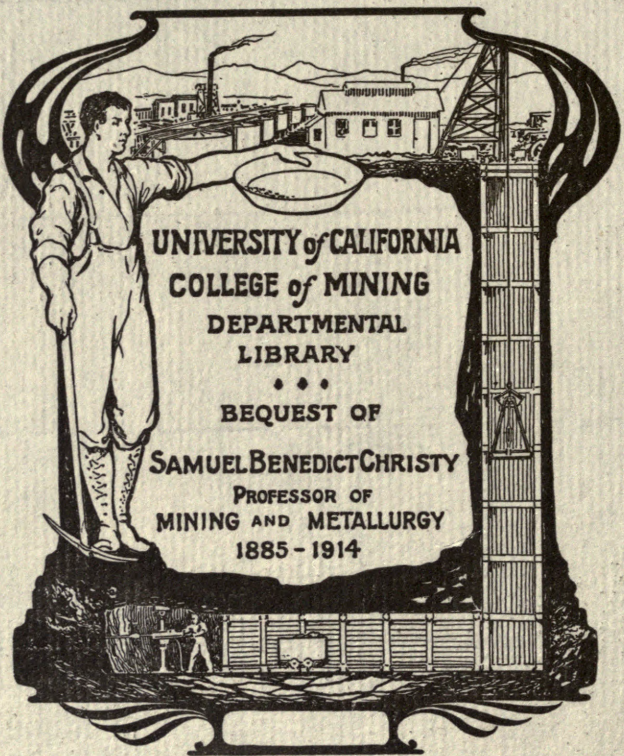
CYANIDE PRACTICE

BY

ALFRED JAMES.



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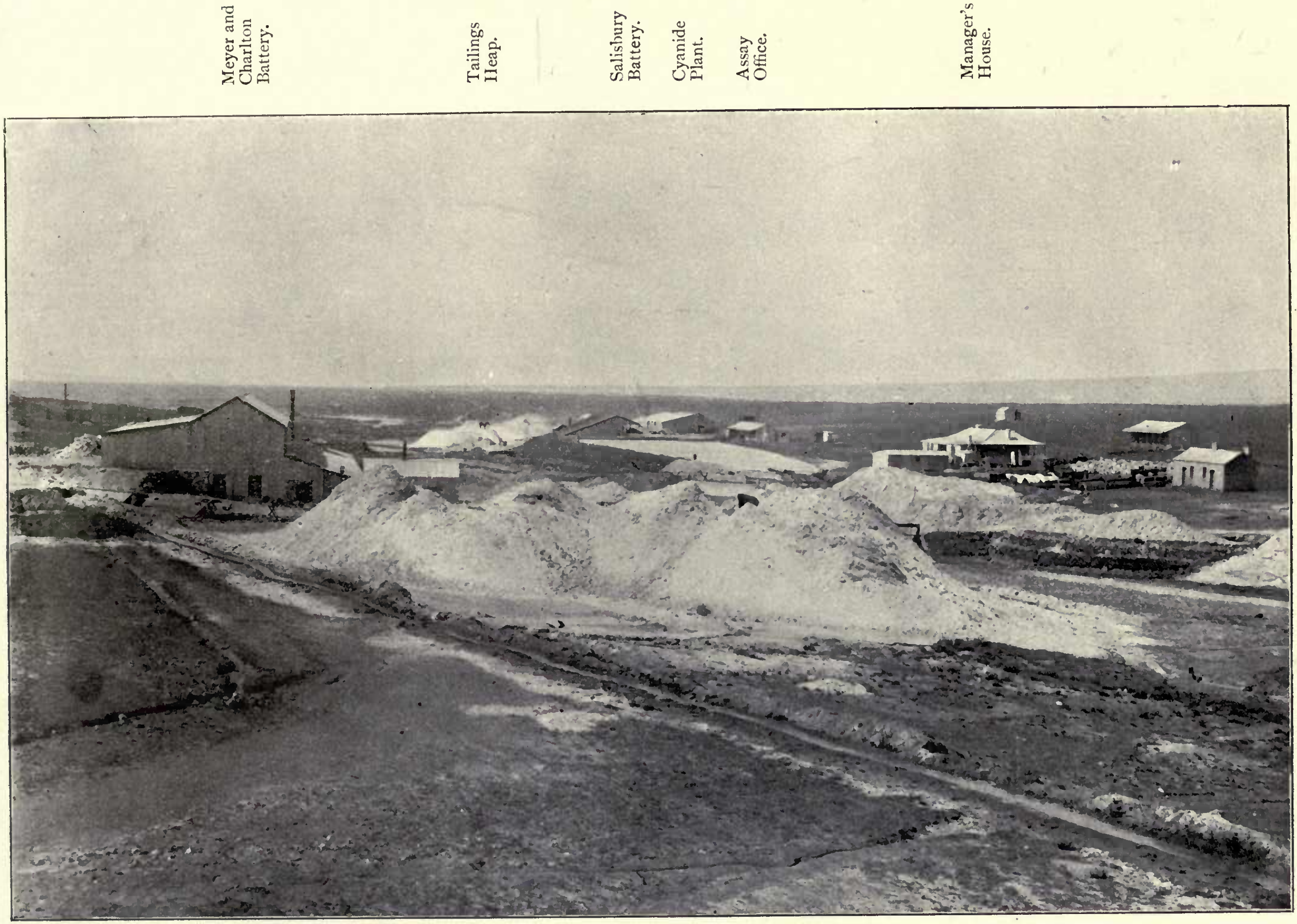

