invest.* No. 3857, Wash., D.C., April, 1946, 5 p.

URANIUM — NORTHWEST TERRITORIES—ELDORADO.—The Eldorado enterprise. By members of the staff.—*Trans. Canad. Inst. Min. Metall.*, Vol. 49, 1946, pp. 423–38 (*Canad. Min. Metall. Bull.*, Montreal, No. 413, Sept., 1946. $1).

URANIUM — PROSPECTING — ULTRAVIOLET LIGHT METHOD.—Locating uranium with the ultra-violet lamp. Jack DeMent.—*Min. Congr. J.*, Wash., D.C., Vol. 32, May, 1946, pp. 26–30. 20 cents.

MINOR METALS—continued.

VANADIUM —The outlook for vanadium; its occurrences and uses.—*S. Afr. Min. Engng. J.*, J'burg., Vol. 57, Pt. 1, Aug. 24, 1946, pp. 697–9. 6d.

VANADIUM — GERMANY. — Recovery of vanadium from iron and steel plant slags.—J.I.O.A. Final Report No. 28, June, 1946, 8 p.

ZIRCONIUM.—Zirconium. Its occurrence, preparation and uses.—*Min. J.*, Lond., Vol. 227, Sept. 7, 1946, pp. 710–1. 8d.

NON-METALLIC MINERALS.

ANHYDRITE—YORKSHIRE — BILLINGHAM.—Billingham mine. G. Eland Stewart. —*Instn. Min. Metall. Bull.*, Lond., No. 480, Sept., 1946, 11 p. 1s. 6d.

BUILDING STONES—PORTLAND STONE. —Portland stone. W. J. Sansom and H. W. Pangbourne.—*Mine & Quarry Engng.*, Lond., Vol. 12, Oct., 1946, pp. 97–104. 1s.

CARNALLITE — UTAH — GRAND CO.— Salts cored successfully with oil field equipment. C. Luther Severy.—*Engng. Min. J.*, N.Y., Vol. 147, Aug., 1946, pp. 86–7. 50 cents.

CLAYS—KAOLIN.—What is china clay? W. Trethewey.—*Quarry Mgrs.' J.*, Lond., Vol. 30, Sept., 1946, pp. 145–8. 1s.

CLAYS — PROPERTIES — BONDING ACTION.—The bonding action of clays: Pt. 2, Clays in dry molding sands. Ralph E. Grim and F. Leicester Cuthbert.—*Univ. Ill. Bull.*, Urbana, Vol. 43, Feb. 6, 1946, 43 p., biblio. 55 cents.

COBALT — METALLURGY. — Recovery of cobalt from manganese ores in the production of electrolytic manganese. (1. Cobalt content of manganese ores. David Schlain. 2. Separation and electrodeposition of manganese and cobalt from manganese electrolytes. J. H. Jacobs, F. K. Shelton, and R. G. Knickerbocker.)—*U.S. Bur. Min. Rep. Invest.* No. 3866, Wash., D.C., April, 1946, 18 p.

FLUORSPAR—NEW MEXICO—LINCOLN CO.—Exploration of Gallinas fluorspar deposits, Lincoln Co., N. Mex. J. H. Soule.—*U.S. Bur. Min. Rep. Invest.* No. 3854, Wash., D.C., April, 1946, 25 p.

FLUORSPAR—WYOMING—CROOK CO.— Exploration of the Bear Lodge fluorite property, Crook county, Wyo. W. C. Dunham.—*U.S. Bur. Min. Rep. Invest.* No. 3877, Wash., D.C., April, 1946, 7 p.

HELIUM.—The properties and uses of helium (including a comprehensive bibliography, 1938–45). Henry P. Wheeler, Jr.—*U.S. Bur. Min. Inform. Circ.* No. 7344, Wash., D.C., April, 1946, 66 p.

LIMESTONE — KILNS — ROTARY. — Rotary lime-kiln practice. W. R. Cliffe.—*Trans. Canad. Inst. Min. Metall.*, Vol. 49, 1946, pp. 413–22 (*Canad. Min. Metall. Bull.*, Montreal, No. 413, Sept. 1946. $1).

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STEAM ENGINES — MAINTENANCE. — Steam engine repairs. Edward Ingham.—*Colliery Guard.*, Lond., Vol. 173, Aug. 30, 1946, pp. 294–96. 10d.

SILVER.

ECONOMICS — CURRENCY — GT. BRITAIN.—Demonetization of silver in Great Britain.—*Min. J.*, Lond., Vol. 227, Oct. 5, 1946, p. 787. 8d.

SILVER — COLORADO—SUMMIT CO. — BIG FOUR MINE.—Exploration of the Big Four zinc-silver mine, Summit county, Colo. R. B. McCulloch and W. P. Huleatt.—*U.S. Bur. Min. Rep. Invest.* No. 3884, Wash., D.C., April, 1946, 7 p.

TIN.

CALIFORNIA.—Survey of tin in California. R. H. Bedford and F. T. Johnson.—*U.S. Bur. Min. Rep. Invest.* No. 3876, Wash., D.C., April, 1946, 14 p.

FRENCH MOROCCO.—Tin and tungsten deposits of French Morocco. E. T. Heck.—*Econ. Geol.*, Lancaster, Pa., Vol. 41, June-July, Pt. 1, 1946, pp. 383–9. $1.

TIN—*continued.*

GEOLOGY—ARGENTINE (NORTH). — Geologia de los yacimientos de 'Estano Madera'. Federico Ahlfeld.—*Industr. Min.*, Bs. Aires, Vol. 6, July, 1946, pp. 52–7. $1.50.

GEOLOGY—BOLIVIA.—Geologia de los yacimientos de 'Estano Madera'. Federico Ahlfeld.—*Industr. Min.*, Bs. Aires, Vol. 6, July, 1946, pp. 52–7. $1.50.

MINING METHODS — DREDGING — QUEENSLAND.—Tableland tin dredge.—*Chem. Engng. Min. Rev.*, Melb., Vol. 38, May 10, 1946, pp. 275–81. 1s.

MINING METHODS — SLUICING — TASMANIA — BRISEIS CONSOLIDATED MINE.—Sluicing operations at Briseis Consolidated N.L.: Pt. 1. H. H. Dunkin.—*Chem. Engng. Min. Rev.*, Melb., Vol. 38, July 10, 1946, pp. 351–62. 1s.

TASMANIA — ABERFOYLE.—Tin mining and milling at Aberfoyle: Pt. 2. H. H. Dunkin.—*Chem. Engng. Min. Rev.*, Melb., Vol. 38, May 10, 1946, pp. 283–94. 1s.

Subject to revision.] [*A Paper issued on November* 12*th*, 1946, *to be submitted for discussion at a Meeting to be held in the Apartments of the Geological Society of London, Burlington House, Piccadilly, London, W.* 1, *on Thursday, November* 21*st*, 1946, *at* 5 *o'clock p.m.*

The Panasqueira Mines, Portugal: Wolfram Mining and Milling: Labour Organization.*

By J. C. Allan, *Member*, G. A. Smith, *Member*, and R. I. Lewis, *Associate*.

Introduction

The Panasqueira mines in Portugal, the property of Beralt Tin and Wolfram, Ltd., cover an area of 15 square kilometres of concessions situated among the foothills of the southern flank of the Serra da Estrêla range of mountains, which run diagonally across the country in a south-westerly direction. The small town of Fundão, 300 kilometres from Lisbon, on the Beira Baixa railway, is the nearest railway station and is 40 kilometres by road from the property ; 16 kilometres of this road were built and are maintained by the company (Fig. 1).

History

Local legend has it that the Romans and, later, the Moors worked the property for cassiterite. Although there is none of the evidence usually found associated with ancient workings in Portugal, it is probable that such did in fact exist. The extensive alluvials that occur in the granite country surrounding the central mass of phyllites on which Panasqueira is situated were more easily worked for cassiterite and it is probable that no large-scale work was carried out at Panasqueira itself owing to the difficulty experienced in separating cassiterite from wolfram. Small mounds of highly-oxidized wolfram have been found in the surface workings of certain cassiterite areas and it is supposed that these represent the useless product picked out from cassiterite-bearing ore by the Moors. The first known recorded reference to mineralization at Panasqueira is found in a descriptive catalogue (dated 1889) of the mining section of an exhibition held in Lisbon.

*Paper received on 27th July, 1945.

FIG. 1.—Map of Portugal showing position of Panasqueira relative to phyllites and granite.

In 1894 an enterprising local inhabitant took to Lisbon samples of wolfram which aroused the interest of the professor of mineralogy there to such an extent that he took up prospecting licences and started production on a small scale. This must have been successful for, in 1898, the Government Gazette authorized the Wolfram Mining Company in Portugal to work wolfram mines in the boroughs of Fundão and Covilhã. This company, later financed by Banco Burnay, continued until 1910, but little is known of its operations or production.

In 1910 the mine was rented to an English company, which, in 1911, purchased the entire rights and operated under the name of the Wolfram Mining and Smelting Company until 1928, when it was incorporated as Beralt Tin and Wolfram, Ltd.

Up to 1910, operations had been on a small scale; little more than tributing with a large number of small adits from which veins were stoped under difficult conditions. In 1912, however, Adit No. 5, the main cross-cut in the section known as Old Panasqueira, was started (Figs. 2, Plate I, and 3). This was the first attempt to cross-cut under the main series of veins at this level. Production of concentrates increased and averaged 200 tons of wolfram per year from 1910 to 1919, rising to a peak of 314 tons in 1916 and falling to 96 tons in 1919, when production ceased owing to the very low price for the mineral.

From 1919 onwards the mine passed through a period of alternate stagnation and operation until 1927, when the company was reconstructed under its present name and intensive development of the tin-bearing areas was put in hand. At the same time, wolfram production was restarted and continued until 1931, after which for a time only tin areas were worked. From figures available, production would appear to have been of the order of 25 to 30 tons per month, consisting of wolfram and cassiterite in approximately equal proportions.

In 1934 the capital of the company was increased and the mine was reorganized and re-equipped under new technical management. Its history from that date has been one of constant expansion. Table I shows the rapid increase in production of wolfram concentrates from 1934 to 1941. The gradual stoppage of supplies from England as well as labour difficulties resulting from war conditions made themselves felt by 1940/41 and it was not until October, 1942, that it became possible once again to bring production back to the higher figures previously obtained.

The price paid for wolfram concentrates, which before the war had been of the order in Portuguese currency of Esc. 10$00 per

TABLE I

CONCENTRATES PRODUCTION

(Metric Tons)

Year to March 31st	*Wolfram Concentrates*	*Cassiterite Concentrates*	*Total Production*	*Average per Month*	*Index Figure*
1934/35	381	108	489	41	100
1935/36	485	189	674	56	138
1936/37	740	163	903	75	185
1937/38	1,103	126	1,229	102	251
1938/39	1,650	132	1,782	148	364
1939/40	1,890	108	1,998	166	408
1940/41	2,368	103	2,471	206	505

kg., rose steadily under the influence of competition to Esc. 650$00 per kg. Trained labour, tempted by these high prices, left the services of the company to work on their own account, a highly lucrative yet illicit occupation. A recruiting campaign had to be put into operation to obtain the necessary labour from parts of the country where mining was unknown. The majority of the men so recruited were of a poor type, drawn by the high wages that the company was obliged to offer. The result of these difficulties was reflected in production, which fell progressively, until, in January, 1942, it had reached the lowest figure since November, 1937.

However, by increase in wages, the building of accommodation for an extra 2,000 men, and the training of labour, it gradually became possible to increase production and, by the time materials became available from Great Britain, the Main Adit section with portal at Barroca Grande (Figs. 2, Plate I, and 3) had been prepared for a monthly production of 300 tons.

In September, 1943, production reached the total of 279 tons of wolfram concentrates. Early in that month orders were received to curtail production, as the high export duties converted a low-cost producer into an expensive source of wolfram. From September, 1943, until wolfram mining was suspended in June, 1944, by the Portuguese Government, production was curtailed. Production from all sources for the period 1941 to 1944 is shown in Table II.

TABLE II

CONCENTRATES PRODUCTION

(Metric Tons)

	Wolfram	*Cassi-terite*	*Total*	*Average per Month*	*Index Figure*
April, 1941/Sept., 1941	1,088	20	1,108	185	453
Oct., 1941/March, 1942	833	2	835	139	342
April, 1942/Sept., 1942	1,057	27	1,084	181	443
Oct., 1942/March, 1943	1,240	44	1,284	214	525
April, 1943/Sept., 1943	1,410	28	1,438	240	588
Oct., 1943/March, 1944	1,005	36	1,041	173	426
April, 1944/June, 1944	333	14	347	116	284

The object of the present paper is to review operations from April, 1934, when mine production amounted to 21 tons of concentrates, to March, 1944, and to describe the manner in which a steady and consistent increase in production and efficiency was achieved.

GEOLOGY

The Panasqueira area lies in very hilly country, ranging from an altitude of 1,083 metres, the highest point on the property, to 600 metres in minor valleys, and down to 350 metres on the River Zezere—a tributary of the River Tagus and the main drainage of the area. This topography, while creating transport and construction problems, offers the advantage of exposing long lengths of vein outcrops, which have facilitated study of the vein systems and have afforded ready means of opening up veins by adits (Fig. 4, Plate II).

Some idea of the value of the topographical situation may be gained from the statement that outcrop workings carried out either by the Company or tributers total some 45 kilometres. While records prior to 1934 are not complete a close estimate puts the total production of wolfram from the concessions at something over 20,000 tons, of which 15,600 have been produced since 1934. This represents some 2,000,000 square metres of total ground broken, all above adit level and, therefore, with no drainage problem.

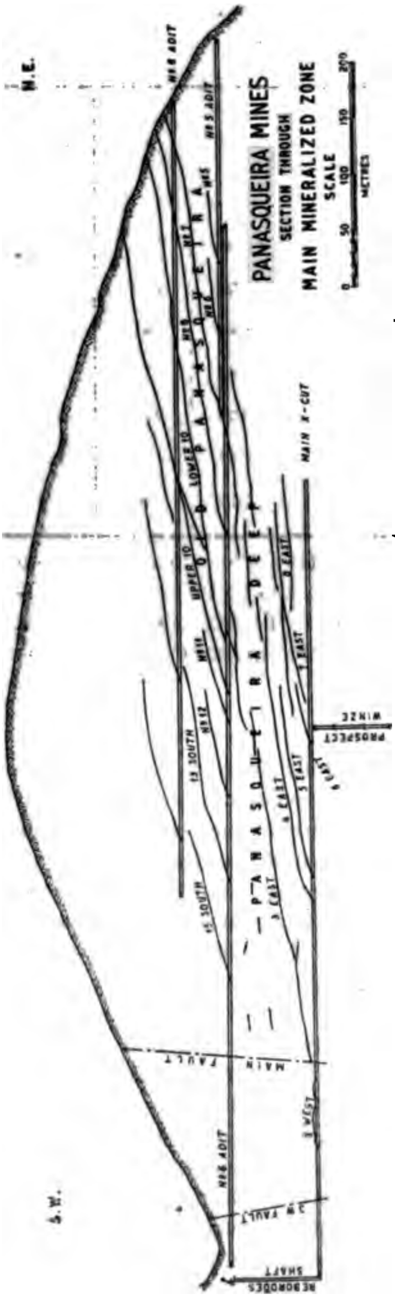

Fig. 2.

Geologically the area is underlain by metamorphosed sedimentary rocks, chiefly phyllites, which in turn are underlain by and almost completely surrounded by granite (Fig. 1). No granite occurs on the Panasqueira property itself, the nearest outcrop being some ten kilometres distant, although Professor W. R. Jones estimates the granite to be at not less than 600 metres below the main mineralized zone.

Mineralization accompanied the granite intrusion and followed joint planes formed as the result of the main crustal movement which had previously affected the Iberian Peninsula. The various joint planes formed can for convenience be divided into those with a low dip and those with a steep dip. The former dip both south-west and north-west and are the most important, as they were the channels generally followed by the infilling mineralizing agents. The steep-dipping joint planes are usually tight and are only occasionally filled with mineralized quartz which in thin stringers connects overlapping lenses on the low-dip planes. There are particular exceptions to this case—notably, two steeply-dipping veins which traverse the main mineralized area in a north-east/south-west direction and dip south-east. It is suspected that these two veins are more important than at first supposed, for indications are that the greater part of the wolfram mineralization lies between them. At the horizon of the Main Adit they are some 300 metres apart, with a tendency to converge both in a north-easterly direction and down the dip.

Orebodies, chiefly confined to the south-west dipping joint planes, are lenticular in form, up to 1·20 metres thick and thinning out at their edges. Their average thickness for a number of years has been 30 centimetres. For convenience of operation and nomenclature, a succession of these lenticular bodies, along the same general line and dip, is known as a vein, the general dip being 10° in a south-westerly direction (Fig. 3). These lenses are continuous on the strike and dip for lengths which vary from only a few metres up to 100 metres and occasionally more. At their extremities, in both strike and dip directions, they desert one joint plane for another parallel to it, a few feet above or below, and generally overlap one another, as shown in Fig. 5 (Plate III). A feature of these extremities is that they are often highly mineralized with wolfram.

The ' veins ' are separated vertically by distances varying from five to twenty metres. In the Old Panasqueira-Panasqueira Deep wolfram area (Figs. 2 and 3) eight main series of continuous lenses have been worked, but the incidence of subsidiary lenses is such that it is not uncommon for mining operations to break through from one to another.

Mineralization consists of wolfram, cassiterite, pyrite, arsenopyrite, some chalcopyrite, and, occasionally, blende, with traces of galena, together with siderite and gilbertite mica, occurring in a quartz gangue. Wolfram is disseminated irregularly, generally as large crystals or as a rib along the foot- or hanging-wall. Fig. 6 (Plate III) shows a typical case of wolfram enrichment. This irregular habit of the wolfram mineralization makes it impossible in practice to determine the value of any vein by normal sampling methods.

A number of faults traverses the area in a north-south direction, but although post-mineralization in age these do not, except for what is known as the Main Fault (Figs. 2 and 3), cause much disturbance of veins. Their correlation is naturally difficult, owing to the lenticular nature of the orebodies, but it is not the throw of the Main Fault which has caused concern but its apparent effect on mineralization. It appears to divide the main mineralized zone into two areas—that of more or less regularly-dipping lenses on the east side and that of irregularly-dipping lenses on the west. The former consistently follow the south-west low-dipping joint planes, while the latter wander indiscriminately from the south-west to the north-west joint planes and vice versa.

Examining this further, it is deduced that the south-west-dipping joint planes were easier of access to lower-temperature solutions, with a resultant preponderance of wolfram values, whereas the north-west-dipping joint planes were tighter and only penetrated by more volatile components, giving rise to the deposition of cassiterite. As Professor Jones reports: ‘The form and continuity of low-dipping joint planes, and hence the mineralized veins, are related to the type of rock in which they occur’. This would have an important bearing on mining operations if these various types could be mapped, but, as he points out, the rock types are so irregularly disposed, both laterally and vertically, that, as in parts of Cornwall, the study of rock types becomes extremely difficult.

The present paper is chiefly concerned with the production of wolfram, which has been the principal objective of the company during the last ten years. The greatest attention has, therefore, been paid to the area of maximum wolfram mineralization—that is, the area where south-west-dipping joint planes predominate. The future, however, is likely to demand an intensive study of cassiterite possibilities and, therefore, investigation into the theory of the causes of differential deposition of cassiterite and wolfram already outlined.

The Panasqueira area forms part of an extensive tin-tungsten mineralized zone which stretches from north Spain through the north of Portugal. The economic area for wolfram, as far as it affects Panasqueira, is, however, limited to a block of ground some 150 metres thick and dipping south-west, with a width of 600 metres on the strike and 800 metres down the dip from surface, as shown in Figs. 2 and 3. These are not necessarily the final limits, but mineralization and vein continuity outside this block are more irregular. It is difficult to establish ore reserves in any accepted meaning of the word. Mineralization is, however, widespread and individual veins have been proved over such wide areas that provided economic conditions permit the working of this type of deposit there are large areas of ore available. There is no evidence to prove that mineralization does not extend below the level of the present lowest workings: there is, in fact, strong evidence in favour of such a supposition, which has recently been strengthened by the cutting of payable lenses in a winze reaching 70 metres below the level of the Main Adit (Fig. 3).

The block of ground outlined is the main wolfram mineralized zone with which this paper deals. It is worth mention, however, that there are five other, so far smaller, zones which have been intermittently worked by the company and intensively by tributers during the war period. These areas contain cassiterite-wolfram mineralization in varying proportions. The relative position of these areas and the respective wolfram and cassiterite zones are shown in Fig. 2 (Plate I).

Development

The vein system, dipping to the south-west at an average of 10°, is exposed at Panasqueira as a series of superimposed outcrops across the face of a steep hillside. In the past, for the purposes of stoping, each vein was taken separately and an adit, usually numbered according to the first vein cut, was driven a few metres below its outcrop. This vein was then stoped to surface, often by the aid of a series of stope box-holes whereby it could be worked up the dip. Any difficulty encountered by the pinching out of the vein or by the weakening of mineral values was overcome by abandoning the lens and opening up a further adit to cut the same vein at some point along its strike. Apparently, little development was carried out ahead of requirements, although some large-scale development had been foreseen, as indicated by the work done in starting an adit 80 metres below No. 5 Adit, and which later, when extended, became the present Main Adit at Barroca Grande.

The direct result of this system of abandoning apparently-poor stopes was that selective mining could be practised and a high-grade ore sent to the mill. Stoping was limited to the more weathered rock near the surface, where breaking was simple and hand drilling possible. It had the disadvantage, however, of widely dispersing the working places and of running the risk of being without ore at short notice should several lenses pinch out at the same time.

As the outcrops of the veins were worked out the original adits were extended to cut the next higher veins in the series, which were then stoped, lens by lens, to the adits immediately above. Raises were put up at random rather than complying with some predetermined arrangement and the particular lens cut was stoped by 'mushrooming' round the top of each raise until extraction difficulties made further stoping impossible. The size of these stopes was limited down the dip by flooding, up the dip by the

This system has proved itself to be the most economical from the point of view of ease of operation and subsequent maintenance. A passage horizon is available for every 16½ metres of stope face. Furthermore, as the vertical distance between passage horizons is reduced to about three metres, the method is more elastic, enabling overlaps to be handled with relative ease. As these rarely have a vertical interval greater than two metres and as a geological study of the vein occurrence along the whole of the stope face exposed often indicates their probable habit, the vein can be kept at the required horizon in the passage face to ensure that, when overlaps

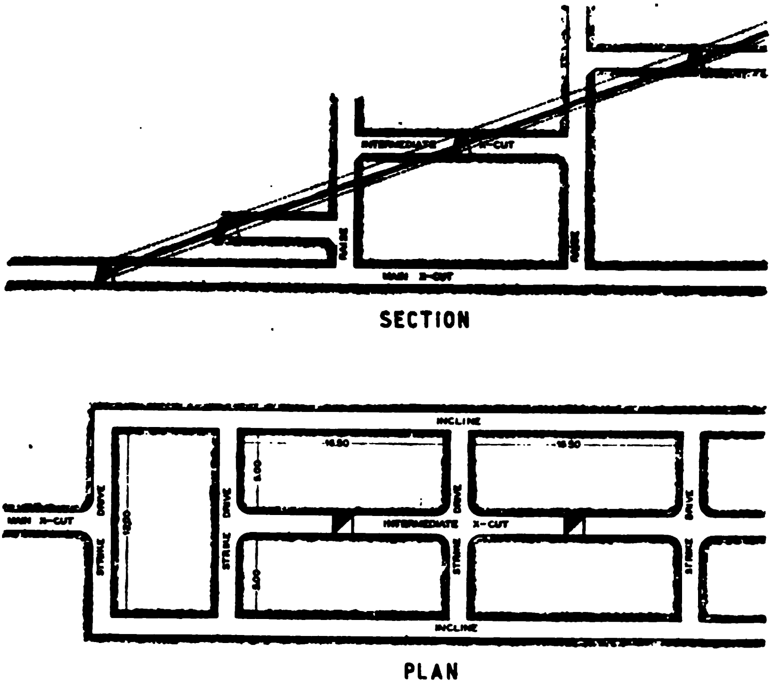

FIG. 7.—Development layout.

occur, the vein will still be in the face. The disadvantage of a longer dip interval between passages, and the resulting increase in carrying distance for the muckers, is overcome by temporarily branching the passages close to the face where necessary.