THE FIRST CYANIDE TAILINGS PLANT IN THE WORLD.
Designed and erected by Mr. Alfred James, at Johannesburg, April, 1890.

CYANIDE PRACTICE.

BY

ALFRED JAMES,

Mining and Metallurgical Engineer;

Member of the Institution of Mining and Metallurgy; Fellow of the Geological Society;

Fellow of the Chemical Society, etc., etc.

LATE TECHNICAL MANAGER AND THE SPECIAL EXPERT IN SOUTH AFRICA,
NEW ZEALAND, AUSTRALIA, UNITED STATES, AND ELSEWHERE,

OF THE

COMPANY INTRODUCING THE CYANIDE PROCESS.

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PREFACE TO THE SECOND EDITION.

A new edition of this book having been called for, the author takes this opportunity of adding the following supplementary remarks :—

INTRODUCTION.—Mr. Hennen Jennings writes that whilst pyritic tailings were first treated, as stated on page 9, at the "Nigel" plant (which also had the first set of intermediate settling pits, with doors for cars, and with separate slime pits, so arranged as to enable them to treat over 90 % of the mill product), the first place at which they tried direct filling was at the "Salisbury and Jubilee" mill, using square tanks with hydraulic classifier. It was, however, at the "Heriot" works, under the immediate supervision of Mr. Betty—who used round tanks—that this process became really successful.

INVESTIGATION OF SAMPLES.—Poor extractions may be due to the action of the cyanide being rendered ineffective by, for example :—

(1) The formation of soluble sulphides.

(2) The presence of coarse gold.

(3) The presence of a refractory product such as sulphide of silver or telluride of gold.

(4) The presence of cyanicides.

(5) The presence of base metals, which enter into combination with the cyanide, and thus consume it.

(6) The impermeability of the ores.

The remedies successfully employed are as follows :—

(1) Eliminate the metallic sulphides, or accelerate the action of the cyanide, by addition of bromo-cyanide, as suggested and put into practice by Sulman, and notably effectively carried out by Picard at Deloro; or grind fine, and add freshly precipitated lead carbonate or other lead salt, such as the chloride, to decompose any soluble sulphide immediately on formation. On the other hand, a refractory ore containing sulphide of antimony—which could not be profitably treated when fine crushed owing to the heavy consumption of cyanide—is yielding *extremely satisfactory results*, to *coarse dry crushing* using very dilute solutions.

A profit of over £80,000 is understood to have been made in this way in two years, from the treatment of an ore carrying from 4 dwt. to 5 dwt. only.

(2) This is usually treated by wet crushing and amalgamation prior to cyanidation; otherwise provision is made whereby the coarse gold in the residues is amalgamated by treatment in pans, riffles, or shaking copper plates.

(3) This may be concentrated out and the concentrates smelted; or it may be fine crushed and agitated with, or without, the addition of bromo-cyanide to the cyanide solution. Separate treatment for the silver sulphide, such as chlorinating roasting,

or lixiviation by hypo is apt to seriously lessen the recovery of the gold.

(4) Cyanicides are, as far as possible, avoided by preliminary washing, and by treatment with lime. In all cases acid salts should be removed before the cyanide solution is run on, and should, if possible, be discharged by a separate channel.

Thus if the ore is acid the water wash should be added from below, and flow off over the top of the vat, otherwise, as pointed out by Feldtmann, the cyanide solution coming into contact with the acid washings at the bottom of the vat, and in the pipe lines, is at once decomposed and complications set in.

(5) Quite the best practice is to concentrate out these base metals when practicable and cyanide the base-metal-free tailings. If this cannot be done one uses very dilute solutions, and if possible coarse crushes. Ores containing copper are successfully treated in the Northern Transvaal and Mexico by precipitating out the copper and thus preventing it from accumulating in and fouling the solutions.

In all cases of heavy cyanide consumptions it is as well to examine the solution and to determine the amount of cyanide accounted for by the metals in solution. Ores high in silver values will naturally consume much cyanide. The author has met with gold-silver ores, not concentrates, for which 6 lbs. of cyanide have been required per ton of ore treated, merely to account for the silver in solution!