In a mine where mineral values and vein occurrence are so irregular, it is essential, if production is to be maintained at an even figure, to have available, by means of adequate stope development, a large length of reserve stope face. The total length of stope face required depends not only on irregularities in the vein occurrence, but also on the rate of stope-face advance. Efforts were therefore turned towards the development of a system of mining that would enable a rapid face advance to be obtained. As is described subsequently, this has not yet been achieved satisfactorily, except in isolated cases and then only by using methods unsuitable for general application.

Until 1940, the area broken per year in Old Panasqueira was greater than that broken in Panasqueira Deep and, as the yield of mineral per square metre of vein area was lower in the former section, a long length of stope face had perforce to be kept in operation, not only to maintain but to increase production. A large main development programme was therefore put in hand to open up Panasqueira Deep as rapidly as possible and from 1940 onwards this area, with its higher yield per square metre, began to predominate, until in the year 1943 to 1944 only 6 per cent of the total area broken was obtained from Old Panasqueira.

TABLE III

DEVELOPMENT (GENERAL DATA)

Financial Year	*Production of Final Concentrates from Mine. Metric Tons*	*Area Stoped. Square Metres*	*Main and Stope Development Advance. Metres*	*Length of Stope-Face in Operation Metres*	*Monthly Rate of Stope-Face Advance. Metres*	*Development Advance per Square Metre Broken. Cm.*	*Concentrates Recovered per Square Metre Broken. Kg.*
1934/35	458	—	1,377	—	—	—	—
1935/36	613	—	2,297	—	—	—	—
1936/37	759	71,911	2,881	1,752	3·4	4·0	10·6
1937/38	1,088	91,532	2,324	2,339	3·3	2·5	11·9
1938/39	1,585	142,234	2,702	3,199	3·7	1·9	11·1
1939/40	1,821	149,030	3,469	3,789	3·3	2·3	12·2
1940/41	2,331	182,421	9,182	4,229	3·6	5·0	12·8
1941/42	1,918	109,898	4,256	2,998	3·1	3·9	17·5
1942/43	2,045	109,360	4,513	2,545	3·6	4·1	18·7
1943/44	2,105	99,725	9,740	2,536	3·3	9·8	21·1

In the financial years 1941/42 and 1942/43 materials were scarce and those available required for productive operation ; development was therefore curtailed. However, with the improvement in this position in 1943/44, maximum development to permit selective mining, principally of Panasqueira Deep stopes, was put in hand, with the result that, as may be seen from Table III, production for that year was of the same order as that of 1940/41, although produced from only half the area. Although some 5,000 metres of stope face were available, it was found necessary to stope only half this length.

The effect of this development designed to maintain stope face in the richer areas, was to permit a high degree of selective mining, and it is felt that the results would have been even more remarkable had not production been forcibly curtailed in September, 1943.

The standard drive has a cross section of 2 m. by 2 m. Drilling is done with 55-lb. jackhammers mounted on jacklegs, 18 to 20 holes per round being sufficient in most cases. A double 'V' cut is generally used and holes are placed with the aid of a hole director. Table IV shows average efficiencies for the years 1939 to 1944.

TABLE IV

DEVELOPMENT (DETAILS)

Financial Year	*Metres Advanced per Machine Shift*	*Metres of Hole per Metre Advanced*	*Kg. of Explosives per Metre Advanced*	*Total Labour per Metre Advanced*
1939/40	0·54	26·22	8·85	6·02
1940/41	0·73	25·78	9·93	5·05
1941/42	0·73	25·98	12·54	5·50
1942/43	1·00	23·80	13·93	8·12
1943/44	1·24	25·61	14·14	8·98

MINING

The vein system at Panasqueira requires a disproportionate volume of waste to be mined compared with that of vein. Furthermore, the roof requires continuous support. So far no method of roof support more economical than that of packwalls built of waste has been evolved. The stoping method as developed to date is therefore costly in labour, as all broken ground has, of necessity, to be handled either as waste for the support of back areas, or as ore and excess waste to surface.

The proportion of ore to waste broken and the weight of valuable mineral recovered per unit of area are small, as is shown in Table V. Therefore, in order to obtain the requisite production of wolfram, large areas must be opened up and operated simultaneously.

TABLE V

VEIN WIDTH, STOPING WIDTH, AND YIELD

Financial Year	*Vein Width. Metres*	*Stope Width. Metres*	*Ore/Waste Ratio of Mill Feed*	*Kg. of Concentrates/Square Metre Broken*
1939/40	0·35	1·45	1/1·00	12·2
1940/41	0·31	1·58	1/1·34	12·8
1941/42	0·29	1·67	1/1·97	17·5
1942/43	0·33	1·63	1/1·82	18·7
1943/44	0·34	1·63	1/1·33	21·1

Stope face advance on individual lenses has been as high as eight metres per month, but the pinching out of lenses and the subsequent stope development required to get the next lens into operation causes interruptions to the stoping sequence of any particular vein. The average face advance is thereby reduced to a figure of some 3·5 metres per month. These factors necessitate a large reserve of stope face and it has been found that normal operation calls for a reserve of some 50 per cent in order to maintain the required length of face to meet production demands in continuous operation. Some idea of the extensive area of workings in operation can be gauged by the fact that in 1944 there were 37,700 metres of tramming ways and 37,500 metres of compressed-air and water pipes installed.

Stoping System.—Any stoping system employed must necessarily take into consideration the following:

(1) The necessity of roof support owing to joint planes in the phyllites. The extent to which the ground can be left unsupported

varies considerably, so that in some sections it is unsafe to leave more than 1·50 metres between the working face and any support.

(2) Owing to the flat dip of the veins all material broken at the face has to be manhandled. The restricted space between the working face and roof support, and the changes in horizon from lens to lens, militate against the use of scrapers or other mechanical devices, although it is still a matter of doubt whether these methods cannot in fact be partly employed with benefit.

(3) The provision of a sufficient length of stope face to permit of an uninterrupted cycle of operations. In view of the comparatively restricted working space it is obviously desirable for the drillman to have a clear run of the face to drill over; similarly, the mucking crew can work more efficiently if given a definite section of face to clear.

(4) Any method employed must provide the maximum possible recovery of all fine material. This is demanded by the friability of the wolfram. There are, however, wide variations in the friability of the phyllite and no combination of hole spacing, powder strength nor burden has yet been devised that gives a high breaking efficiency per drill shift and, at the same time, does not result in at least 40 per cent of the phyllite itself being broken smaller than 1½ in.

(5) The freedom with which veins break from the country rock. This has frequently suffered silicification and pyritization—more particularly in the deeper areas, where veins are frozen to the walls. Where veins are steady, with definite clean joint planes, and where surface water has penetrated, the country rock can often be broken free from the vein, which can be removed with relative ease with hammers and wedges.

(6) Being close to the surface, the workings are more or less damp, particularly in the rainy season, when many places are definitely wet. The effect of this is that it has been impossible to educate the miners to drill from a sitting position, characteristic of narrow stoping on the Rand, and therefore wider stopes have to be carried.

(7) The mine is removed from any important centre of population, so that the bulk of the employees has to be housed on the property. Housing accommodation was strictly limited and the projected expansion programme was to increase the concentrate production ten times to 200 tons per month. Any stoping system, therefore, must have as its object the increase of labour productivity, particularly as the company had undertaken a construction and develop-

ment programme that, in itself, necessitated a considerable labour force.

(8) In view of the low standard of education among the miners and shift bosses it would be advantageous if a single stoping system could be generalized, although the system chosen might not be the optimum economic method for a particular set of circumstances encountered in a specific stope or section of stope.

During the past 10 years efforts have been made to discover a method of stoping which would give optimum economy under all conditions, and a measure of the progress achieved can be seen from the maintenance of approximately static stoping costs in spite of steady increases in the cost of labour and stores. Table VI shows mining costs from 1936/37 until 1940/41—that is, up to the time when costs were affected by war conditions. The fact that the cost per ton of final concentrates has not accompanied the rising cost per square metre results from the mining of richer ground, from the improvement in milling technique, and from the reduction of mineral losses underground by the introduction of improved methods of stoping.

TABLE VI

MINING COSTS (RATIOS)

Financial Year	*Cost/Square Metre, Index*	*Cost of Concentrates, Index*
1936/37	100·0	100·0
1937/38	113·0	105·1
1938/39	112·6	108·5
1939/40	120·5	106·8
1940/41	126·1	106·8

Before 1936 mining costs were based on the tonnage of ore milled and were therefore dependent on the amount of waste included in the ore sent to the mill. In order to obtain an apparent improvement all the mining department had to do was to take men from hand-picking in the stopes and put them on breaking more ground. The increase in dilution would thus be reported as feed to the mill. This led to an increase in the tonnage milled and a reduction in grade until the figure of 0·65 per cent for recovered grade was reached. Table VII, giving figures for the four months December, 1935, to March, 1936, shows this effect.

TABLE VII

DROP IN RECOVERED GRADE FOLLOWING OVERBREAK

Month	*Tonnage Milled*	*Cost/Ton, Index*	*Mill Feed, Recovered Grade, Per cent*
December, 1935	4,741	100	1·145
January, 1936	4,853	103	1·112
February, 1936	5,721	94	0·931
March, 1936	6,743	78	0·870

It was accordingly decided to reduce dilution to a minimum by resuing with the utmost care wherever possible, and by making underground hand-sorting of waste a maximum wherever conditions did not permit resuing. Although this resulted in an increase in recovered grade which, by December, 1936, had risen to 1·34 per cent, this increase was largely counteracted by a fall in tonnage milled and by increased mineral losses in the stopes owing to excessive hand-sorting, with the net result that no appreciable gain was obtained in the tonnage of concentrates recovered.

The urge for increased production and the limitation imposed by the shortage of available labour were clearly irreconcilable if such a policy of strict hand-sorting was to be followed, and the struggle between the two was continued until it became clear that a high recovery of mineral per square metre of veinstuff broken was incompatible with a high grade of mill feed.

Under the conditions obtaining at Panasqueira the conventional unit of tons milled ceases to be sufficiently allied to production to be of any value. Were it possible to determine with reasonable accuracy the tonnage of actual veinstuff mined, it might be argued that there was no need to depart from this conventional unit. On the other hand at Panasqueira, while a record is kept of average vein widths in order to obtain a rough guide as to dilution, the variation in thickness of veins makes the accuracy of the figure obtained doubtful. It was considered necessary, therefore, to adopt a unit that could be determined with reasonable accuracy; one which, at the same time, left undisturbed the full mineral content of the veins themselves. The unit finally adopted was that of square metres of vein mined. Considering the matter from this angle a better perspective is gained of the company's operations as a whole. It thus became clear that grade of mill heads could be purchased only at the expense of removing waste by methods far less efficient than the mill itself.

The other element affected by the increase in dilution is transport. This was overcome by increasing the capacity of the ropeway from mine to mill. From a cost point of view this was relatively unimportant, as the cost of ore transport over the ropeway represented only 2 per cent of the total operating cost.

It is clear from the described nature of the mineralization that the sampling of any vein *in situ* would require a prohibitive number of samples and, in view of the ' bonanza ' values encountered, such sampling could not be taken as a real indication of average values. The conventional alternative to sampling *in situ* would presumably be bulk sampling, by segregating the ore from one vein and sampling it separately, either by means of a special sampling plant, or by running the mill for a specified time on the ore to be tested. To do this without dislocating the whole of the company's operations presents a number of practical difficulties which it has not been found possible to overcome.

It became necessary, however, to devise a method that would provide some gauge of ' payability '. Since the introduction of the square metre of vein mined as the unit for cost calculations, the value of the ore mined has been gauged by the recovery in kg. of final concentrates for each such unit. The width of individual veins and their visually-observed mineral content govern this value, and veins of an unsatisfactory history over a prolonged period in these two respects are abandoned. This visual method of gauging mineralization has been perfected to such a degree that it seldom differs from results obtained in practice by more than 5 per cent, when applied to the mine as a whole.

Before 1934 resuing was the method used throughout the mine. Operations were restricted mainly to the veins in Old Panasqueira, where conditions obtaining at that time lent themselves in particular to that system of stoping. In this sector the veins tend to break free from the wall rock. Being near the surface, there is a certain amount of incipient weathering and the phyllites are on the whole softer and more friable.

Drilling was done dry with 35-lb. jackhammers of English manufacture and with chisel-pointed steel. Spiral-ribbed steel was provided in the softer stopes for the removal of ' cuttings '. Holes were placed at the discretion of the drill runner and were directed to ' knocking off the bumps '. Mucking and drilling took place on the same shift and the stope face was blasted as many as four times during the two shifts worked per day As far as possible the hanging-wall was removed from the vein, the larger pieces of waste being used to build a packwall and such material as could not be stowed behind the wall was trammed separately to surface. The vein-stuff was then broken separately with hammers and wedges.

Too many elements are lacking for it to be possible to make a direct comparison between this method and current practice. It can be said, however, that the normal recovery of concentrates per ton of mill feed was 1·29 per cent and the average monthly recovery represented about 80 kg. per man employed, 80 per cent of the labour force being employed in the mine. The fact that many of the old pack-walls have been reworked for the recovery of 'fines' and that *minus* 2-in. material from the old waste dumps, when treated in Panasqueira mill, yielded as much as 0·8 per cent wolfram, indicates the unsuitability of the system and points to the danger of excessive sorting underground. It would seem that more attention was paid to maintaining a high grade of mill feed than to maximum recovery of mineral from the mine.

By the 1940/41 period, although the recovered grade had fallen to 0·68 per cent, the average monthly recovery had risen to 75 kg. per man, 67·2 per cent of the labour force working in the mine.

The war-time call for increased production and the opening of Panasqueira Deep, where the veins, in general, are frozen to the surrounding country and where the rock is harder, led to the improvements in breaking technique subsequently described under 'Drilling and Blasting'. The necessity of increasing labour productivity and reducing mineral losses led to an alteration in the stoping system.

Particular care was exercised in removing all possible high-grade ore from the veins, before blasting, by the aid of chisels and hammers for bagging and direct transport to the magnetic separator at the mill. In this way the bad effect of blasting high-grade patches of mineral was reduced and gravity plant losses in respect of the treatment of this mineral were eliminated. Table VIII shows the amount of selected ore recovered since the financial year 1938/39, figures for previous years not being available.

The stoping system then tried was substantially that of semi-shrinkage, where only sufficient material was sorted and removed to give working room at the face, the remainder, whether ore or waste, being thrown back to form a scatterpile. Under this system no effort was made to break the wall rock separately from the vein, which, for ease of breaking, was usually carried in the centre of the face and broken with a hole above and one below.

TABLE VIII

SELECTED ORE

Financial Year	*Selected Ore (Tons)*	*Production of Final Concentrates*		*Grade of Selected Ore (Per cent WO_3)*	*Per cent of Total Production*
		From Selected Ore (Tons)	*Total from Mine Ore (Tons)*		
1938/39	123	83	1,585	45·7	5·2
1939/40	408	218	1,821	36·2	12·0
1940/41	778	349	2,331	30·5	15·0
1941/42	642	230	1,918	24·4	12·0
1942/43	1,757	564	2,044	21·8	27·6
943/44	2,284	713	2,105	21·2	33·9

Re-mining of the scatterpile followed some five metres behind the face, with careful hand-picking of waste to form permanent pack-walls. A special effort was made to collect all fines for transport to the mill. The following disadvantages of this system were found : As recovery of ore from the ' shrink ' took longer to perform than stope face advance, the two operations fell out of step, with the consequence that, as the capacity of the mill was increasing more rapidly than that of the mine, there was a tendency to remove a successively greater proportion of ore direct from the face. The increasing time lag between breaking and building permanent pack-walls gave the roof time to settle down on the ' shrink ', the mining of which became an operation requiring expert miners and more timber. Falls of ground with resulting loss of ore became frequent. Waste to make up for ore removed from the ' shrink ' was supplied from the face, causing interference with stoping operations and traffic difficulties in the stope passages. Excessive supervision was required.

This method was therefore considered unsuitable. There was, however, every advantage in maintaining the improved breaking that resulted from a more liberal use of powder, a heavier burden, and the carrying of the vein in the centre of the face, made possible by the fact that the permanent pack-walls were not subjected to the blast.

With this advantage in mind tests were run to see whether a protection to the permanent pack-walls could be provided without the use of a shrinkage pile. Several materials—such as heather mats, planking, and props—were tried and, finally, the building of a temporary scatter-wall at the face in front of the permanent pack-wall was attempted and gave rise to the stoping system in general use in the mine to-day. This scatter-wall is made of selected stones of large size and the shape to which the phyllite breaks lends itself well to its use for this purpose. The scatter-wall is built to a thickness of about 80 cm. and is broken down during mucking operations before advancing the permanent pack-walls. It receives the force of the blast, and ' fines ' are rarely found to have penetrated into the permanent walls more than a few centimetres. These are recovered by cleaning the face of the pack-wall with brushes made of heath.

Although it is found that this system of stoping is suited to most of the stoping conditions obtaining in the mine, it has certain disadvantages which tests have been made to overcome. At the same time as this scatter-wall system was being tested, the Study Department was organized and it became possible to conduct a series of time studies of the work involved in the various methods, the Associated Industrial Consultant's system of observed work units being used throughout as a basis for comparison in each case, the results being valuable for future guidance in the working of this particular mineral occurrence.

The distribution of labour efficiency per square metre and of mining costs for the year 1940/41 are shown in Table IX.

TABLE IX

MINING COSTS (DISTRIBUTION)

	Labour Shifts per Square Metre	*Per cent Distribution of Costs*
Drilling and Blasting	0·52	55·1
Stope Labour :		
Stope Trammers and Muckers	0·83	
General Stope Labour	0·28	
Stope Development and Maintenance	0·26	
	1·37	18·9
Supervision	0·03	2·9
Main Road Tramming	0·29	5·5
Mine General (including timber and support, shops and surveying)	0·28	17·6
	2·49	100·0

Drilling and Blasting

In 1934, except for a few instances of hand drilling in the outlying districts of the mine, all drilling was being performed by 35-lb. dry-drilling jackhammers. These machines were well suited to the physique of the labour available, an important feature in hand-held jackhammer work in flat stopes. From the point of view of mobility they are ideal and in soft rock stopes their performance is good, drilling 30 metres of hole per shift as a maximum.

When harder rock was found in the lower levels considerable test work was made with the object of finding a suitable jackhammer to drill wet and to give better performance in the harder ground. Prejudices, however, as a result of the added weight of the water equipment, were difficult to overcome and no hand-held jackhammer was found that gave improved drilling performance.

Drill Tests.—The decision was therefore taken to employ 112-lb. reverse-feed stopers with 1¼-in. pneumatic columns. These provide reasonable mobility together with a high drilling speed and have the advantage of being a heavy drill, the weight of which is not carried by the drillman. The maximum duty obtained per shift from these drills was of the order of 75 metres. The principal disadvantage of these machines for drilling at the stope face is the length between the point at which the drill is supported and the hole, which causes heavy wear in chuck bushings.

Further tests were run with other types of machine drill until, in 1943, drills for stoping purposes were standardized. A 55-lb. jackhammer mounted on a jack-leg was chosen as being the best for conditions obtaining in the stopes. This drill has a high drilling speed, a low maintenance cost, together with a high degree of mobility. The jack-leg overcomes the difficulty of weight and the drills soon became popular with the men, as high bonus rates could be earned. The maximum duty obtained with one of these drills was 127 metres in a shift.

During the year ended March 31st, 1941, 40,439 drill shifts were worked, of which 55·1 per cent, or 22,269·5 drill shifts, were on production, breaking 182,421 square metres of vein ; 66 drills were normally in service at this time. Drilling and blasting costs constituted 55·1 per cent of the total mining costs, as is shown in Table X. In view of the preponderance of this item in total mining costs, special efforts were directed towards improving breaking efficiency.

TABLE X
DRILLING AND BLASTING COSTS (DISTRIBUTION)

	Per cent
Drilling and Blasting Labour	9·4
Cost of Drill Maintenance, Steel Sharpening, and Steel Distribution	11·1
Cost of Compressed Air	9·4
Explosives	25·1
Drill-Water Supply	0·1
	55·1

In addition to the use of a faster machine, the increase in the metreage drilled per machine shift, as shown in Table XI, was achieved by improving air and water service, reducing lost time, by the introduction of an incentive to increase drillage, by the supply of longer stope faces for each drill, by improving steel sharpening, tempering and supply, and the maintenance of drills. This last was achieved by withdrawing every drill from the mine at weekly periods, whatever its condition, for cleaning and overhaul. A mechanical history card is kept for each machine for recording the metres drilled, the date of entry and despatch from the drill shop, and spare parts used. Lost time was reduced to an average of 40 minutes per shift after time studies had been made to ascertain the reasons for same.

One man is employed in the case of the 35-lb. jackhammer and two men in the case of the other drills.

At the same time tests were run to obtain an improvement in breaking efficiency. The original method of stoping in Old Panasqueira, where drilling and mucking took place on the same shift, did not allow of the placing of holes with any degree of precision. Short holes were drilled in a more or less haphazard manner as the face became clear of muck and the main pre-occupation was to blast the rock without disturbing the vein. By separating the drilling shift from the mucking shift lost time was very considerably reduced and the drillmen were presented with a clear face at which they could work to the best advantage.

Hole Directors.—The use of the hole director, as developed on the Rand, was introduced after extensive tests and by 'benching' the stope face improved breaking efficiency resulted after a short period of time, although there was a tendency for the consumption of explosives to increase. The results are shown in Table XI.

The length of stope face available for each drill was increased so that the best advantage could be taken of the 'benches' exposed by the previous blast. Latterly, each drill was given 100 metres of face over which to drill. The average distance between pairs of holes as measured from collar to collar is about four metres.

TABLE XI

BREAKING EFFICIENCIES

Period	*Metres Drilled per Machine Shift*	*Square Metres Broken per Machine Shift*	*Metres Drilled per Square Metre Broken*	*Kg. of Explosives per Square Metre*	*Metres of Stope Face per Machine Drill*
1935/36/37	17·2	2·05	8·37	1·21	—
1938/39	22·3	3·19	6·99	1·31	—
1939/40	26·5	5·16	5·14	1·39	40
1940/41	37·4	8·19	4·57	1·43	58
1941/42	52·3	13·19	3·96	1·57	110
1942/43	47·8	9·76	4·90	1·88	70
1943/44	58·9	12·56	4·69	1·78	98

N.B.—The poorer results in 1942 and 1943 were caused not only by the employment of untrained labour but also by the poor quality of explosives available.

Owing to the great variations in rock no standard size of hole director can be employed throughout the mine. The burden of the hole directors used varies between 45 and 70 cm., the average length of hole drilled being about 1·50 metres. Only by constant insistence can the permanent use of the hole director be effected as it is unpopular among the men.

Tests run with a view to reducing the consumption of explosives had to be discontinued when the manufacturers were unable to guarantee the supply of explosives of the same composition for any period. Improved results were obtained under test by the use of sand-filled tamping cartridges and by the spacing of explosive cartridges with wooden plugs. The latter test had to be discontinued as the length of the plug depends on the quality of explosive used, which, as stated above, could not be depended upon.

DRILL SHARPENING

The drill-sharpening shop is equipped with three I.R. 34 and one Sullivan Class 'C' sharpeners, the latter being used almost exclusively for shanking and the former three for sharpening $\frac{7}{8}$-in. hexagonal hollow steel with six pointed bits. An average of 862 steels, of which some 40 are hand-sharpened chisel-pointed bits for the light jackhammers, were sharpened per day during the financial year 1943/44, each machine being operated by a single operator. Prior to the men being put on incentive pay an average of 164 steels were sharpened per machine shift. Once on incentive pay this figure was increased to 404.

Heating is done in two rock-drill furnaces, each of which consumes about 140 kg. of charcoal per eight-hour shift. Two Ingersoll-Rand oil-fired furnaces are also available and these were being used for heating all the steel, both for sharpening and tempering, before the charcoal furnaces mentioned were erected as a result of a shortage of gas oil. They are now used solely for tempering and consume about 65 kg. of gas oil per 1,000 steels tempered.

Table XII shows figures obtained for steel consumption and metreage drilled per sharp steel over the three years 1941 to 1944. Improved results were obtained after structural alterations had been made to the shop with improved lighting and general working conditions.

TABLE XII

Financial Year	*Metres Drilled per Sharp Steel*	*Kg. of Drill Steel per Metre Drilled*
1941/42	0·92	0·14
1942/43	1·02	0·16
1943/44	1·43	0·15

The shop is also fitted with two emery grinders, a steel cutter, and forges. The wagon repair shop, drill repair shop, and the welder's shop are all housed in the same building.

COMPRESSED AIR

The compressed-air plant is the biggest individual consumer of power on the property. Reference to the general plan shows the dispersion of the working places. Compressed air was originally supplied to the various centres of mining activity by a number of small compressors of varying size actuated by prime movers of equally varied type. As the programme of expansion demanded a considerable increase in available compressed air, and as the bulk of the existing equipment had passed its useful life, it became necessary to study the whole question of power. To obtain energy from the nearest public supply company would, at that time, have meant the finance and maintenance of a power line of considerable length, added to which the companies concerned had limited water-storage facilities and were therefore liable to seasonal shortages.

The question of a central generating station was also considered both at the mine and, to save transport, at rail head. The final decision was to equip the River Mill with the power plant, described later, the gas engines for which were already on the property, although not erected, and to instal two diesel-engine driven compressors at Barroca Grande. The decision to do this rested largely on the fact that, given pipe of adequate diameter, the pressure loss over long distances is negligible. Further, direct conversion from oil fuel to compressed air cuts out electrical losses in transmission lines, transformers, and motors.

The units finally installed consisted of two sets by Belliss and Morcom, as follows :

A five-cylinder heavy oil engine, developing 482 b.h.p. at site, running at 375 r.p.m., connected by means of a Tex-rope drive to a two-stage air compressor running at 250 r.p.m. and rated at 2,400 cu. ft. of free air per minute to 100-lb. pressure.

A 52 h.p., two-cylinder heavy-oil engine, by Crossley, drives a 29·5 kW, 220 V generating set which provides power for operating cooling water pumps and the engine starting compressor.

The following are the principal operating statistics and refer to the financial year 1940/41.

Average hours operated per month	452
Lb. of fuel oil per 1,000 cu. ft. of air	1·44
Lb. lubricating oil per hour	1·81
Gallons cooling water per minute	40 per engine (130 for whole plant)
Time to raise pressure in system from zero to 100 lb.	8 minutes
Time for pressure to fall to zero after shutting off engines	2 hours 15 minutes

By June, 1942, a new power supply company had built a dam 15 km. from Panasqueira for the generation of hydro-electric power and a 40,000-volt power line had been run through the concessions to join with a second power station some 25 km. distant. At the same time, gas oil was in short supply and the diesel engines required a very thorough overhaul, calling for the renewal of parts unobtainable in the local market. It was therefore decided to electrify both Panasqueira and Barroca Grande sections.

The Belliss compressors can be driven either by their respective diesel units or by electric power. Diesel power was used as it was doubtful whether electric power could be supplied after the war at a sufficiently cheap rate to compete with the diesel engines, or, if it could, to cover seasonal shortages of water.

The cost of compressed air represented 17 per cent of the total drilling and blasting cost in 1940/41, distributed as shown in Table XIII.

TABLE XIII

DISTRIBUTION OF COMPRESSED AIR COSTS

	Diesel-Driven Compressors. Per cent
Wages and supervision	4·2
Fuel	83·1
Lubricating oil	7·6
Stores	1·6
Shops	1·4
Sundries	2·1
	100·0

MULE TRANSPORT

Main Road Tramming.—The Main Adit—3 m. by 2½ m. in cross section—cuts the Panasqueira series of veins 1,275 metres from the portal, and, therefore, beyond the economic limit of hand tramming. The total length of the adit is 1,680 metres and the average gradient 0·62 per cent in favour of the load. The merits of mechanical haulage by diesel or battery loco. were investigated, but it was decided to employ mules, with the following results:

Average number of trains per mule shift	8
Average number of cars per train	8
Average distance travelled per mule shift	23 km.
Average tons/km. useful load per mule shift	75
Tons handled, ore and waste for the year 1940/41	800, 200
Number of mules in Main Adit services	32

The stable crew consists of one foreman and five stable hands; the mule ration consists of seven kg. of feed per day, comprised of beans, maize, etc., and 10 kg. of hay or straw.

The figures given refer to the financial year 1940/41—i.e., under normal working conditions. Latterly, increased activity underground required more mules and in 1943/44 some 40 were employed in the Main Adit alone.

The track is double and of 45-cm. gauge, laid with 12-kg. rails. Cars used are 20-cu. ft. side-dump Hudson cars fitted with roller bearings. These are filled at the raise chutes and run to assembly points, where they are coupled up in trains of eight or nine cars. Each train is coupled to a brake bogey consisting of a wagon chassis, weighted with old rock-breaker jaw plates, on which the driver is seated. Assembly points are so arranged that the driver and mule, on bringing in a train of empties, can turn over to a full train with a minimum of delay. On reaching the portal the same situation obtains; the dumping of cars being the duty of the surface crew, which also makes up the train of empties.

Mules are specially selected for weight and docility and a good mule weighs 350 kg. No difficulty has been experienced in getting the mules to go underground, although many have to be replaced after some two years' service, as they are liable to become unmanageable. Accident rates are high and are mostly caused by the brake bogey running into the back legs of the mules.

The maximum tonnage transported per shift under test was 1,200 tons, with 18 mules underground and a further two on surface for dumping waste.

Internal Transport.—In 1942 transport difficulties resulting from a shortage of petrol and tyres led to the more extensive use of mules as an alternative to lorries and motor cars for internal transport and by September, 1942, a total of 106 mules were in service in the three sections. A history card is kept for each mule, stating the work done each month, days lost due to sickness or accidents, transfers, sales, and any other data of interest. The following data refers to the 22 months from September, 1942, to June, 1944.

Total number of mules in service	106
Total mule shifts worked	46,417
Mule shifts lost as a result of sickness or accidents	3,004
Per cent lost time	6·5
Deaths :	
Accidents	14
Other causes	6
	20

Cost distribution of mule maintenance was as follows (1940/41) :

	Per cent
Labour	14·9
Stores :	
Feed	76·6
Various	8·5
	100·0

Surveying

The survey department is responsible for the measurement of all stope and development advances, control of tramming grades, surface surveys and the plotting of tribute workings. Development headings are measured weekly and surveyed every month, while surveys of all stopes are made at monthly intervals for the purpose of preparing working plans and for the calculation of efficiency statistics. All working plans are drawn to a scale of

1/500, which is the smallest to give reasonable accuracy for the computation of areas, the unit value for all costs and statistics, and to show vein movements and overlaps in sufficient detail. This scale is not, however, sufficiently accurate for the computation of labour incentive bonuses based on stope advance and, when this system was in force, surveys were made at fortnightly intervals and drawn to a scale of 1/250. This was reduced to the normal scale of 1/500 for working plans.

The survey of some 5,000 metres of stope face, involving areas up to 20,000 square metres, takes a week to complete and areas are compensated by a correction factor according to the metreage drilled between the survey date and the end of the month. No correction is made to surveys for the calculation of incentive bonus, as this is paid in respect of the actual period between survey dates.

All adits, cross-cuts, inclines, and stope passages are surveyed by theodolite, while hanging compasses are used for stope-face surveys, which are tied on to theodolite points. These are brought up to date about every six months in order to keep a check on the total area mined.

The staff of the survey department generally consists of two surveyors, two assistant surveyors, four compass boys, and assistants, and while kept as small as possible, is larger than would be the case could stope surveys be evenly spread throughout the month. The basing of costs and efficiency statistics on surveyed areas obviously necessitates that the final month's surveys be compressed into as little time as possible.

Latterly, a geological department has been formed which, in close co-operation with the survey department, maps all geological data of interest.

Ropeway

The principal mining areas are connected with the River Mill by means of mono-cable ropeways. The main ropeway runs from Barroca Grande to the River Mill *via* the angle station at Entroncamento, a distance of 4,150 metres. Subsidiary ropeways carry ore from Alvoroso to Entroncamento (300 metres), from Vale das Freiras to Entroncamento (2,400 metres), and from Panasqueira to Barroca Grande (2,060 metres).

The Barroca Grande-River Mill ropeway was originally designed to carry ore at the rate of 20 tons per hour between Entroncamento and the River and at a rate of 12 tons per hour between Barroca Grande and Entroncamento. By 1936 the entire ropeway capacity had been increased to 25 tons per hour by reducing the bucket interval and by increasing the size of the buckets. The size of the rope was increased from 2¾ in. to 2⅞ in. circumference, extra trestles were built, and a few structural alterations effected to others. In 1938 the capacity of the ropeway was increased to 30 tons per hour by again decreasing the bucket interval and by increasing the speed of the rope and the bucket capacity. The latter was made possible by the removal of the bucket linings.

Details of the main ropeway and operation figures for the period 1939 to 1944 are given in Table XIV. A greater tonnage was

TABLE XIV

ROPEWAY

Type of rope	6/7 Lang's lay 2⅞-in. circ.
Breaking stress	26·9 tons.
Difference in altitude, Barroca Grande-River	214 m. (in favour of load).
Length of ropeway	4,150 ,,
Total length of rope	8,652 ,,
Capacity tons per hour	33
Bucket capacity	330 kg.
Bucket interval	82½ m.
Rope speed	139 ,, per minute.
Number of buckets on the line	100
Driving power	7 h.p.
Ropeway bin capacity	870 tons.
Average running time	91·2 per cent.
Average tonnage transported per hour	34·52 ,,
Average operatives employed per shift	12
Average life of rope	300,000 tons.

transported per hour than the rated capacity of the ropeway. This was obtained by overloading the buckets, which at times were heaped to carry 360 kg. of ore. Cost distribution for the years 1939/42 was as follows :

	Per cent
Supervision	3·2
Operation (wages)	40·5
Stores	7·7
Rope (amortization)	23·2
Power	18·6
Shops	5·4
Transport and sundries	1·4
	100·0

HAND-PICKING PLANT

Having attained the maximum capacity of the ropeway by overloading it to the extent of 70 per cent over its original designed capacity, and as the mill capacity could not well be increased without impairing milling efficiency, the only way by which production could be further increased was by increasing the value of the mill heads sent over the ropeway. This could be achieved either by increasing the extent to which sorting was performed underground or by the erection of a surface hand-picking plant ahead of the ropeway.

Past experience had shown that the grade of mill feed could be increased by careful underground sorting and, in fact, a figure of 1·4 per cent metal for mill heads had been obtained during a short period. It became clear, however, that such an increase could only be obtained at a cost of losing mineral values in the mine and in the waste dumps. Although the question of a surface hand-picking plant had been studied many years before, the difficulty at that time was to obtain a suitable site for its construction, as ore was entering the ropeway not only at Barroca Grande, but also at Entroncamento. It was not until an urgent call for increased production was made in 1942 that it was finally decided to design and erect a hand-picking plant of simple construction that could be put into operation as quickly as possible. Owing to war difficulties, materials for the plant became available only after long delays and it was not until August, 1943, that the plant was first put into operation.

The plant, which is housed in a wooden building, consists of two 30-in. picking belts running in parallel at a speed of 10 metres per minute, with a total capacity of 100 tons per hour. Each belt has a total length of 65 metres, 32 metres being available for picking. The two belts, each driven through a 6-in. worm reducer by a 6-h.p. electric motor, are fed from two 40 cu. m. capacity bins by means of Locker grizzly vibrating feeders, magnetically controlled by a.c.-d.c. generating sets. Sprays of water are allowed to play on the belts as an aid in distinguishing ore from waste. The picked waste is thrown forward into centrally-placed bins, having a total capacity of 350 tons, and is drawn off through chutes into cars. These are trammed to the waste washing plant subsequently described. Ore from the two picking belts falls through an inclined chute to a 30-in. cross-belt fitted with a hand-propelled tripper, and so into the ropeway bins. This cross-belt is driven in a similar way to the picking belts—by a 6-h.p. electric motor through reduction gearing.

The waste washing plant consists of a compressed-air-operated shaker-conveyor fitted with a screen bottom. A strong jet of water is played on the waste as it moves forward, tramp quartz being removed by two men. The washed fines pass through the screen bottom and are led to settling tanks, from which they are shovelled to the ropeway buckets, a special siding having been erected for this purpose. Waste moves over the shaker-conveyor into a bin and is trammed to the waste dumps. This somewhat clumsy layout, governed to no small extent by materials available on site, proved itself satisfactory and the value of recovered fines was found to be such that a second electrically-driven shaker was installed with a view to washing all excess waste from the mine which had been in contact with ore. Although promising results were obtained, no recovery figures were available before the mine was forced to close down.

One of the difficulties to be overcome here was that while the ropeway normally works three shifts per day the mine is worked on only two shifts. The original layout was such as to allow cars of ore to be tipped at track level into two small bins ahead of the Locker grizzly feeders. Later, when a compressed-air hoist became available, the size of the bins was increased by raising the cars up an incline. Use was made of the height thus obtained to stock ore on the dump in case of need.

Table XV shows the progressive efficiency figures obtained over three periods: the initial period from September, 1943, to January, 1944; a second period of two months during which time studies were being made of operations; and the third period when all operatives were on incentive.

TABLE XV

HAND-PICKING PLANT (GENERAL DATA)

	Sept., 1943, *to* *Jan.*, 1944	*February and March*, 1944	*April and May*, 1944
Tons treated	115,653	32,180	25,664
Tons of waste hand-picked	40,900	11,357	10,647
Per cent waste picked	35·4	35·3	41·5
,, **fines in waste**	11·8	11·3	11·2
,, **ore in waste**	1·6	1·5	1·4
Average tonnage per hour	56·3	52·1	66·1
Labour Shifts:			
Belts	16,618	1,874	1,483
Other labour	9,915	3,958	1,579
Total labour	26,533	5,832	3.062
Tons of Waste Picked/Man Shift:			
Belts only	2·5	6·1	7·2
Total labour	1·5	1·9	3·5
Efficiency Indexes:			
Overall labour efficiency	100	127	233
Average wage earned	100	120	163
Labour cost/ton treated	100	94	85

Distribution of costs are shown in Table XVI and the effect of the hand-picking of waste on the recovered grade in Table XVII.

TABLE XVI

DISTRIBUTION OF HAND-PICKING PLANT COSTS

	Per cent
Supervision	8·0
Operation (wages)	69·7
Stores	14·6
Power	5·6
Shops	2·1
	100·0

TABLE XVII

GRADE OF MILL RECOVERY

Period	*Per cent*
1939/40	0·624
1940/41	0·628
1941/42	0·724
1942/43	0·717
1943/44	1·119
1944, April/May	1·376

MILLING

There are two concentrating plants on the property—the old mill at Panasqueira and the River Mill on the banks of the River Zezere. The former was used for treating ore from above No. 5 adit level until the total required production could be obtained by treating ore only at the River Mill, when Panasqueira ore was transported by ropeway to the ore bins at Barroca Grande.

The ropeway station is provided with a turnout with a capacity of 700 tons so that, in the event of a crushing plant stoppage, the ore coming in over the ropeway can be dumped, and treated later when convenient.

A description of the various sections of the mill follows, with diagrams of main-circuit flow-sheets shown in Figs. 8 to 11.

Crushing Plant.—Normally the ropeway buckets are tipped over a 2-in. grizzly into a 30-in. by 18-in. Edgar Allen jaw crusher. which reduces ore to 2 in. Undersize from the grizzly, together with the crushed product, passes to a rotary log washer fed by a jigging feeder. Washing is made necessary, particularly in winter, by the clayey nature of the phyllite fines. The washer and associated screens produce the following products :

— 2 mm. to 20-mesh screen in gravity plant.
— ½ in. direct to crushed ore bin.
+ ½ in. to picking belt.

Before the erection of the hand-picking plant at Barroca Grande, some 18 per cent of waste was removed from the ore at a rate of about five tons per man shift. At present a small quantity of high-grade selected ore is removed from the belt and sent direct to the magnetic separator plant.

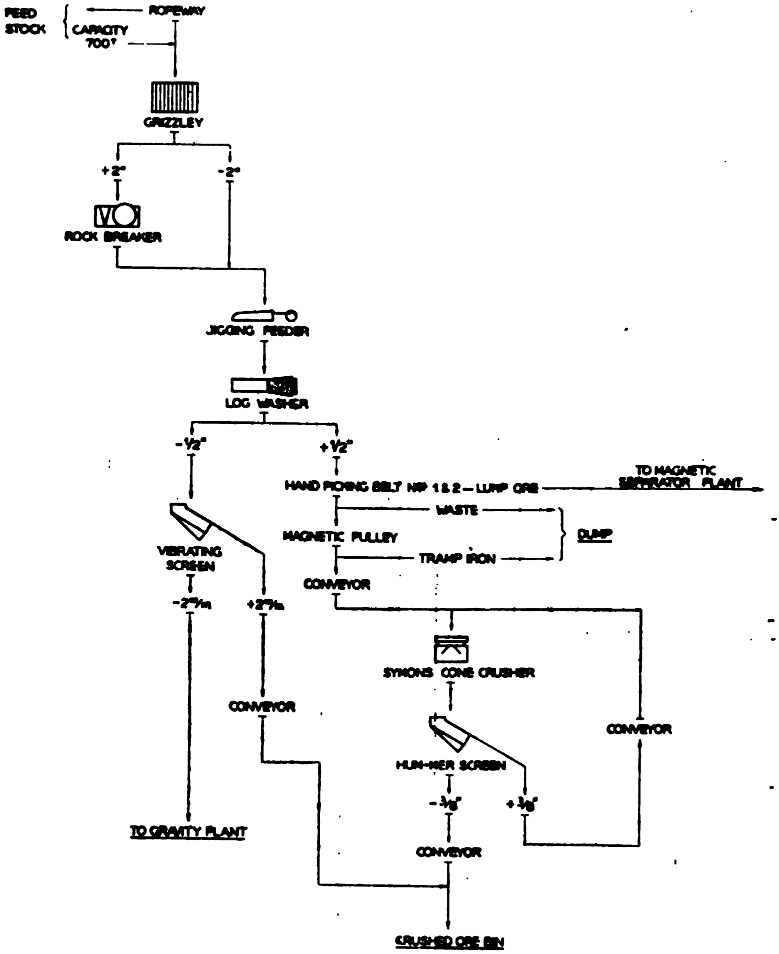

FIG. 8.—Flowsheet of crushing plant.

The ore then passes to a 3-ft. Symons cone crusher in closed circuit with a Type-480 Hum-mer screen, fitted with a $\frac{3}{8}$-in. by 1-in. wire mesh cloth, the undersize from which is discharged, through a hand-propelled tripper, to a 24-m. by $7\frac{1}{2}$-m. flat-bottomed bin with a total capacity of 1,500 tons.

The average life of the more important machine parts in the crushing plant is as follows :

	Tons
Manganese steel crusher jaw plate..............	67,800
Symons bowl and mantle liners..................	34,000
⅝-in. by 1-in. wire mesh screen cloth...........	45,200

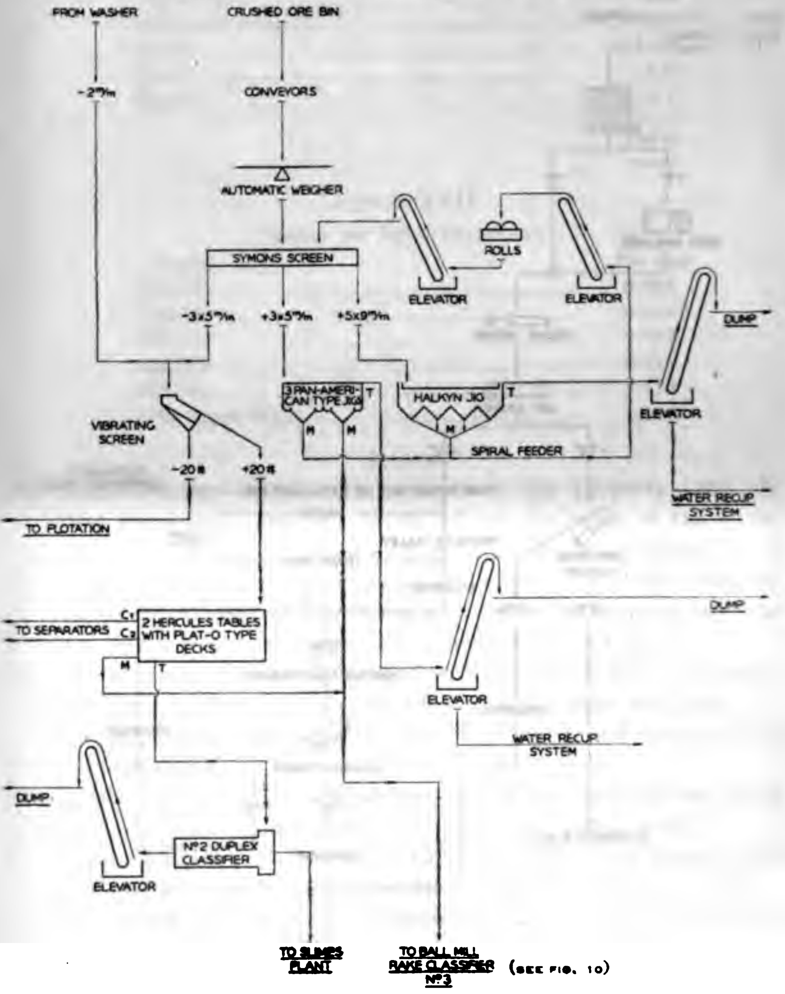

FIG. 9.—Flowsheet of coarse section gravity plant.

Coarse Gravity Plant.—Ore is drawn from the bottom of the crushed-ore bin by belt feeders, of which there are four feeding the main conveyor, which has a width of 16 in. and a speed of 37 metres per minute. Ore passes over an automatic weight recorder to the head of a set of screens. These consist of an 8-ft. by 4-ft. Symons double-deck jigging screen and an 8-ft. by 4-ft. mechanical vibrating screen, which give the following products :

	Per cent
+ 5 by 9 mm. to Halkyn jig	52
+ 3 by 5 mm. to three Pan-American type jigs	22
+ 20 mesh to two Hercules tables with Plat-O type decks	18
— 20 mesh to flotation	8

Average screening efficiency as given by the percentage of undersize in the oversize is :

	Per cent Undersize
Coarse product (+ 5 by 9 mm.)	25·7
Medium product (+ 3 by 5 mm.)	44·6
Fine product (+ 20 mesh)	9·3

The Halkyn jig is a moving-tray type with an 8-ft. by 2-ft. tray. Both this machine and the fine jigs are operated to give middling products and a clean tailing. Halkyn jig middlings, assaying about 5 per cent metal, are fed through a spiral feeder to a bucket elevator, which raises them, together with middlings from the first hutches of the fine jigs, to 24-in. rolls, the product of which is returned to the Symons screen. All jig tailings are discarded.

The coarse tables produce rough concentrates of two grades, averaging 25 per cent and 10 per cent respectively, which are treated separately in the magnetic separator plant. Middlings flow to the ball-mill and tails to No. 2 Stokes Duplex rake classifier, for dewatering and thence to waste.