Having found the amount of the metals in solution, if we multiply the copper by 3 (the exact figure depends on the composition of the copper salt formed; 2·75 is perhaps the most accurate figure, but Ellis considers that 3 in the absence of, or 2½ in the presence of, caustic soda agrees most closely with experimental data), the iron by 7, the silver by 1·2, and the gold by 0·7, we can then determine the consumption due to each metal; the balance, if still considerable, may be due to acid salts in the ores, or in the water used for solutions, or to any special method of procedure or apparatus employed.

A knowledge of the causes of the consumption having thus been gained, the necessary steps can be taken to counteract this accordingly, whether by elimination of the base metals by concentration, by getting rid of the cyanicides by preliminary washing or by decomposing them by the addition of alkali, or by lessening the time of contact required with the cyanide solutions by adding an accelerator, or by coarse crushing and the use of dilute solutions.

(6) Impermeability is remedied by a dehydrating drying, by calcining or roasting the ore; or by fine crushing, or even sliming, as may be best suited to the nature of the particular ore to be treated.

ALFRED JAMES.

January 9th, 1902.

PREFACE.

It is not intended that this little book should furnish a complete history of the cyanide process, or a law report of the various battles of patents to which the process has given rise. Nor does it give details of all the equations that have been published regarding the chemistry of the process, or of the more or less successful modifications which have from time to time been propounded or adopted in various parts of the world.

The earlier chapters have already been contributed by the author to the proceedings of The Institution of Mining and Metallurgy and The American Institute of Mining Engineers, and are now, with additions and some detail illustrations, brought up to date and assembled into book form. A considerable portion of the contents has, however, been written specially for this volume, and is now published for the first time.

It is hoped that the process section will give sufficient information to enable the mining engineer to accurately test his products, design and erect efficient and economical plants, and work them successfully. The practical details concerning the construction of vats and the method of arranging the foundations of double treatment plants should be of some use to the engineer called on to deal with such propositions.

The latter portion of the book specialises and shows the advances which have been made in the treatment of dry crushed and roasted ores, and in the treatment of slimes by filter-press and other methods. This section shows the general tendency of the various fields of the world to evolve types of plant peculiar to their own requirements.

While the same general principles govern the employment of cyanide for the extraction of gold from ores, it will be seen that modifications are many and that new practice is continually being called into existence; and therefore the reader will remember that although the practice described in this paper is the latest successfully employed up to date, it will be as well to see if other improvements have been made before, at some future date finally committing himself to the design of a ten-thousand-ton-a-month installation.

This book, being to a considerable extent a record of the personal experience of the author in the actual working of the process, is not a compilation and is necessarily limited by the non-inclusion of particular developments suggested or carried out by individual workers in various fields, and for particulars of such developments the reader is referred to the Proceedings of such Societies as the Institution of Mining and Metallurgy and to the excellent compilations and summaries of published statements concerning reactions, processes and equipment in "Gold Milling" by Lock, or the "Metallurgy of Gold" by Kirke Rose.

The author desires to express his indebtedness to the many gentlemen formerly associated with him in his technical work, and now in charge of large propositions in all quarters of the globe, for their courtesy in keeping him advised as to the progress and tendency of their individual practice, and to the Cyanide Plant Supply Company for permission to use their copyright Table of the Contents of Vats, and for some of the blocks illustrating this volume.

He also avails himself of this opportunity to suggest to all old members of the Cyanide Club at West Scotland Street that if they will keep him advised as to their movements, he will send to each one at Christmas time a circular or paper, enabling them thus to interchange ideas as to local problems of interest, difficulties to be overcome, progress and improvements effected, new spheres of occupation, social notes, &c.

ALFRED JAMES.

56, New Broad Street,
London, E.C.
December, 1901.