Sand Gravity Plant with Pre-Flotation.—The *minus* 20-mesh product from the vibrating screen at the head of the mill goes to Stokes rake classifier No. 1. The overflow of this classifier is sent to a 15-ft. Dorr thickener, the underflow of which is raised by a diaphragm pump to flow to a conditioner, taking with it the sand product of the same rake classifier. The reagent consumption in 1941 was :

	Lb./Ton
'301' at 10 per cent solution	0·46
Cresol	0·16
Pine Oil	0·11
Gas Oil	0·14
Soda	0·02
$CuSO_4$	0·34

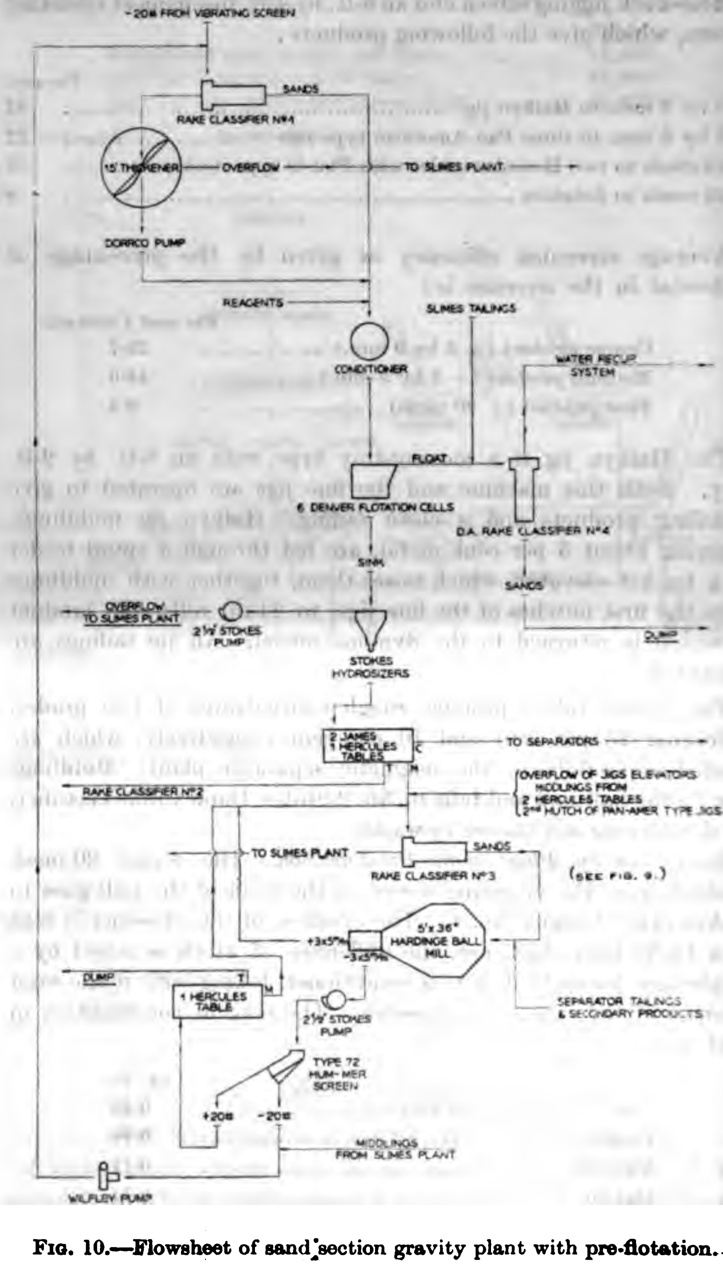

FIG. 10.—Flowsheet of sand section gravity plant with pre-flotation.

Supply difficulties led to tests being made with a view to reducing the quantities of reagents used and, latterly, consumption was as follows :

	Lb./Ton
' 301 ' at 10 per cent solution	0·46
Cresol	0·13
Pine Oil	0·05

The conditioned pulp with a water/solids ratio averaging 3 : 1 is treated in six Denver ' Sub-A ' No. 18 Special flotation cells for the removal of sulphide minerals. The sulphide froth, carrying less than 0·20 per cent metal is discarded, via the Stokes double-acting rake classifier No. 4. Characteristics of the sulphide float are :

Screen Analysis			*Per cent* WO_3
+ 30	mesh	0·66 per cent	0·17
+ 100	,,	20·00 ,,	0·06
+ 200	,,	51·33 ,,	0·09
— 200	,,	28·00 ,,	0·37

The sink product from flotation flows to a batch of three Stokes hydrosizers and is distributed to a Hercules table and two James tables. These three tables make a rough concentrate, carrying 30 to 35 per cent metal, which goes to the drying plant ; a middling, rich in sulphides, which is returned to the ball-mill for further grinding ; and a tailing for discard, via No. 2 rake classifier.

Middlings from all sand tables, together with the second hutch product from the fine jigs, and all floor washings, are sent to rake classifier No. 3. The overflow from this classifier flows to the slime treatment plant, while the rake product goes to a 5-ft. by 36-in., Hardinge ball-mill in closed circuit with a 5-ft. by 4-ft. Type 72 Hum-mer screen fitted with a 20-mesh cloth. A Hercules table is placed in this circuit for the removal of a low-grade tailing. The undersize from the 20-mesh screen is elevated by a 2½-in. Wilfley pump to rake classifier No. 1 at the head of the flotation plant.

Consumption of manganese steel balls, when these were obtained from England, was of the order of 0·07 kg. per ton of original mill feed. Latterly balls cast in the company's foundry gave similar results. The average life of ball-mill liners is as follows : Middle-section liners, six months ; end-section liners, 12 months.

Slimes Gravity Plant.—The slime overflow from the 15-ft. thickener, and from rake classifiers Nos. 2 and 3, flows to a 35-ft. Dorr thickener. The thickened pulp is pumped by a 2½-in. Stokes sand pump to a Richard Janney hydraulic classifier, which also receives the overflow from the hydrosizers. The spigot products are treated on six James slime tables, which give a low-grade concentrate, a middling which is returned to the flotation circuit, and a tailing which is pumped to Stokes D.A. rake classifier No. 4, the rake product being fed to the waste bins and the overflow going to the water recuperation plant.

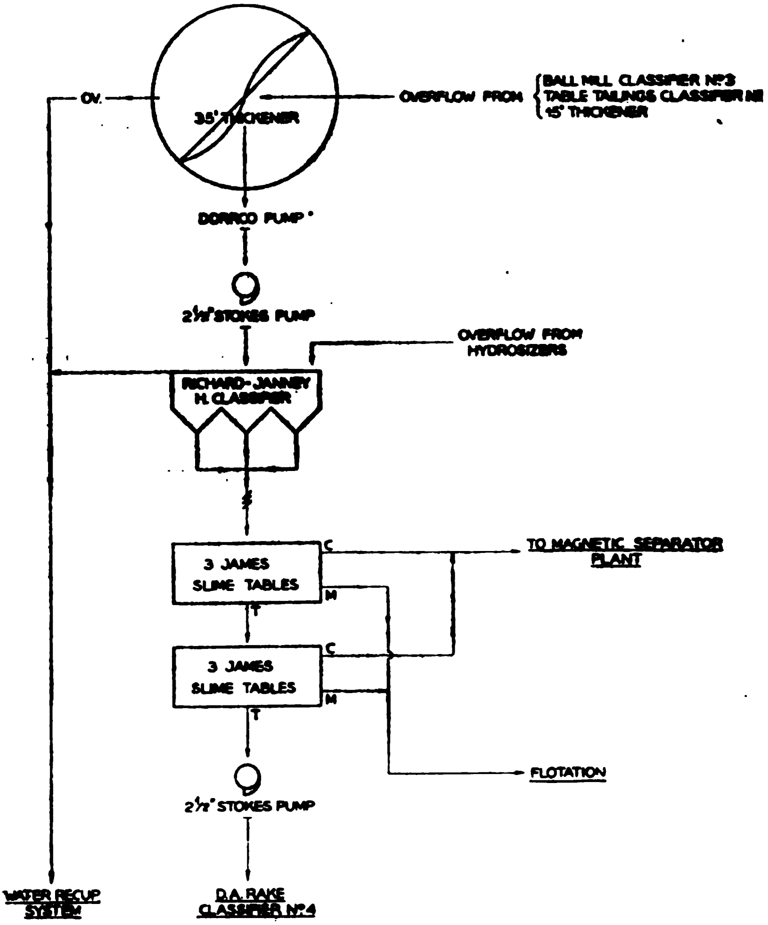

FIG. 11.—Flowsheet of slimes gravity plant.

DRYING AND MAGNETIC SEPARATOR PLANT

Concentrates from all tables are collected in mild steel boxes, having a capacity of 200 kg., which are transported on Morris trucks fitted with lifting devices to an inclined hoist, which elevates them to the head of the drying plant where they are weighed.

Rich gravity plant concentrates from sand and flotation tables are dried separately in six Chinese dryers. They are then raised in a vertical Morris elevator and fed from an overhead gantry to their appropriate magnetic separators through discharge hoppers, each having a capacity of about 600 kg.

Low-grade concentrates from the gravity-plant sand tables are dried in a Huntington-Heberlein wood-fired circular shelf-dryer. This also receives, and handles separately, selected ore from the mine and crushing plant, previously crushed in a small rock-breaker to pass a 1 to 2-in. screen. The dryer feeds concentrates by means of an elevator to a 3-mm. Hum-mer screen, the oversize of which is crushed by a set of 18-in. rolls in closed circuit with the screen. The product from this screen is raised by the inclined hoist to a double-deck Hum-mer screen, where it is divided into three products—*plus* 26-mesh, *plus* 40-mesh, and *minus* 40-mesh, which are then weighed and fed to their appropriate separators.

Separator products fall into similar boxes to those in which gravity-plant concentrates are collected. Finished concentrates go to the bagging room, and seconds are run to the Morris elevator which returns them to the gantry level for re-separation or tabling. Non-magnetics from the treatment of wolfram concentrates are returned to the sands plant ball-mill.

Tin Plant.—Non-magnetics, if they contain tin, are roasted in a Brunton calciner, concentrated in a jig and on a table, after screening, dried in a Chinese dryer, and again separated. Magnetic products, after crushing in an 8-in. rolls, are acid treated in a pan agitator and retabled. Concentrates from the tables are dried and sent again for magnetic separation. Non-magnetics are bagged as 70 per cent Sn.

Wolfram.—Wolfram concentrates are shipped, after some subsidiary tabling when this is required, at an average grade of 68·5 per cent WO_3.

Characteristic ratios of concentration are as follows :

Tons of mill feed per ton of rough concentrates	40
Tons of rough concentrates per ton of final concentrates	3·0

(Figures include lump ore production.)

Water Supply.—The mill is situated on a steep hillside 80 metres above the normal level of the River Zezere. Owing to the wide difference in level between high and low water (a maximum of six metres) the pumping plant for make-up water for all industrial purposes at the River Section consists of two 4-in. 13-stage bore-hole pumps by Sulzer direct-driven by vertical motors. The pumps are erected on a pier above high-water level and can handle low-water conditions.

The overflow from the 85-ft. thickener, together with the overflow of rake classifier No. 4 dealing with slime tailings and pyrite float, is pumped by three 50-h.p. Wilfley sand pumps, one of which is in reserve, to a 71-ft. Dorr traction thickener from which a thickened slime product (about 1 : 1 W/S ratio) is withdrawn and dumped. The clear water is distributed for re-use.

Approximately one ton of make-up water is required per ton of ore milled. Total milling requirements are approximately four tons of water per ton of ore.

Tailings Disposal

The original choice of mill site was not a fortunate one and clearly never envisaged treatment on the scale of the last 10 years. The choice was evidently governed by the proximity of the water supply rather than with a view to future dump space.

After 1941 hand-tramming gave way to mule transport and in 1942 a serious situation developed in so far as further dumping space was not available close to the mill. Some form of mechanical transport was required to reach suitable dump sites 1·2 kilometres distant. The choice being limited by the materials available in the country, two second-hand 60-h.p. locomotives modified for wood firing were acquired, together with 12 wagons with a capacity of 2½ cu. m. each; 1,900 metres of 750-mm. gauge track were laid, using 20-kg. rails.

General

The following are noteworthy features in the described flow-sheet :

(1) The early elimination of clean tailings at coarse sizes.

While this was one of the principal objectives of the mill design, it is only fair to say that its success is due to the nature of the ore, which is remarkably free from finely-disseminated mineral.

A metal balance sheet showing distribution of mill feed and the distribution of the gravity mill products is shown in Table XVIII. These figures do not include material removed by hand-picking.

(2) The inclusion of flotation in the circuit, as distinct from batch work usually adopted in tin and wolfram milling.

The average sulphide content of the ore is about 4 per cent. in the ratio of two-thirds pyrite to one-third arsenopyrite. The sulphide percentage in the flotation feed is, however, increased to 15 per cent by reason of the elimination of coarse tailings.

The disadvantages that might be expected from this procedure are : (*a*) Possible sliming of wolfram by attrition in the cells, and (*b*) slime losses of wolfram as a result of grinding in the middlings

TABLE XVIII

METAL BALANCE SHEET

RIVER MILL

April/September, 1943

IN	*Per cent Dist.*	*Tons*	*Per cent Metal*	*Metal Units*	*Per cent Dist.*	OUT	*Per cent Dist.*	*Tons*	*Per cent Metal*	*Metal Units*	*Per cent Dist.*
FEED—						PRODUCTION	1·05	1,240	68·50	84,940	89·5
Lump Ore.........	1·22	1,448	21·64	31,335	33·0	TAILINGS—					
						Halkyn Jig	40·75	48,263	0·07	3,378	3·6
Log Washer......	7·27	8,609	1·98	17,046	18·0	Pan-American Jigs	18·78	22,242	0·05	1,112	1·2
						Tables...............	30·43	36,040	0·09	3,244	3·4
Coarse Feed......	91·51	108,377	0·43	46,523	49·0	Flotation...........	2·63	3,116	0·16	498	0·5
						Slime Tables	6·36	7,533	0·23	1,732	1·8
						TOTAL TAILINGS ...	98·95	117,194	0·09	9,964	10·5
TOTAL IN	100·00	118,434	0·80	94,904	100·0	TOTAL OUT	100·00	118,434	—	94,904	100·0

ball-mill. As slime losses amount only to some 17 per cent of the total wolfram lost in the tailings, this does not appear to be a serious factor. On the other hand the elimination of sulphides in this manner makes for greater simplicity in the circuit.

(8) The feeding of comparatively low-grade concentrates to the magnetic separators.

The low capacity of magnetic separators and the number of secondary products that result has not encouraged their use as concentrating units. They are normally considered as a necessary evil in the final treatment of complex concentrates whose constituents cannot be segregated by any other means. Provided the valuable minerals are reasonably free from chatty particles at the upper limits of 2½ to 8 mm. they offer the great advantage that, apart from dusting losses, material sent to this section of the plant can be treated and retreated, without the risk of loss, until the products are satisfactory. In practice, of course, there is a limit to retreatment, but all separator plant residues are finally returned to the main circuit.

The costs of treatment in this section of the plant, including bagging for export, amount to 25 per cent of the total milling costs.

Costs.—The distribution of milling costs for the financial year 1940/41 was :

	Per cent
Supervision	5·2
Operation (wages)	19·4
Stores, suspense, and mill spares	36·5
Power	19·2
Shops	3·3
Mill, general	6·5
Sampling and assaying	3·1
Water supply	6·8
	100·0

Mill Power Plant.—This consists of three 250-h.p. two-cylinder Tangye suction-gas engines, belt-connected by Lenix drives to three 184 kW d.c. 220-V generators. A 180-h.p. Campbell gas-engine belt-coupled to a 67 kW 220-V d.c. generator is also available for balancing the power supply.

In the past two Tangye engines were sufficient to take the load. Modifications to the mill, with particular reference to improved water control and anti-pollution measures, together with enlarged workshops equipment, made further power necessary. This is drawn from the third Tangye or from the Campbell engine as requirements demand.

Gas is drawn from suction producers, of which there are four supplying the Tangye engines and one supplying the Campbell. The fuel normally employed is Welsh anthracite, preferably ' Big Vein ', with a 10 per cent mixture of local charcoal made from heath roots. During the war, as a result of a shortage of anthracite, a larger percentage of charcoal has been used with detriment to efficient running.

The following statistical data refer to the Tangye engines under normal conditions as prevailed during the year 1940/41 :

KWh generated	2,161,540
Tons of ore milled	288,503
Total power consumption—kWh/ton	7·49
Average consumption of fuel—	
Per engine hour (kg.)	82·8
Per kWh generated (kg.)	0·63
Labour—	
Man-hours on maintenance per 1,000 kWh	0·63
Man hours on operation per 1,000 kWh	13·20
Average hours per engine/month	459
Milling time lost per year due to power failures	1h. 35m.

As far as possible each unit in the mill is driven by individual electric motors through V-belts.

Sampling and Assaying

The laboratory is responsible for all sampling and assaying, the latter being carried out by two methods—that of chemical analysis and Superpanner assays. A description of the Superpanner is given in the *Transactions* of the Institution, Vol. 49 (1939-40), page 716.

The work of the laboratory can be divided into :

Daily control samples of lump ore, log-washer ' fines ', all tailings, both from gravity and separator plants, and of any product transferred from one mill section to another. These samples, except for that of lump ore, are assayed by the Superpanner. No sample of coarse feed to mill is taken, as after many tests it was found that, unless so large a sample is taken as to necessitate a separate plant, representative results cannot be achieved owing to the heterogeneous nature of wolfram mineralization. A figure for the value of this feed is therefore calculated by difference.

Monthly control samples of all the above by chemical assay.

Periodic control samples of Superpanner concentrates by chemical assay.

Control of magnetic separator products for the preparation of concentrates for shipment and the sampling of these by chemical assay.

Samples for test purposes, control of individual machines, sizing tests, etc.

Assays for outside concerns.

A high-grade concentrate is obtained from the Superpanner, to the weight of which a factor is applied to give the true metal content expressed as a percentage of WO_3. All Superpanner concentrates are kept and are chemically assayed from time to time, the average to date assay value of each type of sample being used as the correction factor. These control assays gave the results, shown in Table XIX, for Halkyn jig tailings between December, 1940, and May, 1942.

TABLE XIX

SUPERPANNER CONCENTRATES FROM ASSAY OF HALKYN JIG TAILINGS

Date of Control Assays	*Period to Which Result Refers*	*Superpanner Concentrates, Per cent WO_3*	*Factor Applied, Per cent WO_3 to date*	*Per cent Error of Factor*
April, 1941	Five months	68·44	68·00	— 0·65
August, 1941	Four ,,	69·79	68·44	— 1·97
December, 1941 ...	Four ,,	65·67	69·04	+ 4·88
May, 1942	Five ,,	65·45	68·00	— 3·75

TABLE XX

COMPARISON BETWEEN CHEMICAL AND SUPERPANNER ASSAYS

Sample No.	*Chemical Assay*				*Superpanner Assay*		
	Per cent WO_3	*Per cent Sn.*	*Per cent Total*	*Per cent Difference*	*Per cent Difference*	*Metal = wt. × F. Per cent*	
1520	0·23	0·09	0·32	− 0·04	− 12·5	0·36	0·56 × 65
1521	0·35	0·10	0·45	+ 0·06	− 13·3	0·51	0·78 × 65
1522	4·07	0·18	4·25	0·08	1·9	4·17	6·42 × 65
T5	0·31	0·04	0·35	− 0·02	− 5·7	0·33	0·50 × 66·45
T6	2·47	0·02	2·49	− 0·10	− 4·0	2·39	3·48 × 68·57
T6 (repeat)	2·40	0·02	2·42	0·03	1·2	2·39	3·48 × 68·57
1523	1·77	10·17	11·94	− 0·60	− 5·0	11·34	16·20 × 70
1523 (repeat)	1·69	10·22	11·91	0·49	4·1	11·42	16·32 × 70

The largest difference between the factor applied and that which should have been used was about 5 per cent, so that results were too high to this extent during the four months September to December, 1941. However, the effect of this error on tailings assays is negligible and results are well within the degree of accuracy obtainable by chemical assay.

Superpanner assays have been checked by direct chemical assay. Some comparative results are shown in Table XX.

While the accuracy of the Superpanner depends to no small degree on the dexterity of the operator, who should work towards getting a concentrate with a constant grade and as low a tailing as possible, it has proved of great utility in assays for daily control purposes and for test work, as results can be obtained at short notice. It is further considered that the element of uncertainty in the sampling, as well as the error inherent in low-grade wolfram assays by any of the known chemical methods are such as to warrant the use of the Superpanner for the purpose of obtaining a close estimate of the value of tailings.

Wolfram Assaying.—The current method used for the assaying of wolfram consists of :

Attacking the sample with HCl, adding HNO_3: diluting and treating with a 10 per cent solution of cinchonine : allowing to stand for eight hours : filtering and washing the precipitate with a 1 per cent solution of cinchonine : adding a few drops of methyl orange : slowly calcining the filtered residue, removing the silica by the use of a mixture of hydrofluoric and sulphuric acids : cooling and weighing: cleaning the residues with alkaline carbonates. This method is used for samples containing more than 40 per cent WO_3.

A similar method is used for low-grade samples, except that the process takes longer, reprecipitation by means of cinchonine being required after dissolving the WO_3 in ammonia and soda for the removal of impurities.

Latterly, tests have been run on a method described in the *Revista da Ordem dos Engenheiros* of Portugal. This is a volumetric assay and, in short, consists of :

Attacking the sample with HCl followed by HNO_3 and aqua regia until no dense dark particles are to be seen : diluting and filtering: washing by decantation with 1/20 HCl and again with a 1 per cent solution of HNO_3: filtering and washing again with the same solution until the precipitate is free from acid : transferring to beaker and adding sufficient standardized solution of NaOH as will dissolve all the WO_3, *plus* a small excess: mixing well, diluting and adding a few drops of phenolphthalein and heating to between 60° and 70° Centigrade for a few minutes and until no yellow particles are visible: adding a few more drops of NaOH: cooling and titrating against a standardised solution of HCl.

The author of the paper describing this method claims that results can be obtained to within 0·20 per cent WO_3, which has been confirmed by tests run in the laboratory, as shown in Table XXI, and which makes the method suitable for all but low-grade wolfram samples.

TABLE XXI

COMPARISON BETWEEN GRAVIMETRIC AND VOLUMETRIC ASSAYS

Sample No.	*Gravimetric Method*		*Difference +/—*		*Volumetric Method*	
	Per cent WO_3	*Average per cent*	*Per cent WO_3*	*Per cent Error*	*Average per cent*	*Per cent WO_3*
Rich Samples :						
R365	68·22 68·50 68·33	68·35	—0·22	—0·3	68·13	68·20 68·10 68·10
C461	63·34	63·34	+0·26	+0·4	63·60	63·60
C462	66·68	66·68	+0·22	+0·3	66·90	66·90
Medium-Rich Samples :						
C406	46·90 46·97	46·93	—0·17	—0·4	46·76	46·90 46·70 46·70
S69	29·45	29·45	+0·05	+0·2	29·50	29·70 29·30
S77	17·10	17·10	—0·05	—0.3	17·05	17·10 17·00
Low-Grade Samples :						
S84	7·09	7·09	+0·21	+3·0	7·30	7·30 7·30
S55	3·19	3·19	+0·21	+6·6	3·40	3·40 3·40
T6	2·47	2·47	—0·07	—2·8	2·40	2·40

The advantages of the new method over the gravimetric method are :

(1) *Rapidity.* Results can be furnished by the laboratory after nine hours, as compared with 25 hours in the case of the gravimetric method.

(2) *Economy.* No platinum crucible is required. No cinchonine is used. Less labour and fuel are required. It is estimated that an economy of over 60 per cent is obtained by the employment of the volumetric method for WO_3 assays.

There is the added advantage that the undissolved cassiterite remaining after taking up the WO_3 can be used for determining the tin content of the sample without having to prepare a new sample for this purpose. An economy of about 35 per cent is obtained per tin assay by so doing, and results compare favourably with those obtained from specially-prepared samples.

Comparative results using in the first place a specially-prepared sample and secondly the residues from the wolfram assay are shown in Table XXII.

TABLE XXII

TIN ASSAYS

Sample No.	*Direct Assay Per cent Sn.*	*Using Residues Per cent Sn.*	*Difference*	
			Per cent Sn.	*Per cent Error*
R352	0·06	0·06	—	—
R365	0·12	0·10	—0·02	—17
UKCC–1447	0·47	0·42	—0·05	—11
„ 1448	0·22	0·20	—0·02	— 9
„ 1449	0·32	0·33	+ 0·01	+ 3
„ 1450	0·56	0·56	—	—
„ 1452	2·61	2·56	—0·05	— 2
„ 1453	0·36	0·32	—0·04	—11

It is quite clear from Table XXI that, with an error of the order of 0·2 per cent WO_3, the volumetric assay is not suitable for assaying low-grade samples, as for example in the case of tailings. However, as the Superpanner is used for most daily control purposes and as a 65 per cent WO_3 concentrate is obtained from it, the use of the volumetric method can be extended to the majority of samples where a high degree of accuracy is not required.

REPAIR SHOPS

The dispersed location of operations, with two mine sections six kilometres apart and with the mill nine kilometres distant from the nearest, required that repair shops be available in all sections, with consequent duplication of equipment. Details are:

River Mill.—Operations making most use of the services of the shops are those of milling and it has therefore been the policy to concentrate the majority of equipment in this section. Shops consist of mechanical workshops, forge, electrical repair shop, foundry and carpentry. The mechanical shops, which have been especially increased during recent years as a result of the difficulty of obtaining spare parts, are now equipped with seven lathes, handling work up to three metres in length and two metres in diameter, a milling machine, mechanical hacksaw, two radial drilling machines, two slotting machines, plate bending machine, plate cutter, two electric welding sets (one stationary and one portable), hand press, and all necessary complementary tools.

The foundry consists of a cupola furnace for cast iron, with a maximum capacity of 1,000 kg. per hour, three crucible furnaces for brass and whitemetal, and a pattern drying plant, all installed in a suitable building with a floor area of 180 square metres. An adjoining building houses the pattern manufacturing shop and store. The output of the foundry depends on requirements and during the financial year 1943/44 was :

	Brass/Bronze	*Whitemetal*	*Cast Iron*
Weight of finished articles (kg.)	8,838	1,129	27,210
Per cent of scrap metal used ...	40	70	90

With the difficulty of obtaining imported spare parts during the War, the foundry offered the advantage of rapidity of work and cheapness, as compared with local suppliers, and dealt with the majority of requirements up to castings weighing 350 kg. The largest casting made was that of a V-belt pulley with a diameter of 1·80 metres. Under normal conditions, the economics of mine foundry work depends on the cost and life of the spare parts manufactured, and details have been kept for future comparison and use.

The carpentry contains sawing and planing machinery and is designed with ample room to handle the repair of tables and launders.

All work in the shops is performed against job orders and a card system is used for costing. New articles manufactured, as distinct from repair jobs, are supplied to the general stores at cost and are issued against requisitions, as obtains with purchased articles.

Thus a check is kept to see that articles are not manufactured on site which can be purchased more economically, unless urgency or additional life warrant same.

An average of 159 fitters, carpenters, foundry men and apprentices were employed per day at the River Mill shops during the year 1943/44. A school for apprentices, usually the sons of workmen, with practical and theoretical training, was run by the engineering staff for several years, until the stoppage of operations.

Panasqueira.—The original shops were in this section, but equipment has been reduced since the closing of the mill. They now remain primarily to deal with some of the mine requirements and contain two lathes, a drilling machine, a slotting machine, forges, and a mechanical saw. Little carpentry work is done except for certain mine requirements.

Barroca Grande.—The shops here are limited to heavy mine repairs not requiring the use of mechanical tools and consist chiefly of blacksmith work and welding, for which an electric welding set is available. In addition to the work mentioned under drill-sharpening, a special section deals with the maintenance of machine drills and the fitting of spare parts.

The carpentry shop in this section is fully equipped with woodworking machinery, erected for the purpose of manufacturing in series all woodwork for house building during the period of intense construction.

Neutralization of Water

Following complaints that crops grown on the river banks or irrigated by river water were being spoilt by acid water from the mine and mill, the problem of its treatment was taken in hand at Barroca Grande and at the River section.

Barroca Grande.—Mine water flows from the Main Adit at a rate that varies between 450 and 1,300 litres per minute. The acidity of this water ranges between 25 and 1,300 mg. of H_2SO_4 per litre, the higher figure being observed during the wet season, as a result of seepage through the old stopes.

Mine water flows to a neutralizing plant, where it is treated by finely-ground slaked lime automatically added as the water passes into a settling tank. The bottom of this tank is fitted with discharge valves from which the precipitated ferric salts are withdrawn. The precipitate, containing some 10 per cent solids, is led to settling tanks sunk in the waste dumps, where it is dried in the sun and shovelled out by hand. These tanks also receive the slime overflow from the waste washing plant. The clear water is discharged over a knife edge and, after passing through a sand filter bed, is pumped for industrial purposes, or is allowed to flow into the river.

The amount of lime added, which varies between 180 and 800 kg. per 24 hours, is such that the majority of ferrous salts have been oxidised and precipitated and that the water discharged is faintly alkaline. The company successfully cultivates land irrigated by this water, as a proof of the efficacy of the system, producing green pasture and maize for feeding the mules.

River Mill.—At the River mill the chief difficulty was the disposal of slimes. These and all water is now pumped to a 71-ft. thickener, as described under milling, lime being added to the pump inflow. Slimes are discharged through a diaphragm pump to tailings cars, where they are mixed with table and jig tailings and dumped. It is found that this mixture sets hard in a short time and does not flow into the river. Rainwater, seeping through the old dumps, is collected in a ditch running along the river bank and is treated with scrap iron to remove any copper salts, followed by lime. The quantity is small and does not, therefore, require the more elaborate plant necessary for mine water at Barroca Grande. Samples of river water are taken at regular intervals above and below the mill and results are controlled by the Government Mines Department.

Study Department

In 1939 the difficulties of supervision and of poor labour led to the decision to call in 'Associated Industrial Consultants' to organize a study department capable of training supervision personnel, to put labour on a rational and equitable incentive system, and to develop a method of measuring the labour cost of various methods of stoping.

The essential principle of the A.I.C. system of study, as applied to this property, is one of measuring the work units expended by each operative in performing the elements of work which, in combination, go to make up the cycle of operations. Work is not subdivided to the degree required by other systems of work measurement and, as applied to mining, where conditions are variable to a high degree both as regards time and place, the system affords only approximate results.

The rating of the effort expended by an operative is measured in appropriate units of work per hour, a rate of 60 U/H being here considered as one of normal expectancy, or that rate at which men earning day's pay should work. Any rate above a 60 U/H is credited with a bonus in direct proportion to the increased effort expended. In general, no more than a steady 80 U/H can be expected without undue exertion, although men were known to work at higher rates for short intervals.

All incentive schedules require each operative to produce a minimum of 480 units of work in an eight-hour shift, although a percentage of this time will be taken up in resting.

Studymen, of whom there were four, were drawn from young Portuguese of approximate School Certificate standard, and checkers, one to each working place, from the local labour force, the only essential being the ability to read and write. Training of the studymen was performed in about a month of intensive work, the manipulation of the stop-watch to the necessary degree of accuracy being obtained in less than a week. Rating of the expended effort to a degree of accuracy satisfactory to the A.I.C. engineer was obtained in most cases within two weeks, although further time was required for studies under especially awkward working conditions.

Training was given by studying simple repetitive operations, the particular one chosen being the manufacture of concrete blocks in hand tamping machines. The operations were mostly of short duration, requiring agility in the manipulation of the stop-watch, but conditions were good and rating simple. Studymen were encouraged to try out the work themselves and to look out for causes of lost time and ineffective effort, altering the process and the layout when required. The operatives were finally put on incentive and high bonus rates were paid. Production per man shift increased from 47 to 140 blocks per man shift, the latter figure being the average obtained during the manufacture of 170,926 blocks.

A general survey of the operations employing labour was first made by the A.I.C. engineer. It was found that 60 per cent of the labour force was employed in the mine, of which 40 per cent can be grouped under the heading ' Stope Trammers and Muckers ', whose essential work is the removal of broken muck from the face and the building of pack-walls for hanging-wall support. It was therefore decided to start with the application of this group of operatives. As already mentioned the system of stoping was altered at the same time that the Study Department was formed at Panasqueira, so that a true comparison between past labour efficiencies, and efficiencies after application, cannot be taken by a mere examination of the figures for labour employed per square metre. The system of semi-shrinkage stoping, which was in operation immediately before the Study Department was organized, is more costly in labour than the scatter-wall system introduced at that time. However, semi-shrinkage stoping was never performed to completion, as is shown by the large areas of pack-walls awaiting re-mining after the system had been in operation for 18 months. For, at September, 1938, there were 48,000 sq. m. of pack-walls awaiting re-mining, against a normal 10,000 required by the system. To obtain a true comparison of the labour employed before and after application, the labour required to stope the excess 38,000 sq. m. would have to be taken into consideration. In spite of this excess work, labour productivity increased as shown :

	Stope Trammers and Muckers/sq. m.
1939/40	1·26
1940/41	0·83
1941/42	0·94

In 1939/40 the figure for Panasqueira section, the first to be applied, was 1·08 as compared with 1·24 for the year 1938/39.

Before the application of the studies method it is calculated that the rate of work of the average operative underground was of the order of 45 U/H. After application of the tramming and mucking labour in the stopes, the average unit hour of this group of operatives rose to 72. This increase in efficiency resulted, therefore, in a labour saving of 33 per cent and an increase in wages of 20 per cent.

The Study Department has perforce to be an unpopular organization among the engineering staff, as its work is confined to no particular section of the operations and it has no direct executive control. Its business is to work out the terms for the creation of a willing and contented labour force, earning good money, with its entire effort directed towards the production of useful work. This can be achieved only by obtaining the full co-operation of the engineers in direct control of operations, who must see that Study Department recommendations are put into effect. In fact, the success of the application in its early stages was in direct proportion to the interest and co-operation given by the section engineers.

Here the presumably uncommon feature must be emphasized: that of a labour force willing and keen to work under the incentive system, but with supervision showing a lack of interest which at times became little less than open hostility. This was the greatest difficulty that had to be overcome and, although finally surmounted, no real co-operation can be said to have been achieved except in a few individual cases. Any incentive system breaks down unless adequate supervision is available to see that bonus is not earned at the expense of the quality of the work performed and while, if bad work is performed, the incentive system is usually blamed, it is in fact more often insufficient or inexperienced supervision that causes the trouble. The fact that many of the stope bosses, and even some of the shift bosses, could neither read nor write goes a long way to explain the difficulty of obtaining adequate supervision.

Other essentials for any incentive system to be successful are that the labour force should be fully cognisant of the system and that there should be present the urge to earn higher wages. When, therefore, in 1942 a shortage of labour was felt, which was overcome by recruiting men from distant regions of the country, the new men employed had no knowledge of the system. Furthermore, as the labour turnover rose, it became impossible to keep all the men informed and interested. In addition, a war bonus of 60 per cent diminished the urge to earn higher wages by an increased expenditure of effort. The system, therefore, slowly fell into disuse, except in isolated cases where control was easier and labour turnover small. Mine labour efficiencies fell as shown :

	Trammers and Muckers/sq. m.
1942/43	1·55
1943/44	1·62

The earlier success of the system had been due not only to the fact that labour increased its effective effort and willingly improved its output with a view to earning higher wages but that supervision had taken on a different outlook on the whole problem of labour productivity and utilization.

Although the incentive system, as far as mine labour was concerned, fell into disuse in 1942 for the reasons stated, the Study Department was maintained and became more an organization for carrying out tests and for measuring the efficacy of various systems of mining than for the application of labour on incentive. It continued to investigate the reasons for any deterioration in efficiency and to advise the mine executive of any factors interfering with efficient work, and was also responsible to the manager for all control statistics.

PERSONNEL

The distribution of labour at the end of 1940 is shown in Table XXIII.

Staff consisted of six British engineers, 22 Portuguese engineers and assistant engineers, together with 75 office, stores, timekeeping, and medical personnel. Administration is divided into technical and commercial sections, each with its superintendent under the manager. The superintendent of the former is the engineer in charge of the Study Department and statistics.

TABLE XXIII

DISTRIBUTION OF LABOUR, 1940/41

Mining	2,070
Ropeway	34
Milling	291
Shops	239
Surface and maintenance	233
Guards	18
	2,885

COSTS

The distribution of operating costs for the financial year 1940/41 is shown in Table XXIV.

TABLE XXIV

DISTRIBUTION OF OPERATING COSTS, 1940/41

	Per cent
Mining:	
Stoping	50·3
Development	12·1
Milling	18·0
Ropeway	1·8
Overheads	17·8
	100·0

Taxation and Export Duties.—Taxation in respect of labour, covering unemployment and family allowance, represents 7 per cent of the total payroll, or 8 per cent of total operating costs. The mining tax consists of a nominal tax on the area of the concessions and a 2·6 per cent tax on the value of concentrates produced. Total local taxation amounts, therefore, to about 8·2 per cent of total operating costs ; this is additional to British taxation. On the other hand there are the export duties, which, as earlier stated, convert this low-cost producer into an expensive source of wolfram imported.

WELFARE SECTION

As previously mentioned, one of the company's difficulties was that of obtaining a stable labour force. In 1934 accommodation was limited to seven staff houses, 98 married quarters for workmen, and bunkhouse accommodation for 560 single men. The majority of the labour force was drawn from the surrounding villages and absenteeism amounted to over 30 per cent, especially during periods of agricultural activity, bad weather, and at week-ends. Added to this, the average distance covered daily by the men walking to and from work was as much as 15 kilometres.

By 1938 housing accommodation had been increased by the construction of an extra eight staff houses, 274 married workmen's quarters, and bunkhouse accommodation for 300 men. It was not, however, until the majority of the local population left the company's service to work as tributers that it was found of urgent necessity to construct sufficient houses for the large majority of company's employees who were recruited from distant parts. The difficulty was overcome by the construction of temporary wooden bunkhouses for single men and by increasing the permanent accommodation for staff and labour. By 1943 the total accommodation, including temporary bunkhouses, amounted to :

Staff quarters	Married	44
	Single	30
Workmen's quarters.........	Married	660
	Single	1,482

Some of the married quarters were used for single men, and in October, 1943, a total of 5,251 persons was living on the property, 3,536 of whom were company's employees, the remainder comprising women and children.

Social amenities have been improved and the company's policy has been one of creating a mining tradition and a stable population by encouraging permanent settlement of families in company-owned houses. Preference of employment is given to the sons of men living on the property. Schools have been built in all sections and are run under State control. Wash-houses are also provided. The cultivation of vegetable gardens has been encouraged and a total of some 12 acres of company-owned land has been allocated for this purpose.

A welfare officer is employed to deal with all labour questions, including the allocation of housing accommodation, ration cards (a war-time measure), canteens, sanitation, etc. Record cards are kept showing details of employees and families and monthly attendance.

Medical Services.—All employees are by law insured against accidents, and these, together with the general health of the community, have since 1936 been taken care of by a resident doctor. A part-time doctor also assisted and a second permanent assistant was added in 1943. Before the new hospital was constructed, two small temporary hospitals dealt only with sickness and minor accidents, serious accidents being sent to Lisbon. Three first-aid posts and a fully-equipped hospital are now available. The latter has accommodation for a minimum of 40 persons and includes an X-ray apparatus, operating theatre and all up-to-date medical equipment. The doctors' services are supplied without charge to all employees and their families living in company's houses. All men are medically examined before entering the company's service and their record of accidents is registered. Table XXV shows statistics for the year 1943/44. The death rate due to accidents over the last five years has averaged one death per 384,000 shifts worked.

TABLE XXV

ACCIDENT AND SICKNESS STATISTICS, 1943/44

Number of accidents involving the loss of one or more days :		
Underground	257	
Surface	182	
		439
Total loss of days due to accidents		9,000
Accidents not involving the loss of days		10,419
Hospital cases treated :		
Sickness	636	
Accidents	295	
		931
Patients treated other than accident cases		10,268
Accidents :		
Days lost per 1,000 shifts worked		4·88
Accidents (involving the loss of days) per 1,000 shifts worked		0·24

Canteens.—During the war, when most items of food were scarce, the company organized and subsidized canteens in each of the three sections, principally for the use of bachelors. During the financial year 1943/44, 1,000 soups, 300 meals, 900 bread tickets, and 500 wine tickets were supplied per day. The average cost of a complete meal represented about 15 per cent of the average daily wage earned.

Provision stores have been built and are run on a non-profit basis, with a maximum monthly turnover of over £10,000. Staple food supplies were subsidized during the war period.

Recreation.—Clubs for all employees are provided in the three sections and there is also a cinema with a seating capacity for 500. These are run by employees, expenses being met by subscriptions and club profits.

Active sports are encouraged and amenities include three football grounds, skating rink, swimming pool, clay-pigeon shooting pitches, basket-ball, and a staff tennis court. Inter-section competition is keen. No village or community is complete without its band; that at Panasqueira ran a popular and financially successful life until 1944.

Church.—The church, situated at Panasqueira, was latterly enlarged and handed over to the Church authorities, and there is a resident Catholic priest, to whose stipend the company contributes.

Fire-Fighting Services.—The company maintains a voluntary fire brigade equipped with a mobile motor-driven pump and complementary equipment. High-pressure water points or water tanks are available throughout the main centres of population and industrial buildings.

ACKNOWLEDGMENTS.

The thanks of the authors are due to the Chairman and Board of Directors of Beralt Tin and Wolfram, Ltd., for their kind permission to publish this paper and to the mine staff for their co-operation.

*** *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.*

CROWTHER & GOODMAN (*Branch of The Argus Press Ltd.*), 2, Carmelite St., London, E.C.4.

Plate I.

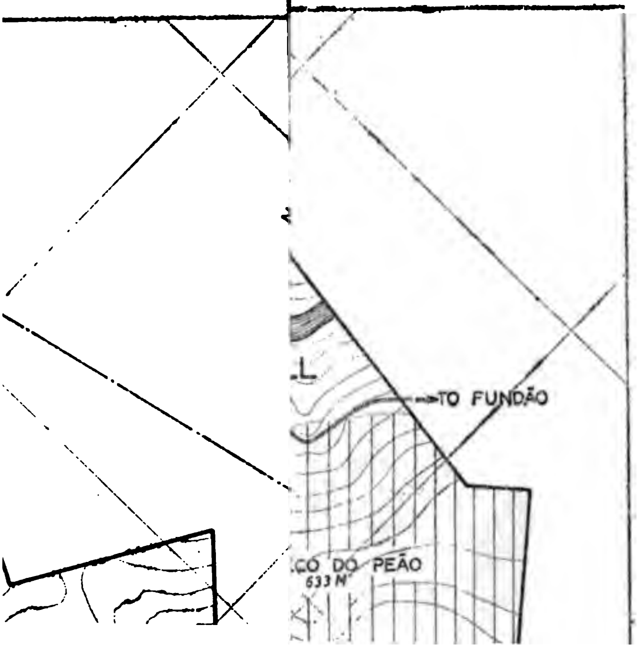

Fig.

J. C. ALLAN, G. A. SMITH, and R. I. LEWIS: *The Panasqueira Mines, Portugal: wolfram mining and milling; labour organization.* *Plate II.*

FIG. 4.—Part of concession showing accidented topography of country and tribute workings on vein outcrops.

FIG. 5 (showing small overlap).

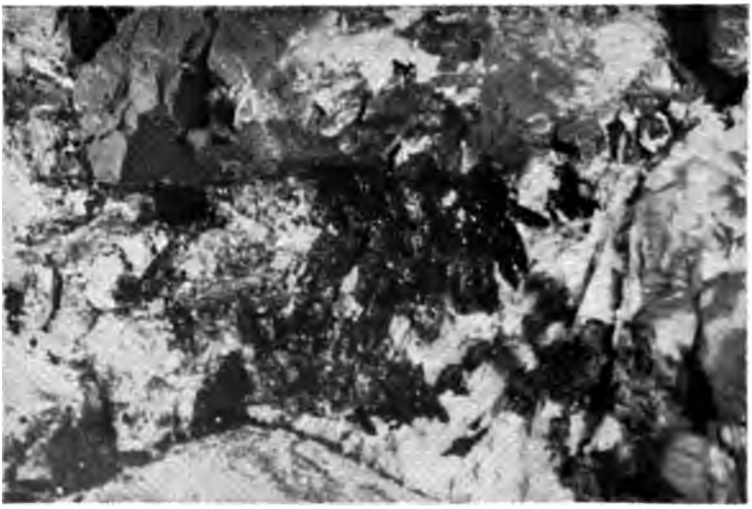

FIG. 6 (showing wolfram enrichment).

AUTHOR'S REPLY TO DISCUSSION*

ON

Notes on the Development of the Blyvooruitzicht Gold Mining Company Limited, South Africa.

By A. Savile Davis, *Member*.

Mr. A. Savile Davis: I would like to express my thanks to Dr. F. E. Keep, who presented the paper, and to those members who contributed to the discussion.

In reply to Mr. Hodgson: Sludge is pumped to a small sump at the 3,000-ft. intermediate station and from there to the surface by a second three-throw pump. The separate column is now being

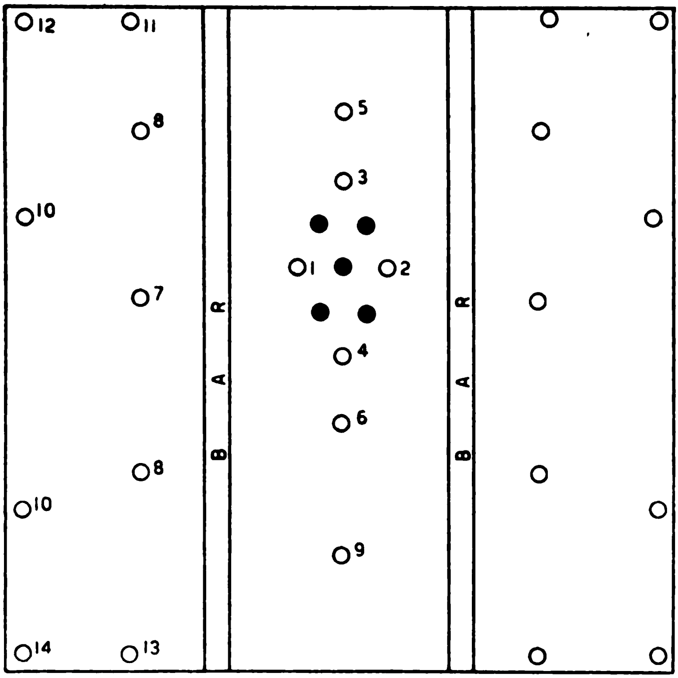

Fig. 4.—Standard burn round, 8 ft. × 8 ft., 30 holes (Blyvooruitzicht Gold Mining Co., Ltd.)

Cut holes are timed singly and only the first one requires to be fully charged. Easers are collared 27 in. from the centre line and finish not nearer than 21 in. from it. Bars are rigged not less than 2 ft. 6 in. apart. It is possible in some cases to reduce the number of uncharged holes from 5 to 4. Shorter jumpers are used for cut holes.

* *Bull.* 477, March, 1946.

[illegible]

[illegible]

[illegible]

[illegible]

[illegible]

[illegible]

FURTHER CONTRIBUTED REMARKS

ON

A Survey of the Deeper Tin Zones in a Part of the Carn Brea Area, Cornwall.*

By Brian Llewellyn, *Member.*

Mr. Robert A. Mackay: Mr. Llewellyn's factual paper has been greeted by a number of tin-mining engineers of Cornish experience with whom I have come into contact abroad as a paper that has long needed writing.

On the economic side his stress for the need of amalgamation, or at least of joint air, water, and exploration plans, leaves no room for disagreement. Alternatively, these mines must die before they need. There is, however, one point on which I would like to suggest a modification of Mr. Llewellyn's views. He stresses the well-known control of mineralization by the killas-granite contact. This is an incontrovertible association, which has been accepted by most geologists of the past to be due to direct derivation from the granite. However, according to a number of recent workers, it is a structural association (1) (2) (3)†, the mineralizing solutions having used the same general channels as the invading solutions, which, together with heat, had converted the pre-existing rocks to granite (4). The granitizing and mineralizing solutions may have had the same source, as well as using the same routes.

Mr. Llewellyn has given a most valuable statistical interpretation to the position of the Cu, W, and Sn zones with respect to this contact and he very rightly makes clear that the relation of any point in this series to the surface is quite immaterial, except for the supergene migration. He does, however, seem to omit a factor. Whether the older or newer theory be correct, the position of the lode relative to the top of the granite, now eroded away, is nearly as important as the lode's relation to the contact. In the case of the older theory, this is because it is farther from the top of the cupola (5), in the case of the latter either by reason of the location

**Bull.* 477, March, 1946.

†Figures in parentheses refer to the list of references given overleaf.

of channels ([5]), or the tendency of a solution to rise within a structure ([6]), before passing from the brittle granite to the semi-pervious killas ([7]).

In either case, but less so in the latter, the farther down in depth the greater the distance from the top of the granite and, therefore, the less the probability of strong mineralization.

There is no reason to expect this change to be sudden, however, and Mr. Llewellyn's recommendations are therefore by no means contra-indicated. This aspect should, however, be borne in mind when planning an exploration and development campaign in depth. For example, there is a distinct likelihood that the total distance from the Cu to Sn zone within a lode, and the depths of the latter, will tend to be somewhat less. Perhaps Mr. Llewellyn can tell us if his data go far enough to say whether this is yet apparent.

In particular this conception calls for a greater proportion of diamond drilling, preferably from underground set-ups, to investigate the conditions ahead before exploration by driving is undertaken.

References

([1]) Locke, A. Granite and ore. *Econ. Geol.*, vol. 36, 1941, p. 448.

([2]) White, C. H. The abyssal versus the magmatic theory of ore genesis. *Econ. Geol.*, vol. 40, 1945, p. 36.

([3]) Mackay, R. A. The control of impounding structures on ore depositions. *Econ. Geol.*, vol. 41, 1946, p. 34.

([4]) Reade, H. H. Meditations on granite. *Proc. Geol. Ass.*, vols. 54 and 55, 1943 and 1944, pp. 64–85 and 45–93 respectively.

([5]) Emmons, W. H. Relations of metalliferous lode systems to igneous intrusives. *Trans.* A.I.M.E., vol. 74, 1926, pp. 29–70.