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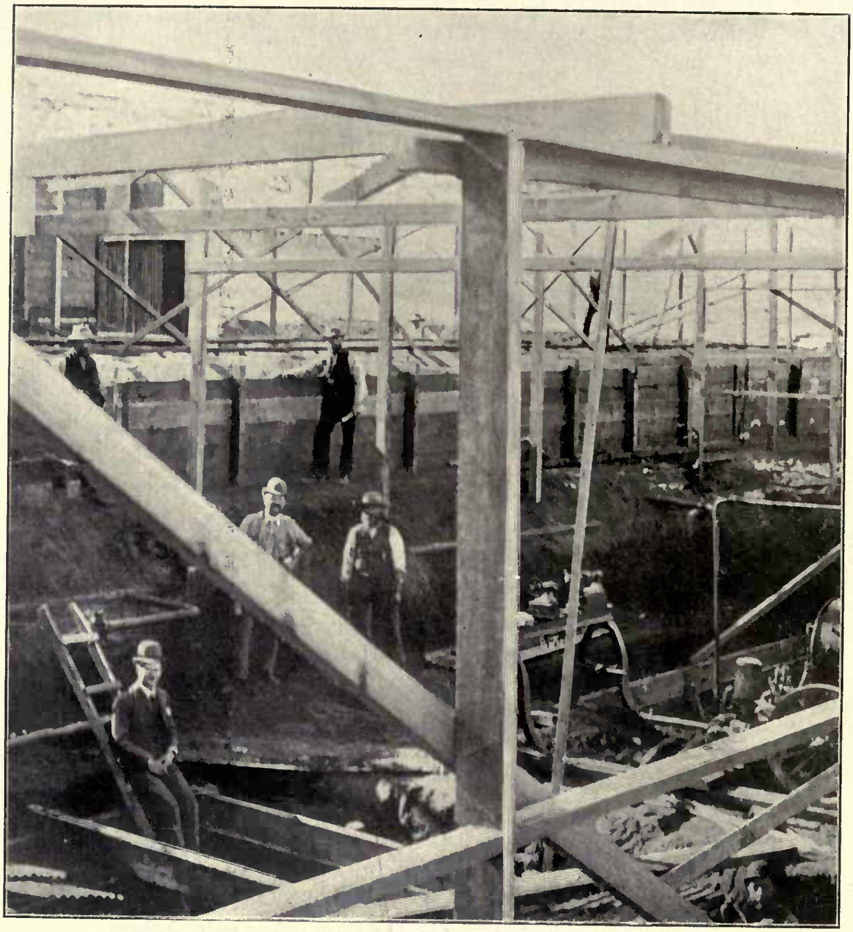
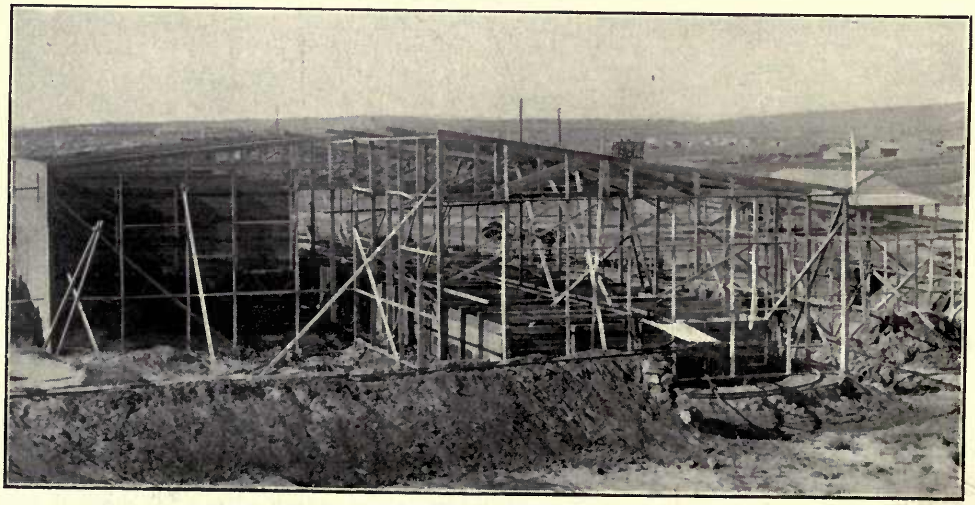




ORIGINAL ROBINSON TAILINGS PLANT.
The First Plant designed specially for Tailings Treatment, and without Agitators.
December, 1900.

INTRODUCTION.

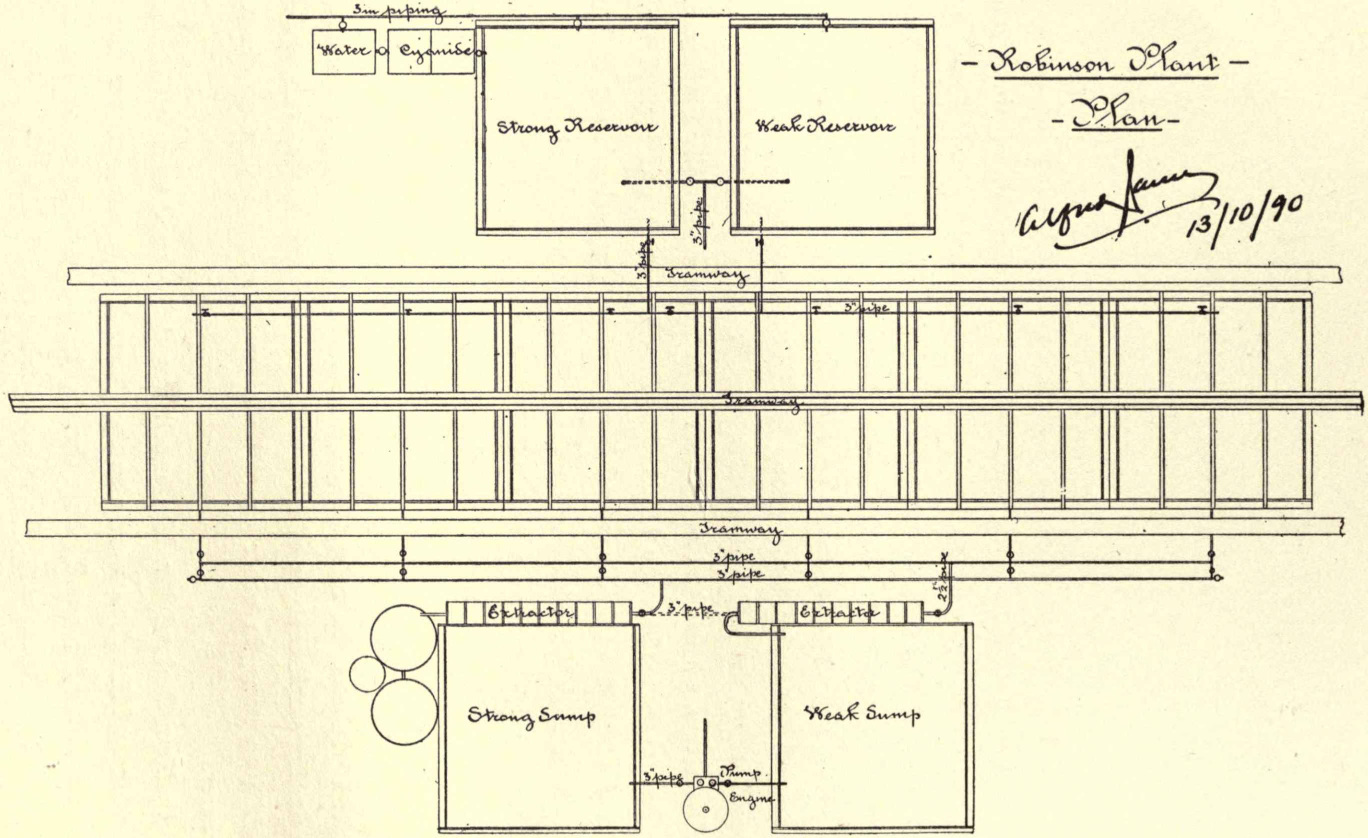
POSSIBLY the most remarkable tribute to the efficiency of the cyanide process is the practical relegation to the background of the numerous processes which formerly so persistently reappeared under different names and in the hands of different inventors, claiming highest possible extractions at a minimum of cost.

The cyanide process made no such claims: exploited with the greatest vigour and enterprise by a group of Scottish capitalists, it overcame all opposition by its efficient results, and to-day, with the exception of amalgamation, no method of extraction is more universally employed.

In October, 1887, the MacArthur-Forrest patent was applied for. By the end of 1888 expeditions had been sent to Queensland (Mr. McIntyre), New Zealand (Mr. McConnell), Africa (the author), and America (Mr. MacArthur), and later on to Chili, British Columbia, India, and Mexico. The Straits Settlements and Russian Asia were also exploited for cyanide purposes.

From the four earlier expeditions, that to Africa excepted, but little resulted. It must be remembered that at the outset the process consisted in agitating small charges of refractory ore or concentrates—in Queensland in old iron pans, in other countries in small wooden vats of about 4-ft. diameter.

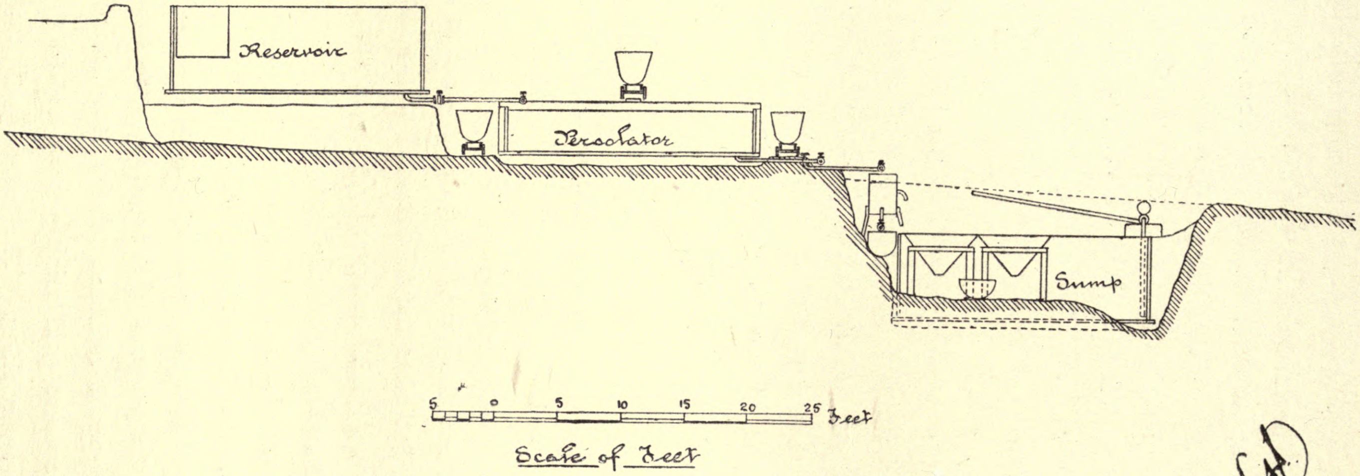
In New Zealand Mr. McConnell was doing good work, on the ores at the "Crown Mines," as were also the Queensland representatives, but, with the exception of a small plant for the treatment of concentrates at the "Sylvia," N.Z., their example was not followed.