([6]) Newhouse, W. H. Some relations of ore deposits to folded rocks. *Trans.* A.I.M.E., vol. 96, 1931, p. 225.

([7]) Mackay, R. A., *loc. cit.*, p. 24.

AUTHOR'S REPLY TO DISCUSSION*

Mr. Brian Llewellyn: Mr. Robert Annan has raised the points of the increased cost of working and the yield of the ore expected to be developed. In regard to the first, statistics would appear to show that the trend of tin prices has been steadily upward since the beginning of the century, in spite of low-cost alluvial production and without taking into account exceptional periods of scarcity during war years. It seems likely, therefore, that rising metal prices will keep pace with increase in working costs. As to yield: I should expect this to improve when operations are limited to the more favourable depth zones, although

*Bull. 479, July, 1946.

I do not expect any future average yield in depth to equal the high yields (up to 40 lb. black tin per ton) which were got from the Dolcoath Main lode and other lodes in close proximity to the granite outcrop.

The lateral extent to which tin ore will persist in the granite would seem to depend upon structural control—by faulting and cross-fissuring of the ore feeder channels—and can only be determined by underground exploration. Information on this point is quite inadequate and my comment is that in the past it would have to be a rich prize indeed to compensate for the difficulty and expense of hauling and pumping through deep inclined shafts, frequently crooked.

Mr. Thomas Pryor is correct in his statement that 'Old Tom's lode', or the easterly extension of Reeve's lode, has been unproductive of tin in the East Pool mine. It has also been generally of narrow width and is believed by the author here to occupy a fissure partly sealed (by torsional movement ?) between two feeder channels in the North Crofty and Barncoose mines. It is significant that this lode is commonly loaded with fluorspar and it quite possibly contains tin at greater depth. It is perhaps unfortunate that the past production from the workings on this caunter lode was included in the tables, when computing the potential tin ore awaiting exploration. If, however, this production is omitted, the picture drawn is only slightly faded and the discovery in depth of a few important orebodies would restore the shade !

It is true that the Dolcoath Main lode, the Engine lode of East Pool, and the Great Flat lode were all south dippers and were more productive than any of the north-dipping lodes, but is the reason not to be sought in their flanking positions relative to the granite boss ? I agree that the rule of the stronger character of the north-dipping lodes now has many exceptions in deeper development.

The author hopes to be forgiven for omitting to include additional plans at shallower depths and supplementary cross sections ; his excuse is that he did so for the sake of clarity. The faulting of the various north dippers by lodes dipping south is somewhat confusing, especially as the sections would have to be drawn to a small scale. In the author's view the plan presented (Plate II) was sufficiently complex and this was reduced from a plan originally drawn to twice the scale. I am sure that all members will agree that the construction of the new mine model at South Crofty is a progressive step, and there is no doubt that this will facilitate an understanding of the geological structure of the lodes.

Mr. H. G. Dines contends that the slope of the tin zone is much flatter than the slope of the granite-killas contact. With special reference to the cross-section through Dolcoath and Roskear (Fig. 7), I should like to ask him whether he has sufficient proof that the cause of the flattening of the top of the tin zone is solely the settling down of the isotherms, as propounded by him, and not the close proximity to the elvans of the only lodes explored in depth. Attention was drawn to this influence under the heading of 'Productiveness of Lodes' on page 8 of the paper. Is it not speculative to attempt to delineate precisely any upper limit to the tin zone with the scanty information available of the northern part of the area? It is also worthy of note that tin ore occurred in the upper parts of the lodes at the western end of North Roskear mine only a short distance from the line of the section. I agree with Mr. Dines's general assumption of a somewhat flatter slope of the tin zone than the granite surface, but I also think that from a broad viewpoint and with due allowance for a probable upward roll of the granite at greater depth the two slopes will eventually prove to be substantially parallel.

In referring to the greenstone having been penetrated in South Roskear Mr. Dines has mis-read my paper, in which, on page 11, I drew a distinction between the northern part of the area and those to the N.W. and N.E.

Mr. Dines does not make himself quite clear as to the exact *loci* of the emanative centres and I should like to ask him if he considers that these are in any way connected with the junctions of lode systems. My thanks are due to this speaker for producing the cross-sections, which provide a valuable supplement to the discussion.

I wish to express my thanks to Mr. Gilbert McPherson for pointing out a printer's error—'granite-killas contact' was intended. I am pleased that he referred to the lenticular nature of tin ore-shoots, which I had omitted to emphasize, for it is important to appreciate the influence of undulations on the width of lode fissures.

Mr. J. H. Trounson criticizes the author's correlation of lodes presented in Table I on the ground that the different lodes are not of distinct appearance. In the past claims were frequently made that a particular lode could be readily distinguished and this might well be the case in a particular locality, where a lost lode was recognized by such features as width, dip, and mineral composition. But in tin lodes where the composition is so largely a matter of the depth zone and may change rapidly in quite a short distance, while the width is governed by structural peculiarities, it would indeed be remarkable if recognition by such means were possible over a considerable length of strike, *unless the strike were parallel to the granite-killas contact and there were little or no faulting.* Such conditions are rarely met with and the author prefers to rely rather upon determination of position and course of a particular fissure, being of the opinion that a fissure will maintain its general direction in an inelastic rock until it is disturbed by faulting or folding, in which case that direction will be subjected to local alteration. The author agrees with Mr. Trounson that there are big possibilities in Zone B, but he did not emphasize the south-dipping lodes of the Dolcoath-Carn Brea series or the Entral lodes on account of their relative inaccessibility at present.

No mention was made of the 'Complex' lode in South Crofty mine, as the author considered it to be an example of the kind of phenomenal mineral association which upsets preconceived theory, adds zest to the exploration of tin deposits in depth, and demonstrates the importance of the close study of structural control. The subject is beyond the scope of the paper.

Dr. W. R. Jones's far-sighted contribution is especially welcome at the present time, when the tendency is generally towards a short-term policy with a disregard of the future.

The author also wishes to thank Mr. J. C. Allan for his shrewd observations, with which he is in general agreement, and to add that the figure of 60 per cent tin extraction was assumed by him in his calculations whenever it was not possible to deduce this precisely.

In reply to the President's remarks on pumping the author agrees that more might have been said on this subject, but would excuse himself on the ground that the main object of his paper was to outline the ore-finding possibilities. The question of adequate pumping is, of course, of paramount importance when planning any increase in the scale of mining operations in an area containing so many abandoned mines and it is becoming increasingly evident that a survey of all old adit drainage systems is vitally necessary. Mr. C. V. Paull has now covered most of this ground and presented some very interesting figures, having gone very thoroughly into the factor of the saving of manpower by the changeover to electrical pumping. The author would add that the present prohibitive cost of coal has now introduced another and a dominating factor in favour of electrical pumping—that is, the economy to be obtained by the centralization of electric power, with its attendant reduction in coal transport costs, which appears to outweigh the high efficiency of the Cornish pumping engine.

As regards diamond drilling, Cornish tin lodes may be both friable and fissured, particularly those with a chloritic matrix, and the author has observed many instances where it would be impossible to obtain either core or sludge recovery which would provide any reliable indication of lode values.

In reply to Mr. Maurice Gregory : Evidence of lode characteristics in depth is naturally lacking, for deep mining was mainly confined to the strongest lodes on the immediate flanks of the granite bosses —for economic reasons. With setts usually limited in extent it would rarely have been profitable to work one or two lodes only in depth in an outlying area. Courage and big capital are now required to do this exploration in depth on the larger scale necessary for economic working. Such an undertaking, if commenced from surface in almost unknown country, would most certainly be speculative. If, on the other hand, it is approached systematically from existing workings at moderate depth in the manner briefly outlined by the author, the requisite caution is provided. To the old Cornish miner's saying ' Where it is, there it is ! ' may be added another : ' If you don't look for it, you won't find it ! '

I find it difficult to reply as precisely as I would wish to the question put by Mr. J. B. Richardson, for any forecast of future mining and milling costs must depend upon factors largely unknown at present—such as labour, power, and machinery costs, apart from the most economical rate of production. The latter can only be arrived at after further exploration has given some indication of grades and tonnages of ore to be blocked out. It seems reasonable to expect that working costs can be reduced to 35s. 0d. per ton on a medium tonnage of, say, 600 tons a day, with mining costs 17s. 0d. per ton, development 5s. 0d., and ore treatment 10s. 0d. per ton, and it should be possible by selective mining to maintain a grade of 25 lb. black tin per ton. This grade should, in the author's opinion, be the target.

Mr. R. A. Mackay's contribution is full of interest, but I would point out that it was not one of my objects to deal with the origin of cassiterite deposits, a controversial subject which itself would require more pages of the *Bulletin* than the whole of my paper. I agree with him that some authorities consider that many granites were formed from pre-existing rocks by invading solutions and not by consolidation from magma, but he will also agree with me that many authorities do not accept the theory of granitization. Even the most ardent supporters of the last-named theory have not, however, suggested that cassiterite-bearing veins of quartz, quartz-porphyry, aplites, and pegmatites, which traverse rocks of different types (even limestones in some countries), have been formed by changes in the rocks through which they traverse. I agree that structure is a most important factor in the control of mineralization and that is why I stressed in the paper the importance of the careful location of the position of the granite contact in relation to the killas. With regard to Mr. Mackay's statement, with which I agree, that the position of the lode relative to the top of the granite is of importance, I would add that it is a matter to which I gave great attention in coming to my conclusions. Mr. Mackay has recently advanced an interesting theory concerning ore deposition, but it is a subject beyond the scope of the present paper.

Several speakers have criticized the author's rough estimate of £50,000,000 as the potential value of tin ore awaiting development. It is, of course, quite impossible to attempt an accurate forecast of the value without a convenient yardstick and the author would point out that this figure was presented solely with the object of drawing a mental picture of the possibilities in the light of past experience. It is admitted that the tendency in the past was to mine more selectively to a higher grade and this makes comparison all the more difficult. At the same time the figure given is not so fantastic as may at first sight appear. Five or six tin lodes one fathom in width, of a 25-lb. grade, and worked to a depth of 200 fathoms over the length of the area with a 50 per cent payability ratio would be worth this amount of money at less than the present tin metal price. I submit that the chance of such a discovery is a very reasonable mining risk.

In conclusion I wish to express my thanks to all those who have taken part in the discussion of the paper and have added to its value by their suggestions and contributions.

DISCUSSION IN JOHANNESBURG

ON

Crushing and Grinding Efficiencies.*

By T. K. Prentice, *Member.*

(Note.—*This Paper was also presented at a General Meeting of the Chemical, Metallurgical and Mining Society of South Africa held on 27th February,* 1946, *and the following is an abridged report of the discussion at that Meeting.*)

Dr. J. T. McIntyre (President of the Society) said the new ground broken by Mr. Prentice in his stimulating paper had, in all senses of the word, been very extensive, and he was sure they had all profited by listening to his lucid description of these most interesting experiments. There was a wealth of data in his paper which would be found of great assistance to future research.

Mr. Prentice had been very careful to give all the detailed data from which his conclusions were drawn so that others might in the light of their experience amplify or suggest other interpretations on the figures stated. His own experience in comminution of rock experiments had been that the results obtained could be widely variable. They had found as a tentative result in rock drilling that the average size of the sludge particles decreased (which meant the surface area increased) with increased drilling speed. This implied that the apparent efficiency ratio of rock comminution from slow to fast drilling was changing by a matter of many times over. In drilling as distinct from milling, however, it was not their wish to produce a lot of fines, but rather to punch a hole through the solid rock *in situ* with the minimum amount of energy.

He was also reminded of the Izod impact test on steels, which could give very different results for the foot-pounds of work required to fracture the same type of steel depending upon its previous history, particularly as regards heat treatment. Was it a far cry to suggest that they gave the reef some form of heat treatment before attempting to crush it to the desired fineness for extraction of the gold ?

Mr. Andrew King said the author submitted the results of a large amount of detailed work in a clear and concise form ; the

*Bull. 477, March, 1946.

paper was one which would receive much widespread attention, not only in the metallurgical world, but in other industries where fine grinding was an important operation.

It was of importance to note that the efficiency figures for crushing and grinding units which the author had arrived at, using Rittinger's law, were much nearer what would be expected than the corresponding figures using Kick's law. Gross, who carried out a thorough investigation on behalf of the United States Bureau of Mines and other Institutions, in summarizing recent work on sub-sieve sizes had made the following statement : ' The work done by the Bureau of Mines at Salt Lake City and by Martin, in which surface determinations of the crushed materials confirmed the Rittinger law, would seem to be final '.*

General statements had frequently been made that in grinding to a certain sieve size a large amount of over-grinding took place with consequent waste of power, etc. As the author had pointed out these statements generally applied to flotation plants, where a slime-free product was usually preferred and where over-grinding frequently resulted in decreased mineral recovery. So far as the Witwatersrand was concerned, however, it had to be remembered that in modern plants the aim usually was to grind a large proportion of the pyritic portion of the ore to *minus* 325-mesh and comparisons on one mine seemed to show that, where classification permitted of this being done, some appreciable improvement in residue value could be obtained.

An investigation of the number and size of fine particles produced indicated, however, that a very large proportion of the new surface area was in the *minus* 20-micron size and it would appear, therefore, that much useless work and over-grinding was being done ; from that point of view, further shortening of mills and more classification seemed to be indicated, and consequently it would be necessary to satisfy themselves that, in their modern plants, they had arrived at a suitable compromise.

In connection with the percentage of total power input which was available for grinding in large cylindrical mills the observations shown in Table XXIV were made at the Venterspost reduction plant with the assistance of the electrical engineering staff.

The percentages of full power lost in driving the empty mills, although somewhat lower, were in general agreement with those given in Table XXI of the paper. It would appear, however, that additional allowance must be made for losses in bearing friction due to the weight of the grinding load and pulp ; in other

*Gross, J. United States Bureau of Mines *Bulletin* 402 (1938).

words, the power required to revolve the ' dead load ' of the mill was not consumed in grinding. To some extent this appeared to have been the view of Gross and others who carried out work from the United States Bureau of Mines.*

TABLE XXIV

	kW	*h.p.*	*Per cent of full load*	*Per cent of full load available*
Primary ball-mill (9 ft. diameter by 10 ft. long), equipped with double helical spur and pinion, reduction gear, roller bearings on pinion shaft, and oil-lubricated trunnions.				
Full load..	298	399	100	—
Motor only..	7·8	10·5	2·63	—
Motor + Reduction gear	10·5	14·1	3·60	—
Motor { + Reduction gear + Empty mill with liners	19·2	25·7	6·45	93·55
Secondary tube-mill (8 ft. diameter by 16 ft. long), equipped as above :—				
Full load..	173	232	100	—
Motor only..	6·6	8·8	3·80	—
Motor + Reduction gear	8·1	10·8	4·66	—
Motor { + Reduction gear + Empty mill with liners	15·0	20·1	8·67	91·33

Returning to the Venterspost mills ; it was estimated from calculations of average load that the additional power so consumed was 2·8 per cent, or 11 h.p., for the ball-mill and 2 per cent, or 4·5 h.p., for the tube-mill. The percentages of power available for grinding then became 90·8 and 89·3 respectively, and calculations of efficiencies would give slightly higher results on account of the lower percentages of power available.

Regarding grinding tests and calculations of new surface areas carried out for one ball-mill or tube-mill in a large plant, the difficulties encountered in making tonnage determinations and the

*Ibid, and *Trans. A.I.M.E.*, Vol. 112 (1934), ' Milling Methods ', p. 25.

uncertainty of obtaining reliable samples tended to throw a little doubt on the results calculated from the figures obtained; the thought had occurred to him, therefore, that where grinding units were operated separately, those tests could possibly be carried out more accurately by working on new feed tonnages and classifier overflow grading analysis and he hoped to investigate this possibility.

Mr. F. Wartenweiler said Mr. Prentice had answered the challenge of low mechanical efficiency in their crushing and grinding operations and deserved their gratitude. He had checked the findings and theories of other investigators in the theoretical field and had formed his own by careful laboratory research and practical determinations which disclosed original thought and were of great value. He had given them a definite basis and methods on which they could measure their milling efficiencies in more concise terms than they had had heretofore.

It was noted, not without some satisfaction, that the efficiency of the main working units of milling, given at 32·8 per cent for the gyratory crusher, 34·7 per cent for stamps, and 23·4 per cent for cylindrical mills, was an apportioning of the work done by the modern milling plant on a fairly well-distributed basis.

On observing the action of a crusher it was apparent that a great deal of work or power input which was in the form of compression was used to overcome elasticity, particularly with a schistose rock and shale, and to overcome cohesion and binding power of crystal interfaces with a crystalline rock—such as banket and quartzite. One was impressed with this visible evidence and inclined to think that the crusher performed certain work which was not measurable because actual fracture might not take place until the succeeding stage and that therefore it did not receive full credit.

It was pleasing to note how well the Californian stamp battery showed up. Agricola would feel that his fine effort to publicize the stamping machine was well repaid. This awkward and seemingly unmechanical contrivance was given high place.

The cylindrical mill—under which came the pebble-mill and the ball-mill—had the more difficult task, inasmuch as it dealt with the later and finer stages of communition and had an action less direct and perhaps more of the hit or miss order than the other machines. In studying Table III its task seemed nearer that of the chain reaction of atom smashing.

Mr. Prentice had stressed that bending force was more effective an agent in communition than compression, per unit of force applied. This advantage would, no doubt, be studied and followed up. The heavy loss of power by transference into heat was awkward and left a large field for investigation and invention. In sub-arctic winter climates, milling plants utilized part of this heat for maintaining tolerable working conditions in the buildings. One would like to apply it in assisting a useful concurrent chemical reaction.

Dr. O. A. E. Jackson said much information had been published on the energy required for the comminution of rocks, which was an important and frequently expensive branch of the science of metallurgy. Mr. Prentice had, in his paper, added new facts to increase their knowledge of this subject. He had drawn attention to the fact that, no matter what the energy required might be to effect rupture *per se*, under their present methods and using the various machines at their disposal, it was unavoidable to expend, before such a rupture occurred, far greater proportions of energy in preparation for the same. It was, therefore, the sum of those energies which must be considered from the practical veiwpoint as a basis for computing the efficiency of comminution.

The factors involved in crushing rock were more complicated than they might appear to be superficially and a knowledge of fundamental crystal structure was necessary ; so that the problem invaded the province of the physicist. Mr. Prentice had mentioned the energy required to overcome the elasticity of the rock ; there was also the matter of plastic deformation which must be considered and which, in many cases, was of considerable magnitude.

Mr. J. Ellis said a picture of the atomic structure of minerals seemed to be necessary if their behaviour under stress was to be understood. What followed was an attempt to indicate a connection between Mr. Prentice's results and the atomic structure of the material he considered—namely, quartzite. It supported his results, and, perhaps, made them more comprehensible than under the simple conceptions of the strength of materials based upon the first, and necessary, assumption of elastic strain in an isotropic medium. Actually the strain was not elastic and the medium not isotropic. Further, this method suggested some limitations to Mr. Prentice's conclusions and indicated that quartz and quartzite might be ideal materials.

The method was descriptive and not precise—a severe limitation, which must be recognized. If precision were attempted, this contribution would be a maze of complicated mathematics, would probably lose much of its value and, moreover, the results would still remain qualitative, because the physical properties of almost all mineral crystals and of all rocks were too complicated for such calculations to be made. An explanation of the observed behaviour of rocks and their constituent minerals was all that could be attempted.

The problem of the breaking of minerals interested the geologist studying the deformation of rocks, the physicist studying the physical behaviour of matter, the mining engineer studying his problems of breaking rock and of rockbursts, as well as the metallurgist, although each might view it in a different aspect. Thus, the geologist considered fracture and plastic flow under conditions of high-confining stress and long periods of slow deformation, whereas the metallurgist was confronted with no confining stress and instantaneous fracture. Yet, failure was the object of the researches of both, and so the problem was essentially the same for both, although the similarity might not be appreciated until the atomic structure of minerals was considered.

Looking at the problem from a purely theoretical viewpoint, Rittinger's hypothesis—namely, the amount of work that must be done to break a rock was proportional to the increase in area of the broken particles—was fundamentally correct. Considering only crystalline solids, what were the forces holding the particles together—the bonds which must be broken to fracture the material? Before that could be answered, one must discover what the particles were. They were ions—that is, the atoms of the constituent elements, each carrying an electrostatic charge arising from a characteristic excess or deficiency of electrons. For example, in common salt, the sodium ion might be considered to have lost the single electron it had in the outer, or M electronic shell, and so had an overall positive charge, equal in magnitude and opposite in sign to that of the electron which it had lost. The chlorine ion was a chlorine atom, which lacked one electron to complete the outer, or M electronic shell, now possessing the extra electron, so imparting an overall negative charge on the ion equal in magnitude and of the same sign as that of the electron. The sodium and chlorine ions thus carried opposite charges, and were attracted to each other by a force proportional to the magnitude of the product of their charges and inversely proportional to the distance separating them.

A crystal was then an arrangement—in three dimensions—of the ions, in which the energy of the system was a minimum under the conditions of formation.

An ionic crystal was a three-dimensional arrangement of ions, bound together by the forces described. Although the silicates and most rock-forming minerals were not simple ionic crystals, the description applied to them.

When a crystal was broken all the bonds binding the ions on the break were broken, and the work which must be done would, for a given crystal, be proportional to the number of ions on the break, which was proportional to the area of the plane of fracture. The energy required was proportional to the area of the surface of fracture.

In actual practice the amount of energy required to separate the ions on a plane was less than calculated, owing, it was thought, to imperfections in the crystal lattice.

The work required to produce one square foot of fracture surface in quartz had been calculated to be 0·124 ft. lb. per square foot, which was based on the assumption that the force to be overcome was only that between the oxygen ion and its nearest silicon neighbour. It should, however, be the sum of the forces between the one ion and all the others in the part of the crystal from which it was being separated. Therefore the force required for the work necessary to produce fracture might be greater than the calculated amount.

It was based upon a further assumption that the crystal was broken by a simple tensile force. But was a crystal in fact broken in that way? Could it be broken by an internal tensile force?

Although a tensile force could be exerted upon the ions at the bounding surface of a crystal—as, for example, by immersing crystals of salt in water, it could not be conveniently applied in the solid. Compressive, shearing, and bending forces were the ones commonly applied to break solids.

The actual relative displacement of the ions in a crystal must conform to three types—compression, tension, and shear. Fracture occurred only when the external forces, which in crushing were compressive, were so resolved by the crystal that internal tension of sufficient magnitude arose.

How were the external forces resolved by the crystal into stresses which produced fracture? In the compression tests the specimens had broken on shear fractures, indicating that there had been a separation on shear planes, although the compression across such planes could not have been the minimum. Those test pieces were polycrystalline specimens and so the distribution of the internal stresses was not so simple as that in a single crystal.

It appeared that quartz was a mineral which would give results closest to the ideal proposed by Rittinger, because it was composed of only two kinds of ions, silicon and oxygen, arranged in a manner such that there were no planes of weakness—quartz had no cleavage; the distribution of atoms on the average breaking surface was essentially constant, all the ions were bonded together by the same forces, and finally, because it was a mineral which did not contain impurities. Further, it appeared that the estimated amount of work required to produce fracture was low because it did not take into account two factors :

(1) The summation of the force between a given ion and all the others, and

(2) the method by which the fracture was probably produced.

During elastic deformation of a substance the amount of work done was proportional to the volume or the weight of the test piece. It must be emphasized that this applied only to the elastic deformation and that when considering breakage they were considering non-elastic deformation. Assuming that the work done in non-elastic deformation was proportional to the area of the break produced, then the total work done to break a rock should be the sum of two factors—the work done during elastic deformation (y), which was proportional to the volume of the weight of the specimen, and that done in non-elastic deformation (x), which was assumed to be proportional to the area of fracture surface.

So long as y was small relative to x, then Mr. Prentice's figures would support Rittinger's hypothesis, because the figures showed sufficient variation to hide the value of y.

But y varied with the type of structure. If the elastic work which could be done on a test piece in compression be taken as y, it was $y/3$ for a simply-supported beam and $y/9$ for a cantilever. If the average value of x *plus* y' for centre bending was 3·17 and for end bending 3·87, and for compression 3·46 ft. lb. per square foot, then,

$$x \text{ plus } y' = x + \frac{y}{3} = 3{\cdot}17 \quad \ldots\ldots \quad (1)$$

$$x \text{ plus } y'' = x + \frac{y}{9} = 3{\cdot}87 \quad \ldots\ldots \quad (2)$$

$$x \text{ plus } y = 3{\cdot}46 \quad \ldots\ldots \quad (3)$$

If y was solved between (1) and (2) the result was negative, which was wrong, and was presumably due to the inherent error in the figures and the relative magnitudes of x and y.