- Robinson Plant -

- Plan -

Alfred James 13/10/90



— Section of Robinson Plant. —

These two Plans are reproductions of the original designs by the Author, with the working instructions omitted.

In America Mr. MacArthur was unfortunately unsuccessful in interesting the mining industry in his process, and it was not until some years had elapsed and the process was a great success elsewhere that the mining industry in the United States tackled the problem and made it the huge local success it now is. But, having once taken to it, the Americans developed the process on characteristic lines, which were quite independent of the methods pursued elsewhere, until at the present date some of the largest dry-crushing ore-treatment plants in the world have been designed and are worked by Americans.

On the other hand, the African expedition was admittedly a striking success. Confronted at the outset by the paucity of refractory ores and concentrates, the process was modified by the adoption of percolation and very dilute solutions, so that even tailings could be treated with success, and it was this adaptation of the process to the treatment of tailings which founded a new industry, the results of which are now as world-wide as gold-mining itself.

Up till this time not a ton of tailings had been treated anywhere in the world by lixiviation with cyanide solution, and the author is not aware of a single plant which was at that time working the cyanide process anywhere in the world except those worked and run by the patent-owning company and the small concentrate plant at the "Sylvia," built by Dr. Scheidel. But after the "Robinson" works showed a profit of £2,000 a month on an initial outlay of £3,000, practically the whole of the mines on the Rand adopted the process, and at the same time converted themselves into dividend payers;—for it is a most remarkable fact that scarcely six of the Rand mines would pay any dividends at all but for the profits received from the cyaniding of their tailings.

Any statement of the progress of the cyanide process would, however, be incomplete without special reference to the founders of the industry which has helped to convert gold mining from a rash speculation into a sound commercial undertaking. The success of the process was undoubtedly due to the far-seeing body of enterprising Scotsmen, chief among whom were Messrs. Leonard Gow and the late W. A. Vérel, of Glasgow, who took it in hand. Believing in their process, they trained a technical staff and equipped expeditions and sent them out at great expense to all the mining districts of the world. And in addition they maintained a testing and training institution, in which all modifications and suggestions affecting the process were investigated and tested, and a vast amount of original research work undertaken.

It is to their enterprise that such investigations as those detailed by the author in the chapter dealing with "bromo-cyanide" were rendered possible and carried out by, in addition to himself, such men as MacArthur, Ellis (who has initiated or carried out probably more original research and investigation work regarding cyanide reactions than any man alive), Akitt, Young, Marriner, Simpson, Leslie, Jolly, Russell, Jack, Warden, Muir, Miller, Jessop, Merrylees, Weir, Williamson, Johnston, Orr, Wilson, McIntyre, McConnell, William Jones, Hunt, Pitblado, Sawers, and many others to the extent of upwards of one hundred chemists.

It must be a matter of keen regret to all fair-minded men that the parent Company have reaped, comparatively speaking, but little benefit from their enterprising efforts. Bad management in various distant territories there may have been, but it must be admitted in favour of the parent Company that they have retained the respect, the good will—and the royalties—of all the countries personally worked and administered by them (such as New Zealand and India), and it is to be regretted that some of the sub-companies have been less fortunate.

To Mr. J. S. MacArthur the world owes the inception of the—we will not say idea, which may have occurred to others, but of the—proposal by which a solution of cyanide was to extract the precious metal from its matrix. It is true that Dr. Wernher von Siemens ran MacArthur very closely with practically the same process; so closely that originally the MacArthur-Forrest patents were refused for the South African Republic (Transvaal) owing to the prior patent of Dr. Siemens; and it may be that Mr. MacArthur himself obtained his idea—or was again reminded of it after a first fruitless investigation—from Hannay's patent of 1886. Be this as it may, all the taking of evidence and special commissions in Africa, Australia, and America only resulted in confirming the fact that until MacArthur took up the matter not a single ton of ore, concentrates, or tailings had been treated for the extraction of the gold by the process of lixiviation with dilute cyanide solutions and the recovery of the precious metal therefrom.

MacArthur undoubtedly deserves to be regarded as the father of the industry, not merely for the conception of the scheme but for the complete process whereby the dissolved gold was also recovered.

An application of much investigation, of scientific knowledge, and skill was necessary before the simple method was evolved of running the solutions through a mass of zinc thread—even the most suitable fineness of which was carefully determined—and to-day, fourteen years later, this process of recovery of the bullion has still scarcely a competitor. The zinc-lead couple has practically thrown out electric precipitation, and this zinc-lead couple was proposed and patented by MacArthur in 1894.

It has been the custom to regard the recovery of the gold as perhaps only a minor portion of the process, but, as one who has been intimate with all the details of the exploitation of the process, the author feels it his duty to express his high respect and admiration of the scientific skill whereby this portion of the process was worked out by MacArthur. Commencing with metallic zinc and sheet zinc, investigation was carried on with zinc fume or dust, zinc filings, scrapings, granular particles rods and threads of varying degrees of fineness, with the result that the method publicly propounded still holds the field.

In one other point a personal explanation is also perhaps due. The author desires unhesitatingly to place on record his knowledge that the recommendation of the use of extremely dilute solutions is also due to Mr. MacArthur and the Doctors Forrest. In the earliest days of the process such dilute solutions were made use of as to excite the incredulity of the technical staff of the parent Company who purchased the patents. As early as May, 1890, the author was successfully leaching "Robinson" tailings, with a "strong" solution of only 0·25 °/o KCy (and at that time only one of solution was used to two of ore), and the first large scale treatment ever carried out in Africa (on "Salisbury" tailings—"buddlings") was with solutions carrying only 0·225 °/o KCy on the ore treated.

The one instance in which strong solutions were employed was the test (to which we had been publicly challenged) of the "Percy" concentrates, carrying over 25-oz. of gold per ton! To make this a sure and rapid success—an actual extraction of 99 °/o was obtained for a treatment of less than 24 hours—a solution of 1·5 °/o on ore charge (3 °/o actual strength) was employed, but this was contrary to MacArthur's suggestion or desire.

It is due to Mr. MacArthur that this should be mentioned, as, at a meeting of the Chemical and Metallurgical Society some time since, the new President appeared to have been misinformed as to the strength of solutions ordinarily used in the introduction of the process, or recommended by the inventors.

In spite of the practical nature of the MacArthur-Forrest process, it took nearly four years of hard work before it was a recognised commercial success; but the successful adoption of the percolation process to the treatment of tailings, together with the continuous precipitation method afforded by the specially designed extractor boxes, made what had before been regarded as a pretty chemical scheme into a sound commercial success. And that this opportunity was afforded is due to the enterprise of Mr. Cordner-James and his associates of the Gold Recovery Syndicate, and to the far-seeing liberal policy of the African mines controlled by Messrs. Wernher, Beit and Company. But for the generous terms offered by Mr. Hennen Jennings, whereby a block of 10,000-tons of "Robinson" tailings was placed at the disposal of the Gold Recovery Syndicate, the success of the process would not only have been assuredly delayed, but might have reasonably been much less definite.

On the other hand, by his apparently liberal offer, Mr. Jennings was able to secure for his companies, without expense or delay to themselves, a proved process of the first rank, all details of the working of which he had had an opportunity to previously thoroughly investigate, and he was thus enabled to lay on a settled basis the foundations of that dividend-paying industry at Johannesburg, which is now without par in the history of gold mining.

But before leaving this subject of the introduction of the process, it will be interesting to glance at the improvements which have been made since the starting of the "Robinson" plant. These will be seen to be almost entirely mechanical, and it is naturally gratifying to the author, that the fundamental design, which he laid down for the "Robinson" plant, both as to the treatment of tailings by percolation, and generally as to the arrangement of the plant, of the circulation of solutions, and of the precipitation of the bullion, have proved so satisfactory in practice as to remain practically unaltered even to the present day.

And yet so many improvements have been made that, at the first glance, it might be difficult to recognise the relationship or resemblance between a recent double-decked steel vat plant and the old round-vat "Salisbury," or square-vat "Robinson" plants.

The following are probably the principal improvements up to date :—

(1). The adoption of vats of huge capacity. This improvement was due to that indefatigable worker, Mr. J. R. Williams, who, when he could not get wooden vats of greater diameter than about 25-ft., built his "Langlaate Estate" plant of cement, each vat being 40-ft. in diameter, and holding over 400-tons. This was a great advance over anything previously attempted.

(2). The adoption of bottom-discharge. This was introduced at the new "Robinson" plant, by Mr. Hennen Jennings, and his chlorination expert, Mr. Butters.

(3). The adoption of direct filling. This is one of the greatest improvements ever made in the economics of the process. It converted an intermittent system into a continuous one, and reduced labour costs by avoiding much of the handling previously necessary. This system was introduced by Mr. Jennings and Mr. Fearnside Irvine, his assistant, at the Nigel plant—which was also the first to treat pyritic tailings.