Solving y between (1) and (3) : $y = 0{\cdot}4$ approximately and $x = 3{\cdot}0$, or when $y = 1$, $x = 7{\cdot}5$ approximately.

Little reliance could be placed upon this figure. The ratio y/x might lie between 0 and, say, 8. In other words, the conclusion reached from these figures was that all the elastic work was eventually expended in breaking the rock, or that the elastic work which was not so expended was negligibly small.

If the strain was elastic only within the proportional limit, then the energy of elastic strain appeared to be negligibly small. Mr. Prentice's figures were in accord with this.

The lack of proportionality between compression and expansion of individual atomic bonds might explain the fact that most of the non-elastic (plastic) deformation was concentrated on planes regularly spaced through the crystal about one μ apart. Most of the energy of strain was concentrated in those few planes, along which it was presumed that the break would appear. It would be seen that when the strain on this plane had reached the amount indicated, the energy of elastic strain in the crystal would be available to do some of the additional work required to produce failure. And so, although the work done in elastic strain might be lower than Mr. Prentice suggested, it was, nevertheless, available to do work in breaking the crystal.

As no crystal was isotropic to stress, the stresses in a polycrystalline aggregate would vary rapidly in magnitude and direction from one crystal grain to another. Thus, the actual effective internal stress, which was generally a shearing stress, reached the intensity sufficient to produce fracture in only a few planes in a polycrystalline aggregate.

It was interesting to note that when, as for example in the compression test, the work done on the specimen was high, the surface of break was also great ; yet, when the same test piece—say, diamond drill core—was broken by a hammer blow, the small amount of work was concentrated on a single, and therefore small, plane of fracture.

Materials which readily flowed plastically—could shear indefinitely without rupture—might absorb a great amount without rupture. For such materials the amount of work required to produce fracture might be high. Single crystals of tin had been deformed to four times their original length, and even some ionic crystals to twice their original length without fracture. In such cases Rittinger's hypothesis might not be supported so well as by quartzite.

Whether a crystal would flow plastically depended upon the rate of application of the stress, the temperature of the specimen relative to its melting point, and the confining pressures.

One might expect a certain minimum stress to be necessary before any plastic flow was produced in a single crystal, though it was doubtful whether this was so if the rate of application of the stress was very slow, especially for metallic crystals, and more so if the temperature was close to that of the melting point of the crystal. Thus, ice, in glaciers, was close to that of the melting point of the crystal, and ice flowed plastically under very small stresses. The same applied to lead at temperature 200° and 300° Centigrade below its melting point. However, although these factors were important to the geologist in the solution of problems of metamorphism, they were probably insignificant in the metallurgist's problem of grinding and crushing rocks, although they might partly account for the difference in work required to crush and to grind the same material. The significance of the difference in the rate of application of the stress could be appreciated from the behaviour of glass, which was plastic towards a stress increasing slowly, but brittle to a sudden stress.

An estimate of the energy required to crush micaceous minerals was complicated. In fairly coarse crushing the energy was used: (1) To break the crystals on their cleavage planes, and (2) to bend the crystals. The first was proportional to the area of the break, and the second was probably not. In finer crushing or grinding the energy was used: (1) To continue to break the crystal on the cleavage planes, (2) to break the plates, and (3) to bend the plates. (1) and (2) were proportional to the area of the break, but the proportionality was different, and (3) might not be proportional to the area of the broken surface.

The amphiboles were a group of minerals characterized by having the SiO_4 tetrahedra arranged in chains of unlimited length, held together by cations. The pyroxenes had their SiO_4 tetrahedra arranged in strings, held together by cations. Minerals of these two groups might behave in a way similar to the micaceous ones, although the effects might not be so marked.

But rocks were polycrystalline material, and might therefore break in two ways—around the crystal grains, and through them. In the first case, if the intercrystalline material was plastic, or certain of the crystalline constituents were plastic—for example, micaceous—then the simple relationship for the work required would not hold, and the amount of work be great. Quartzite consisted almost entirely of quartz, and the crystalline grains formed a mosaic, so that it was virtually impossible to break the rock around the grains. It then appeared that quartzite was an ideal rock consisting of ideal crystals. When coarse shale was crushed, the fracture would probably take place between the grains and the amount of work to be done be small. When shale was to be crushed finer, and fracture must be, in part, through the crystal grains, it might well be in planes of weakness—for example, the cleavage of the micas—leaving stronger crystals and other planes unbroken. Finally, if it was to be ground very fine, not only must the stronger crystals be broken, but such crystals as micas must be broken along planes which required far more work to be done than was required for fracture along the cleavage planes.

By publishing his results, Mr. Prentice had made a valuable contribution to the knowledge of the subject of breaking rocks, but it appeared to the speaker that quartzite, which was the rock studied, was an ideal rock, and that his principle might be modified for other rocks.

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SECOND ORDINARY GENERAL MEETING

OF THE

FIFTY-SIXTH SESSION

Held in the rooms of the Geological Society, Burlington House, Piccadilly, London, W.1,

ON

Thursday, October 17th, 1946.

Mr. G. F. LAYCOCK, *President*, in the Chair.

DISCUSSION

ON

Anglo-American Magnesium Production,

By P. L. TEED, *Member*.

The President said that there were two important and interesting papers to be submitted for discussion. The first paper—Anglo-American Magnesium Production, by Major P. L. Teed—was one for which members, he was sure, had been waiting for rather a long time, but earlier publication was not possible owing to the present restricted paper supplies. The author was fortunate ... able to be present to introduce his paper.

Major P. L. Teed said that during the war period he acted both as Technical Adviser to the Ministry of Aircraft Pr... tion and as Non-Ferrous Metallurgical ... Commonwealth Sci... in the ... dual capacity he ... the schemes on ...

...erested ...iple of ...itions ...eri- ...h a ...nd ...d

...h ...ar. ...uld ...nter- ...sen to ...agers— ...s, carried ...ngineering

... to which the ... from sea-water ... respectively— ... ourcefulness of censorship. That explained a number of omissions and, in particular, why there was no mention of any individual or any company and why the paper was without a bibliography.

Since it was printed he had received a number of letters with regard to it. One correspondent made the slightly unjust complaint in that the writer ought to have referred to a particular development but his reply was that the paper was intended to cover the war period, while the development in question had taken place subsequent to the conclusion of hostilities. Secondly, he had been reproached for not giving details of the concentration of magnesite by froth flotation. Since most of the great papers on froth flotation had been delivered before the Institution he thought that the development in question merited a paper in itself and, therefore, he asked to be excused for giving only a few lines to it in what was intended to be a general survey.

Professor Leslie Aitchison* said, that on reading through the paper one appreciated that magnesium was an extraordinarily interesting metal. Its lightness, combined with its strength, made it potentially very valuable for various engineering industries, particularly those concerned with the transport and movement of materials from one place to another and especially for reciprocating parts, etc. Although magnesium had proved itself to be most valuable in war-time for incendiary bombs and in similar directions involving the burning of the metal, he thought it had not fully satisfied expectations for peace-time applications. In his opinion, there were two main reasons for that. The first was that magnesium and its alloys corroded readily and the second was that magnesium was a material which did not easily allow itself to be formed, worked, or shaped in the cold. It was almost impossible to draw, press, or spin it until the metal had been heated to a moderate temperature. The latter difficulty could probably be accounted for in terms of the crystalline structure of the metal, this being the close-packed, hexagonal form. It was a general experience that most of the metals having a close-packed hexagonal structure did not readily permit of working at ordinary temperatures.

The other phenomenon—the considerable liability of the metal and its alloys to corrode—he thought was associated in a great measure with the extent to which they contained certain impurities. He thought one of the great values of the paper was its demonstration of the high degree of purity to which magnesium from the extraction processes had been brought, both in this country and in America, particularly in respect of iron.

*Professor of Industrial Metallurgy, Birmingham University.

He wondered whether Major Teed would help those interested by giving still more data in respect of a quite unique example of large-scale metallurgical development. They hoped conditions would never occur again which would justify that kind of experiment; it was only under the extreme stress of war that such a very large sum of money would be spent on such production and consequently all possible lessons should be learnt from what had been done. He would, therefore, like to ask Major Teed for a little more information about the economics of the processes covered. He feared Major Teed was a littly chary of giving precise figures, no doubt as a result of the censorship which the paper had received, but thought that if he could provide the relative costs of production of magnesium by the different processes described in the paper such information would be of real interest to the members of the Institution and justify him, perhaps, in asking for it. In his opinion, the paper was a splendid record of a magnificent achievement on the part of the Allies during the last war.

Colonel W. C. Devereux* said he could not think of anyone more highly qualified than Major Teed to talk on the subject covered by the paper. It had been his pleasure to see, both here and in the U.S.A., a little of the first-rate liaison work to which Major Teed applied himself during the war with such complete devotion, energy, and scientific skill. With typical modesty he had refrained from mentioning his own contribution to the developments he had described, but he thought it no exaggeration to say that his efforts in co-ordinating Anglo-American research and development in magnesium production and utilisation had played a significant part in fulfilling the production plans and in achieving that remarkable expansion in the supply of magnesium which was one of the most remarkable industrial achievements of the war.

It was particularly gratifying to him that Major Teed should be the first to present a comprehensive review of that great international co-operative effort, particularly since he had chosen to dedicate it to the men—the scientists, engineers, and managers—who, working against time under very abnormal conditions, carried out the great technical advances and the tremendous engineering achievements he had described.

Having been responsible for two of the plants to which the author had referred—for the production of magnesia from sea-water and for the thermal production of magnesium respectively—he could vouch for the tremendous energy and resourcefulness of

*Managing Director, Almin, Ltd.

those engaged in the work in those desperate years which now seemed so far away. Thinking of them one could not help making comparison with the present time, and wishing that some of the enthusiasm then engendered by a common cause and a clear objective were more in evidence today, when the problems before them, although of a somewhat different nature, were just as great as they ever were.

Three main points appeared to emerge from Major Teed's survey: First, the possibility of considerable economies in the electrolytic process, both in the manufacture of the magnesium chloride and by improved cell design; secondly, the successful establishment of the ferro-silicon reduction process as a means of producing high-purity magnesium on a commercial scale without a separate refining operation; and, thirdly, the demonstration of the practicability of carbon reduction, which now remained a challenge to engineers to implement the success of the metallurgists.

Now that war requirements were over, what return were those in England to get for their share of the Allied investment of some £110,000,000 in magnesium expansion? In the United States producers had hurriedly revised their peace-time schedules for a demand they had not foreseen and were reopening plants that had been planned to close at the cessation of hositilities. In England, however, total fabrication of magnesium products was at present less than half that of 1939 and only one-tenth of the production potential. It looked as though they were writing off a possible asset as a total loss. One reason might, perhaps, be indicated in the Appendix, Plate I, where comparison of the impurity elements showed a wide divergence, not only in the allowable quantities, but also in the elements considered important enough to specify. Hanawalt, of the Dow company, in his classic paper on the effect of impurities on corrosion resistance, demonstrated that if certain elements were kept below their tolerance limits, magnesium and its alloys had as good resistance to sea-water and spray attack as the corrosion-resistant aluminium alloys. There was no doubt that American producers, regardless of their methods, were facing up to this as a means of creating a widening demand for a new stable material. No fabricator in this country would feel happy to specify domestically-produced magnesium for any application requiring corrosion resistance without advancing such precautions and protective treatments that would frighten the prospective customer to some less temperamental material.

The second reason was undoubtedly one of production cost. Apart from plant efficiency, they came back to the almost thread-bare argument of power cost, in which magnesium was quite a junior appellant. There were not many places in the world where hydro-electric power was of lower cost than that from the best thermal units. In this country coal should not be considered as a potential export until improved mining methods could satisfy the demands for cheaper power. The Institution of Mining and Metallurgy was a body well fitted to consider whether that defection in coal production should continue to prevent a sizeable fabricating industry from access to a not inconsiderable home and export market. Production of magnesium in Great Britain could hardly be justified if it was only rendered possible by restriction of imports, or as a subsidized industry, neither of which offered any incentive to produce the best quality metal. The alternative, of course, was to follow the precedent of aluminium and nickel, and to allow competitive imports from Canada, where the ferro-silicon process was operated efficiently and economically.

Mr. R. D. Hume* stated that as he had severed his connection with magnesium production nearly three years before he was not in a position to give details of the process with which he was closely associated during the war. There were, however, one or two practical points which he learned in making the process a commercial proposition and he would like to pass them on in case they would be useful to members who might meet similar difficulties.

One of the major problems in connection with the process was to provide a metal retort which would stand a temperature of 1120°C. at a high vacuum and although various grades of alloy steel were investigated they were extremely costly, and not obtainable in sufficient quantities for the output that was required during the demands of the war period. It was eventually found that a sheath of a thickness of 3/32 in. containing approximately 25 per cent nickel and 22 per cent chromium permanently fixed round the retort afforded a complete protection to a retort composed of ordinary mild steel and this method was used commercially in the plants operating the carbide process. An important point was that if a sheathed retort should be overheated, resulting in serious scaling of the sheath, a repair could be quickly and satisfactorily effected by welding a fresh portion of the sheet material in or over the badly-scaled area.

A similar problem was encountered in melting the magnesium metal, as produced in the retort prior to casting into ingots, insomuch as the furnaces employed were oil-fired and the steel pots scaled badly through flame impingement. Various reasons made it

*General Works Manager, Foundry Services, Ltd.

impracticable to modify the furnace design, but protection was afforded by means of a plumbago sheath. This sheath was designed to protect the base of the pot and extended slightly above the area impinged by the flame. It was 2 in. thick and was an easy fit over the bottom of the pot. The useful life of the pot was prolonged from about 200 hours to 750 hours and the period required to melt the charge increased by only 5 per cent.

With regard to magnesium fires: on one occasion more than a ton of magnesium swarf was well alight and no progress could be made with flux, sand, or asbestos blankets, etc. In their anxiety to prevent distortion of the steel framework of the building the works fire brigade played their hoses on the adjacent steelwork and platforms in order to keep them cool. One active member, however, took it upon himself to direct the water onto the main body of the fire. As there was no immediate deterioration in the position water was played on the fire in considerable quantities and only two minor explosions took place. It was not suggested that water was a safe remedy for dealing with magnesium fires; in most circumstances it should undoubtedly be avoided, but in that particular instance the actual combustion was hastened without causing any serious explosions, and damage to the framework of the building was prevented.

Dr. S. W. Smith said that Major Teed had, to use his own words, reviewed the great achievements of teams of technical men inspired with the necessity of producing magnesium with the very minimum of delay and without undue regard to cost. He was sure they would all agree that the achievements of those men had been of very great service to the Allied cause during the recent critical years. The over-all picture of magnesium production during the war was, as he said, one of a great task well done. Where so many had played their parts, it would, perhaps, have been invidious to mention individuals or companies by name. To those who were specially interested, however, bibliographies were available, dealing with production in this country and in various parts of Canada and of the United States. Dr. Cecil Desch had referred to these enterprises in the masterly review of magnesium production and fabrication which he delivered to the Royal Society of Arts some three years ago. Metallurgy had had many fascinating and romantic periods to its credit, but it was true to say that few, perhaps, had surpassed that of magnesium production in the speed with which developments had taken place in so many different directions.

Dr. W. F. Chubb* said he had been particularly interested in Major Teed's discussion of the processes for the production of magnesium by distillation, these methods having been established for many years past. The process was first developed on a commercial scale in England under the management of Mr. Hume and his purpose in speaking was to try and correct certain misapprehensions which had arisen. He was going to refer to magnesium production in America and more particularly in Canada, in which he had taken an active interest. In 1939 a Mission went out from England headed by Sir John Greenly and it was that body which was responsible for the war-time development which took place. During the first year of its existence, the question of magnesium production came to the fore and as Consulting Metallurgist to the Greenly Mission he had suggested that the authorities in Canada who were working on this problem should be given details of the established English process in order that production might commence as quickly as possible. Subsequently a description of the Canadian plant was published in several technical journals, notably in the *Compressed Air Magazine* of the United States. The description given dealt in part only with this Canadian development, but nevertheless gave sufficient detail to show one or two points of departure from the original English practice.

Dr. Chubb then described in detail the difference between the Canadian process and the original English method and concluded by saying that when he was in Canada the cost of production was estimated to be about 18 cents per lb., but he believed it was subsequently reduced. He could not speak of United States' costs of production, but he thought they would be approximately parallel.

Sir Lewis Fermor said he found himself most interested in the paper from quite a different point of view from that which had been discussed. Last year he was enquiring into the amount of manganese which occurred in aluminium, and the effect of that manganese upon the properties of the aluminium.† He noticed that the table at the end of Mr. Teed's paper gave the composition of various brands of magnesium, from which it appeared that manganese could not be regarded as an impurity. He was wondering whether magnesium without any manganese at all was technically valuable. He made that remark because in the table of British

*Consulting Metallurgist, formerly Professor of Metallurgy in Istanbul University.

† 'Manganese in Industry', *Proc. S. Wales Inst. Engrs.*, Vol. 61, No. 2, p. 42 (1945).

statistics manganese was invariably recorded with a maximum figure in the example, whereas in the American statistics a minimum percentage of manganese was given in every case. He would like to enquire what was the function of manganese in magnesium.

Mr. A. B. Lisle* said that the two or three speakers he had heard discussing the paper before the Meeting had talked on current political issues, future progress possibilities, fabrication matters, some manipulating hazards and other aspects not really related to the paper. Much more could be said on all of those questions, but he would confine his comments to the paper itself which seemed to him to comprise exclusively an historical survey of the recent past.

He said that the author was to be congratulated on a paper that was very comprehensive, at least in regard to magnesium production. He had been singularly successful in keeping the matter objective, free from personal references, and without readily-obvious allusion to any of the concerns that had worked in that field. It did, however, seem advisable that a number of corrections should be made, and elucidation of some doubtful points was also desirable.

On the matter of magnesium, the datum line in the mind of the author appeared, however, to be the delayed entry of the United States into the war at the end of 1941, and that mental attitude was doubtless responsible for some of the omissions, as well as for some statements that appeared to be not strictly accurate. Although the United States facilities were praised highly, there was no emphasis upon Canada's wonderful, if not superior, potential electrolytic capacity. Having personally, eighteen months earlier, spent some time in Canada on the magnesium production problem, under the orders of the British Government Department concerned, to whom he believed the author was one of the technical advisers, he could firmly assert that the United States performance, large as it was, made unnecessarily late contribution to the Allied cause. The errors at that time of the Home Government's advisers and politicians should not be glossed over, especially as they now had enduring consequences for the Empire in magnesium progress.

In the last paragraph of p. 3 the author had mixed together United States and British performances, and the general conclusions he had reached, therefore, were, in numerous instances, likely to prove misleading. On p. 4, second paragraph, the author said that new and untried processes 'had to be supported'. They were supported, but not of necessity. In Britain the decisions

*Director, F. A. Hughes & Co., Ltd.

occurred only following recommendations by poorly-informed consultants, who, unfortunately, often gave the impression that they regarded themselves as having a monopoly of patriotism, or honesty.

The statement on p. 7 regarding chlorine consumption was surprising. In actual fact, there seemed no indication that a saving in chlorine was consequent on the elimination of peat. It was sure that that economy followed upon closer operation control and was especially related to greatly improved purity of the basis material. On p. 9 and 11 the author gave, in considerable detail, an American electrolysis process. It seemed strange that the electrolytic process that mainly supplied the United Kingdom and largely the United States also, throughout the war, was treated as non-existent.

It was surprising that a statement was made on p. 10 of the report that 'boron, although reduced to the elemental state, tends to drop through the electrolyte into the sludge'. They knew from certain American publications the extraordinary damage to the metal output which most minute traces of boron would occasion and they also knew it to their cost from what occurred at an English shadow factory. The paper gave no hint of the relative economics of the various processes and he thought the author could, with advantage, tabulate the respective tonnage outputs in: (*a*) the United Kingdom, and (*b*) the United States, by each of the numerous processes. Such figures would certainly help to illustrate the real merits of each one.

With regard to the reference on p. 16 to the aluminium reduction method; if power consumption in that case was so much more favourable, what were the factors that made the product more than twice as costly as metal manufactured by either of the electrolysis processes?

After reading the paper, one was forced to wonder whether the author had never seen the House of Commons Select Committee's Report of February, 1944. That gave, as regarded the United Kingdom, the true practical picture of magnesium production, especially during the war and, he thought, clearly showed, among other things, why the aluminium reduction process was abandoned.

On reading p. 16 one was reminded that the author had for long been strongly infatuated with the trans-Atlantic silicon reduction achievement, but it should be remembered that the prior work in Germany and Britain, on which patents existed before 1939, anticipated these American efforts. Even the high vacuum which the author emphasized was used in that original work, and a considerable quantity of magnesium was made, so that the Canadian and American performance was in no sense new, although possibly it appeared so to new entrants in the field. The chrome-nickel retorts—impossible to obtain here during the war—ought to be emphasized.

The cost of installation and maintenance of the various plants was not indicated in the paper and for effective study of the subject knowledge of these figures was most valuable. At the conclusion of the paper the author gave an extensive description of carbon thermic reduction methods. One would ask what were the proved costs and what was the insignificant total output that had been achieved by those in either of the two countries under consideration. In the United Kingdom it was believed that the method had been finally abandoned. Some warning should be expressed in relation to the Appendix. The alloys and specifications were not capable of direct comparison between the United Kingdom and America for various reasons that would take far too long to explain.

Finally, he expressed the hope that in the final printing of the paper for the records of a famous Institution the author would see his way to arrange for amendments to be made, in the sense of some at least of those now suggested.

Major Teed, having thanked those who had referred to his paper in such laudatory terms, said he would like to reply to one point which had been raised. The four-fold censorship which he had mentioned ruled out any detailed disclosure as to costs of production. This, he considered, was entirely right, for the vast expansion achieved was not a commercial, but a war undertaking, in which the processes and procedures adopted bore little relationship to those which would normally be used to secure commercial production. It was, however, a matter of common knowledge that during the war magnesium was sold in the United States at a profit at 20 cents per lb. but a number of producers had had to be paid a higher price. The other points he would reply to in writing.

DISCUSSION

ON

An Experimental Investigation of the Effects of High Temperatures on the Efficiency of Workers in Deep Mines.

By A. CAPLAN, M.D., M.R.C.P., and J. K. LINDSAY, *Member*.

The President said that in view of the increased depths at which mining operations were now being carried on in many parts of the world, this paper had great topical interest and was of prime importance. He was sure that it would be greatly appreciated by all mining engineers. Mr. Lindsay, one of the authors, was present and would introduce his paper.

Mr. J. K. Lindsay said that the problem of working efficiency at high temperatures was one which was becoming of increasing importance, not only to mining engineers, but to many industrial undertakings throughout the world. On the Kolar goldfield the wet bulb had from experience been long accepted as an accurate index of comfort or discomfort. It had also been long accepted that the kata did not truly reflect an individual's personal feeling of comfort or discomfort. A temperature of 90° F. wet bulb was considered to be the approximate point at which a serious falling off in working efficiency took place. It was felt, however, that the question was one which warranted a more detailed examination, hence the present investigation.

Mr. Lindsay then summarized the procedure adopted in the investigation and the conclusions arrived at, as set out in the paper, and went on to say that since the paper was published his attention had been drawn to a publication by H.M. Stationery Office, entitled 'Environmental warmth and its measurement' by T. Bedford, a publication which was extremely useful to those interested in the problem. Bedford considered that at temperatures near the endurable limit effective temperatures make too much allowance for the dry bulb, and that the wet bulb alone might be a better index than the present effective temperature. His own opinion was that further investigation would be necessary to settle that point.

Dr. J. T. McIntyre said that the authors had done a striking piece of work on rather an important, if not the most important, study of the deep-level mining problem. Their results and conclusions were very clearly set out and would be read with great interest in other parts of the world, and more especially in South Africa, where they were now face to face with somewhat similar difficulties to those outlined in the opening remarks of the paper.*

These aspects of human efficiency in relation to environmental conditions were fraught with many difficulties of interpretation, however, and right away they were faced with measuring or estimating the amount of work (or energy expended) that should be considered a reasonable output under a given set of conditions. The authors had got over that hurdle, as far as Kolar conditions applied, by taking the maximum that their type of indigenous labour could produce without force or incentive in what appeared to be a satisfactory temperature condition represented by 83° F. wet-bulb reading.

As regarded maximum output, he would suggest that psychological as well as physiological aspects should be considered in some way yet to be determined, in order to obtain a fuller appreciation of the limits involved. That, of course, could only be done with many more subjects taking part in the experiments than the six men quoted.

It was of interest to note that a series of United States Army experiments were completed in the year 1944 and published in March, 1945,† giving certain limiting temperatures for a work rate corresponding to a total energy expenditure of 250 to 300 calories per hour on a continuous basis up to four hours' duration. The men in those tests marched at about three miles an hour with 20-lb. packs in an enclosure kept at various temperatures and humidities, and the effect was as if they were in an air current of about 260 feet per minute. Under the conditions of test the following schedule of wet-bulb temperatures appeared to fit the observations made on the 13 white men with ages ranging from

**Note by Dr. J. T. McIntyre :* In this connection, it is interesting to refer to a new Statute of the Union of South Africa, the Deep Level Mining Research Institute Act of 1946. The Act states briefly that the objects of the Institute are to promote and to undertake investigations and research in deep-level mining in the Union and in matters incidental thereto, with the object of inventing, discovering and developing such methods of mining, and such processes and apparatus, as will extend the limits to which ore may be profitably exploited. The Institute is a joint long-term venture, the expenses being shared by the Government and the mining industry, but other interests may identify themselves with the research work.

†EICHNA, ASHE, BEAN, and SHELLEY. 'The upper limits of environmental heat and humidity tolerated by acclimatized men working in hot environments'. *Journal of Industrial Hygiene and Toxicology*, Vol. 27, No. 3, March, 1945.

18 to 30 years and who had been enlisted as army volunteers (Table VI). Furthermore, it was noted that (*a*) at the upper limits

TABLE VI

U.S. ARMY TESTS
Limiting Wet Bulb Temperatures °F.

	Relatively Easy	*Difficult*	*Impossible*
Completely saturated environment	92·5	94·0	96·0
120° F. dry bulb temperature......	90·0	92·0	94·0

sweating was extremely profuse ; most men averaging 2·25 litres per hour and some reaching 3·5 litres per hour ; and (*b*) acclimatized men working above the upper environmental limits developed marked physiologic changes, and undesirable, frequently disabling, symptoms similar to those sustained by unacclimatized men when first working in the heat.

In commenting on Table VI he suggested that it was hardly applicable to conditions in South Africa or those in the Kolar mines, for a variety of reasons, two of which might be cited : (1) the posture of men in mining operations was different from that of the United States Army tests and the work rates were likely to be very different ; and (2) the mean radiant (confining wall) temperatures were different ; thus in mining they were above the dry-bulb air temperature, not below as noted in the United States Army tests (the effect would be to make a lower wet-bulb temperature necessary to counteract this in mining).

As a very rough comparison with the United States Army tabulation, the present author's figures might appear as in Table VII for the air velocities, basic rate of work, etc., used in the Kolar experiments. Those figures should constitute a useful guide to mining engineers and no doubt others would bring forward data to support or modify them in the future. Thus he hoped for a more detailed analysis of Dr. Caplan's and Mr. Lindsay's work from Mr. Barenburg of their organization in Johannesburg and he would not attempt to anticipate his figures.

TABLE VII

KOLAR GOLDFIELD TESTS
Limiting Wet Bulb Temperatures ° F.

	100 *per cent Continuous Work*	60 *per cent Continuous Work*	0 *per cent Continuous Work*
For Kolar environmental conditions	83·0	90·0	93·0

Professor E. J. King* said he was particularly happy to be present at the meeting and to hear the paper because it was almost exactly a year since he had visited the Kolar goldfield. He had been very impressed with the cooling plant on that field, which pumped a quarter of a million cubic feet of refrigerated air per minute down the mines. When he went down the shaft it seemed very cold indeed, but when he got down to the working levels it became hotter and hotter. One wondered how men could stand such temperatures, but they were working there and apparently quite comfortably. The atmosphere appeared to be bone dry and this probably accounted for the ability of the workers to withstand the heat. The absence of any moisture and the lack of visible dust were the two things which impressed him most.

Professor King then gave details of an experiment carried out by Hynes in India to test the effect of exercise in the heat. A concrete platform about 18 in. off the ground was used and the men, weighted with bags of stones, stepped up and down for six minutes. It was interesting to see how men in different stages of recruitment could stand that exercise. Men who had only been in service for a few weeks stood it possibly for two minutes ; men who had been in service for several months would hesitate after about three minutes but would rally with encouragement and complete the six minutes ; and men who had been in service for several years would complete the six minutes without faltering. By every medical test applied all these men were equal, but there was some physiological difference between them which was not properly understood—the difference, in fact, between a man in training and a man not in training. Recent work in Switzerland suggested that the difference might be associated with the cytochrome-C content of muscle.

Ideas as to the effect of heat on the human body were changing. A lot had been learned during the war and they knew that there came a time when the body could no longer get rid of the heat that it had to get rid of. When the temperature rose much above 106°, it was apparently not possible, in a great many cases at least, for the temperature to come down again. The man suffered from heat stroke, and often died, because his heat controlling mechanism had failed.

It is work such as they had listened to that afternoon which is helping to elucidate the still obscure physiological mechanisms concerned in the control of body temperature, and which would lead to a better understanding of the factors involved in the adaptation of human beings to difficult environments.

*British Postgraduate Medical School.

Mr. S. E. Taylor said that the authors were to be congratulated on their investigation into the effect of atmospheric conditions on the efficiency of certain mine-workers. As a piece of research work, its importance lay as much in the method as in the results. For much research laboratory work was essential and effective, but for research into the working efficiency of human beings it was almost impossible to reproduce in the laboratory all the conditions at the working place. However, the difficulties of research under working conditions were great, and the importance of the method adopted by the author lay in the way in which those difficulties were overcome and in the adoption of a task which was a perfectly normal one for the mine-workers concerned. As regarded the results, it was almost more important to find the optimum conditions for efficient work than to find the condition when efficiency fell off and when efficiency ceased.

In designing an air-conditioning plant for the ventilation of a mine it was necessary to decide upon the temperature and condition of the air that it was desired to produce in the working place. With a clear knowledge of the optimum atmospheric condition for efficient work and the condition when efficiency began to fall off it was possible to design an air-conditioning plant with some confidence that it would result in the maximum efficiency being obtained.

It seemed probable that the optimum condition for efficient work would be found to vary in different parts of the world owing to the differing environmental conditions in which labour was accustomed to live. It was, therefore, desirable for experiments to be devised and conducted in all mines where an improvement in conditions might be expected to result in improved efficiency. In any mine where it was found that the conditions differed considerably from the optimum conditions for efficient work, it was probable that air conditioning would prove advantageous economically and at the same time beneficial to the health of the workers.

Mr. Thomas Pryor said that as a result of their experience the authors of the paper considered that for Kolar the wet-bulb temperature was a reasonably accurate index of comfort conditions underground and definitely a better index than the wet kata reading. He earnestly hoped that this finding of the authors would not result in any tendency to discontinue measuring air velocities in working places underground. In his opinion it was always essential to measure and record three factors—dry- and wet-bulb temperatures and the air-velocity—to provide the necessary information to control comfort conditions. At Kolar the high temperatures and dryness of the air in the deep levels caused air velocity to have less effect than in mines where the air was more humid and lower in dry-bulb temperature, but at Kolar, as, indeed, in all mines, it was the ventilating current which must remove heat and dust out of the mine.

It was useful to have some single index figure, although only an approximation, to compare comfort conditions at one place with another. So many diverse factors affected human comfort and efficiency, and conditions varied so widely, that it was a vain endeavour to seek a relative index which would be really precise. It was far more important to have a clear grasp of the measureable factors which affect comfort. He was glad to see that the authors recommended the wider use of the effective temperature scale in mining, for the effective temperature chart did bring out the important features that, where wet-bulb temperatures were below 88° F., the velocity of the air had a marked effect on comfort conditions; that the cooling effect of air velocity diminished as dry- and wet-bulb temperatures increased until, for certain combinations of dry- and wet-bulb temperature, the air velocity had no effect on comfort; and that after this stage had been reached, increased velocity increased discomfort.

The authors remarked that the 'effective temperature scale drawn up by the originators does not reflect sufficiently the beneficial effects of air movement at higher ranges of temperature' and his own impressions at Kolar confirmed this. The difficulty would probably disappear by employing the effective temperature chart corrected for environmental warmth, as was used in the Navy, where comfort and efficiency conditions in high temperatures had received much study. Modern ships of war carried engines of very large horse-power installed in cramped spaces, and the heat radiated from these engines was a serious problem. In consequence, in certain compartments of the ships, especially when the men were at action stations below deck in tropical waters, the ventilation conditions could be very uncomfortable. The problem of heat radiated from the surroundings in which men work applied, of course, also to mines like Kolar where the temperature of the rock was high. At Kolar, initial rock temperature was 148° F. at 9,000 ft. vertical depth and increased 1° F. per 111 ft. of additional depth. The Navy corrected the effective temperature for environmental warmth by using globe-thermometer readings, which recorded the radiated heat, instead of ordinary dry-bulb thermometer readings. The disadvantage was that a globe thermometer required *20 minutes* to acquire and record the temperature of the surroundings. Full particulars were given in a booklet recently published by H.M. Stationery Office for 2s. 8d.,* including beautifully-printed large-scale effective temperature charts which should find a place on the walls of every mine survey office. This booklet contained much information of value and interest to those who were concerned with deep and hot mines.

When one was considering the factors affecting efficiency, it was the human and psychological factors, not directly measurable, which had more effect on the workers than the actual temperature and humidity. The low output remarked on by the authors as having taken place in a main level where comfortable conditions prevailed was, in his opinion, more due to other human factors than to the fact that the South Indian, acclimatized to hot working places, did not like to work in relatively cool conditions.

They now knew a good deal—and the effective temperature chart summarized much of the necessary knowledge—about the physical conditions which were required for reasonable physical comfort, but their knowledge of the social, political, and psychological factors affecting efficiency and output was less. When dealing with the physical conditions, they knew that acclimatization to the surroundings played a great part in the efficiency of the worker—but no acclimatization was at present possible as regards the social and political conditions, for in every country labour would not return to pre-war standards. The new conditions demanded new methods of approach, especially in India, where the attitude of labour had been profoundly affected by the war and by the political changes now in progress. It was necessary for the human factors to be studied with no less zeal and interest than had been paid to the physical factors of temperature, humidity, and air-velocity, and he knew that Mr. Lindsay had much at heart the importance of the human factors on the well-being and effectiveness of labour.

Sir Thomas Holland said that he could not find in the paper any reference to the barometric pressure. In mines differing in depths there must be differences in pressure and corresponding difference in the amount of oxygen which a man could take with each breath. Mr. Lindsay had stated that his findings were at variance with those of other workers and had given a quotation from Haldane, going on to say that this quotation emphasized the inaccuracy of Haldane's earlier belief that at 85° F. wet-bulb

*War Memorandum No. 17 'Environmental warmth and its measurement', issued by the Medical Research Council for the use of the Royal Navy.

temperature hard work was virtually impossible. He would imagine that if they compared the depths at which the experiments were made at Kolar, or, alternatively, the barometric pressure, with the depth or barometric pressure of the experiments conducted by Haldane, the result would not show any great difference.

Mr. Lindsay said he would like to thank all those members who had taken part in the discussion; it was very gratifying that the paper had raised so much interest. The actual temperature gradients at Kolar were 1° for every 172 ft. to a depth of 5,000 ft. and then 1° for every 115 ft. down to 9,000 ft.

He thought perhaps the reference to Dr. Haldane was rather misunderstood. In the paper they suggested that at 83° wet-bulb efficiency fell off, but they discovered that the Indian worker could still produce very hard physical work at a temperature of 95° wet-bulb, and they suggested that the difference might have been due to the fact that Dr. Haldane's men were working in a coal mine, whereas the subjects they were dealing with were Indians, used to high temperatures from birth and acclimatised by years of working under hot underground conditions. They had not referred to barometric pressure because they had not so far considered that barometric pressure had any real effect on workers underground. However, there was some difference of opinion amongst various mining engineers on the field regarding that point, and some engineers contended that at certain barometric pressures there was a marked decline not only in the efficiency of the workers, but in their own efficiency. At certain depths they maintained that they did not feel physically fit when making their daily round. It was a subject which they had not investigated, but it might warrant further investigation.

Mr. Sydney Taylor had brought out a point which was very material—that it was just as important to find out what the optimum temperature was for efficient work. The 100 per cent line they had drawn might not necessarily be the line at which they could get the most out of their labour.

A cordial vote of thanks to the authors of the papers was carried by acclamation.

CONTRIBUTED REMARKS.

Mr. B. R. Lawton*: The authors, when quoting my paper (1)†, express a desire to learn whether my remarks to the effect that acclimatized men 'can work quite efficiently' at a wet-bulb temperature of 88° F. imply the performance of efficient work in an environment having an effective temperature of 90°. They

*British Colliery Owners' Research Association.

†Figures in parentheses refer to the bibliography given on p. 42.

state that the calculated effective temperature for some of the working places with a wet-bulb temperature of 88° F. described in my paper is 90°. Their estimate is correct for the particular case (Ref. No. 25 Table I) ([2]), where the wet-bulb temperature was 88° F. and the air-velocity 200 ft. per min., but I neither stated nor implied that efficient work was being performed in such conditions. My original statement, quoted in full, reads as follows :—‘ With a sluggish air-current and a wet-bulb temperature of 88° F., the conditions are definitely uncomfortable, but with air-movement of about 400 ft. per min., the wet-kata cooling power is 11, and it has been observed that acclimatized men can work quite efficiently ’. This implies that efficient work, in working places having a wet-bulb of 88° F., can only be performed when the air-velocity is maintained at about 400 ft. per min. Conditions approximating to the above limits were given in Ref. No. 26, Table I; they were, dry-bulb temperature 91° F., wet-bulb temperature 88° F. and air-velocity 320 ft. per min. In these conditions it was observed that the amount of work performed by the workers compared favourably with that done in adjacent working places having lower wet-bulb temperatures. Moreover, the men were only partly stripped, which meant, in my opinion, that they did not find the conditions unduly oppressive. From the nomogram on p. 3 of the present paper the effective temperature of this working place was 84°, and not 90°, as suggested in the authors' comments. When effective temperature is used as an index, my original observations on the subject of the performance of efficient work at high wet-bulb temperatures may be restated as follows : acclimatized men have been observed to work quite efficiently when the effective temperature was 84°.

Additional support for this opinion was forthcoming during another investigation I made some few years ago. In this investigation a series of tests was carried out underground in a return airway where the air temperature was constant at 107° F. D.B., 81·5° F. W.B. The normal air-velocity was 280 ft. per min., but, by means of a suitable regulating device, it was possible to increase the velocity of the air flowing past the subject, who pedalled a Martin's Ergometer during the tests. The conditions maintained during the tests were as follows :—

(1) The working periods lasted for 15 minutes. The total working time for a complete test was 1 hr. 15 min.

(2) There was a rest period of 5 min. between working spells during which time the pulse rate and body temperature—taken

orally—of the subject were observed. In one test observations of the subject's skin temperature were obtained with a contact-type thermo-junction.

(3) The subject endeavoured to maintain a constant work output of 4,000 ft. lb. per minute. This represents an output of 180,000 ft. lb. per hour when due allowance is made for rest periods.

It is possibly of interest to contrast this rate of work with that done by the subjects employed in the work tests conducted by Yaglou and McConnell([2]). In their tests the rate adopted was 3,000 ft. lb. per min. and an hourly rate of 90,000 ft. lb.

I made seven tests at air velocities ranging from 290 ft. per min. to 1,010 ft. per min.—i.e., over an effective temperature range of approximately 83° to 86°. No sign of severe exhaustion was displayed by either of the subjects, but both complained of tiredness in the legs after each period of work. There was no abnormal rise in body temperature—the highest temperature being 99·9° F. after 75 min. on the ergometer in an air-current of 485 ft. per min. Subject W. H. during four tests had an average body temperature rise of 1·12° F., whereas J.B. showed an average increase of only 0·35° F. The rise in pulse rate also varied in the two subjects. The average increase in the pulse rate of W.H. was from 98 to 156, or 63 per cent increase ; that of J.B. was from 84 to 123, or a 46 per cent increase.

Dr. Caplan notes similar variations in the rise in pulse rate and body temperature of different individuals and suggests that blood pressure would be a more reliable index when considering limiting conditions for underground work. In my investigation I secured only one set of readings of the skin temperature of the subject, but I have often wondered if skin temperatures would serve as a suitable index of the comfort or discomfort experienced by the individual. Would Dr. Caplan care to express an opinion on this matter ?

In conclusion I would like to ask the authors if they consider that when progressive dehydration of an individual occurs during the course of a shift, is it possible for certain areas of the exposed body surface to become dry when exposed to high velocity air-currents of low humidity and high dry-bulb temperature ?

REFERENCES

([1]) LAWTON, B. R. Atmospheric conditions in Deep Mines (25th Report to the Committee on 'The Control of Atmospheric Conditions in Hot and Deep Mines'). *Trans. Instn. Min. Engrs.*, Vol. 93, Apr., 1937, p. 46.

([2]) *Ibid.*, p. 42.

(9) McConnell, W. J. and Yaglou, C. P. 'Work tests conducted in atmospheres of high temperatures and various humidities in still and moving air'. *Trans. Amer. Soc. Heat. Vent. Engrs.*, Vol. 31, 1925, p. 102.

Dr. C. G. Douglas*: The information given in this paper is, I think, of very great interest and adds materially to our knowledge about the relationship between high environmental temperatures and man's capacity to tolerate such temperatures and still to do useful work. Although other factors may come into play it would appear that under extreme conditions the wet-bulb temperature still holds its place as a very good and simple criterion by which to estimate whether or not particular environmental conditions are tolerable or intolerable. I think that it would be well worth while following up further the behaviour of blood pressure, for it looks as though this might prove a measurement of some importance.

When Haldane in 1905 originally stated the limit of wet-bulb temperature which could be tolerated by man it should be borne in mind that the figures were in the main based on the behaviour of but a few subjects who were not really acclimatized to high temperatures, for I do not think that either Haldane or his colleague Boycott would have claimed at that time to be acclimatized, while I myself, who acted as a subject in some of Haldane's laboratory experiments, certainly was not. The significance of acclimatization subsequently became apparent and Haldane's early figures may in consequence need some revision.

A difficulty is bound to arise in comparing the figures obtained by different observers for the reduction of efficiency due to rise of environmental temperature, owing to the fact that the initial basis for the computation may be different in different cases, and this difficulty is, I think, fully recognized by the authors. If under 'reasonable' conditions one group of men naturally works harder, with therefore a higher rate of heat production, than another group, the former should be more readily affected as the environmental temperature begins to approach the intolerable level, and the calculated efficiency of these men will fall off sooner and more rapidly than in the group of men whose natural rate of work is more leisurely.

Mr. J. H. Hohnen: I would like to ask the authors if their investigations into the physical reactions of mine workers to

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conditions of extreme heat have led to any conclusions on the mental effects of oppressive temperatures at depth. In an earlier paper* Caplan explained that fatigue is due to a combination of many factors, and adds ' Imperfectly oxygenated arterial blood (due to the rapid passage of venous blood through the lungs) adversely affects the higher centres in the brain . . .' This state might result in a condition of carelessness.

Spalding and Parker† concluded from earliest results of the improved air conditions at Ooregum that there was a marked ' reduction in the number of trivial accidents that occurred underground ' and said that ' it had been noticed for some years that the accident rate fluctuated annually according to the temperature '.

It would seem, then, that there is a definite point at which mental efficiency deteriorates and it would be interesting to have the authors' views as to the temperature and periods of time for which fully-acclimatized officials and workers can remain 100 per cent alert.

Mr. Jack Spalding : The question as to the best indicator for comparing the conditions in hot working places is an important one and one over which there is considerable difference of opinion. An indicator should be such that readings are simple to obtain and it should afford a true comparison of the various conditions within the extremes encountered in a single mine or group of mines. Since racial, climatic, dietary, and other extraneous factors vary the quality of the labour employed and, therefore, the effect of underground conditions on that labour, true comparisons between conditions in mining fields in different localities cannot be expected, and such figures as are obtained must be treated with reserve.

Each of the three commonly-used indicators of underground working conditions (the effective temperature, the wet kata cooling power, and the wet-bulb temperature) have various advantages and disadvantages, and in selecting the indicator to be employed on a mine these should be considered.

Effective Temperature Scale.—Scrutiny of this shows that in saturated or nearly saturated atmospheres small changes in velocity cause considerable changes in effective temperature, whereas the drier the air—that is, the wider the difference between dry- and wet-bulb temperatures—the less is the effect of velocity. Thus at 120° F. dry, 80° F. wet-bulb, from stagnance to 1,500 ft. per minute, a 2° fall in effective temperature is shown (from 90·4° F. to 88·4°). Those with experience of such conditions know, however, that that

**Trans. Instn. Min. Metall.*, vol. 153, p. 154.
†*Trans. Instn. Min. Metall.*, vol. 149, p. 578.

figure is entirely erroneous. Air at 120° F. and 1,500 ft. per minute is intolerable after a certain length of time, because sweat is dried from the body quicker than it can be formed, so that eventually the skin becomes dry and the air thereafter has a net heating effect. This is greater on the bare skin than when clothing is worn. Actually in air at 120° F., as the velocity is increased above 500 ft. or 600 ft. per minute, the effective temperature should start to rise, and both with and without clothing at 1,500 ft. per minute it should be not 88·4° F., but over 100° F.

This large error in the effective temperature scales is due to the fact that they were constructed from the observations of men passing from air at one set of conditions to air at another; they were based on the momentary feeling at the change. This gives most misleading results, for on the one hand acclimatization is neglected, and on the other the record is made before the man's body has had time to accustom itself to the new conditions—that is, before the pulse rate, blood pressure, and volume of sweat produced have settled down to those new conditions. The effective temperature scale is therefore definitely not to be recommended for hot dry mining.

In order to allow for radiated heat a corrected effective temperature scale has been devised, in which the difference in temperature between the containing walls and the air is taken into account. In the majority of mine workings this extra complication is not necessary, for, in all but newly-opened ends in rock at high temperature, the temperature of the rock surface is found to be within a degree or so of the air temperature, provided of course that no sudden change has been made in the ventilation.

Kata Thermometer.—The chief disadvantage of this instrument is the lack of simplicity in operation—with thermos flasks of hot water and stop watches it is the instrument of a specialist and not of the mine executive on his rounds. It is therefore only suitable for large mines.

In the majority of mine workings the air velocity lies within fairly narrow limits. Complete stagnance is never, or should never, be encountered. In all workings there is a slight drift of air, and the movements of men at work increase this to the equivalent of at least 100 ft. per minute. Since men seldom have to work in high velocities, it can be said that in general, therefore, the air velocity in most workings lies between 100 ft. and (say) 600 ft. per minute. In many mines those limits can be narrowed still further.

Except in mines where the air is near saturation it is found that the kata thermometer pays far too much attention to the air velocity, which, between the above limits, does not have a very great effect on the conditions. In air near saturation on the other hand, air velocity is of paramount importance and the kata thermometer is there a fair indicator of the true condition.

Wet-Bulb Temperature.—The great advantage of this indicator is its extreme simplicity—a single straightforward reading and the answer is obtained. For this reason alone it has much to recommend it. Its disadvantages are that it takes no account of the dry-bulb temperature in hot dry mines nor of the velocity in both wet and dry mining.

Between the different workings in a hot mine the variations in dry-bulb temperature are not great and usually lie within a range of 10° to 15° F. Such a small range of dry-bulb temperature does not greatly affect the condition of the working. Within the limited range of air velocities mentioned this factor also is comparatively unimportant, so that in the majority of mines, expecially those which are dry or semi-dry, the wet-bulb temperature is the best indicator of the condition of the working places.

Conclusion.—For routine work in a single mine or group of similar mines the wet-bulb temperature is the simplest indicator and gives a sufficiently accurate comparison between the conditions in the various workings. In dry or semi-dry mining it gives a truer indication of those conditions than either of the other methods. In wet mining the kata thermometer and the effective temperature scale are widely used, but if both are applied to the same mine their findings are sometimes at variance. This suggests that one at least is inaccurate. That most open to suspicion is the effective temperature scale, as being the more artificial. Of the two the kata thermometer is, therefore, probably the better.

When conditions become extreme it is generally agreed that the importance of the wet-bulb factor increases until it completely overshadows those of dry-bulb temperature and air velocity. In such cases it alone should be the criterion of condition.