(4). The successful evolution of the slimes treatment process with which the names of J. R. Williams and Von Gernet will be so honourably associated.

After the success of the process in South Africa had become so well established, the other gold-mining territories of the world began to take to the process, each field modifying the method of application to its own particular requirements. The details of these local modifications and developments will be discussed in a separate chapter entitled "Recent Cyanide Practice," to be found at the end of this volume; and the more noteworthy of them will also be described more fully after the preliminaries of the process have been dealt with—such as the investigation of samples, the equipment necessary, and the usual method of extraction employed, which form the basis of the first section of this book.

CHAPTER I.

Investigation of Samples.

THIS subject will be treated not so much from a laboratory aspect as from a works standpoint.

One naturally forms an idea of the nature of the constituents present in a sample of ore from its appearance, whether it is friable, clayey, oxidised, or pyritic; whether there are any incrustations of metallic salts present, or any indications of copper, iron, zinc, lead, or other metals, and the forms in which these occur.

Samples of concentrates are usually panned to ascertain the presence of coarse or fine gold, pyrites, or amalgam.

Ores are usually ground to pass a 30-mesh sieve, and tests of the extraction are made of this fineness and compared with results obtained by still finer reduction. It must be borne in mind that a laboratory grinding to 30 mesh usually leaves the produce more coarse and granular than that produced by a stamp mill with screens of the same size. Not infrequently it is found that contact with solution for a longer time gives equal extractions to that obtained by finer grinding and shorter contact, and in this case the problem naturally is solved by commercial considerations of size of plant and amount of ore to be treated. It is a very notable fact that in the case of friable oxidised ores, of not high grade, good extractions have been obtained *without any grinding whatever*, the stuff being put through a stone breaker set fine, and sifted.

Tailings and concentrates are almost invariably tested as received, it being recognised that regrinding is more expensive than longer treatment, or even stronger solutions.

The usual practice is to take 1,000 gr. of the sample and to agitate this with 500 gr. of 0·5 °/₀ solution in a 12-oz. corked bottle, which is placed in a box on a shaft and left to revolve slowly overnight. On the following morning the solution is filtered off, the consumption of cyanide ascertained in a measured portion of the filterate by titration with $\frac{N}{10}AgNO_3$ standard solution ; and, after the sample has been washed, the total remaining filtrate and washings are evaporated to dryness with litharge, to determine the amount of gold in solution. The treated sample, now termed the "residues," is assayed. The amount of gold recovered from the solution compared with that found in the residues gives the extraction, which should be checked by an assay of the same sample before treatment. In evaporating the solution to dryness with litharge, it is advisable to add the litharge prior to commencing the evaporation ; widely divergent results having been obtained in consequence of the neglect of this precaution. The most correct method is to pass through zinc shavings, or dust, till solutions are gold free, dissolve the shavings in sulphuric acid, and cupel the residue.

If the cyanide consumption has been heavy, say, over 0·15 °/₀ on the ore, it is usual to test 1,000 gr. of the sample for acidity with a standard soda solution made up of such strength that each c.c. of soda solution per 1,000 gr. of ore tested equal 0·1-lb. of commercial caustic soda to be added per ton of ore. " Percolation " tests are then made, using an alkaline wash, or water and alkaline washes, prior to lixiviation with cyanide. If soluble salts are present they should be removed by water washing prior to the alkaline treatment. Where the consumption of cyanide has been low, but the extractions have not been good, longer period of contact, increased fineness of grinding, altered proportion of cyanide and solution to ore, and agitation instead of percolation, may improve the results, and tests are therefore made on these lines. In some exceptional cases it has been found that the extraction increased exactly in the ratio of consumption of cyanide ; thus a consumption of ·01 °/₀ KCy gave a 20 °/₀ extraction, and ·04 °/₀ KCy gave an 80 °/₀ extraction, the increased consumption being obtained by longer contact. Again, in some obstinate cases of cubical iron pyrites in a silicious gangue, agitation of the ore mass by jets of compressed air gave a fairly good extraction; agitation by oxygen instead of air did not improve these results, nor did the addition of hydrogen peroxide. This has led to the introduction of wooden launders in place of pipes for conveying the solutions from percolators to extractors, as this method aërates the solutions.

With refractory ores containing considerable amounts of sulphides of silver, copper, lead, iron or zinc, good extractions have been obtained from roasting ores which were not economically treatable in the raw state.

"Bottle tests" have hitherto been mainly employed; but, after a general idea of the behaviour of the sample has been gained, it is usual to make all tests by percolation, in view of the adoption of that type of plant when possible. In certain exceptional cases, however, usually with concentrates or certain silver ores, agitation is found to be practically indispensable, and in this event the larger tests are made in revolving barrels, or vertical stationary cylinders provided with agitators.

Percolation tests are made in bottles, lamp glasses, half-barrels or larger tubs, according to the amount to be tested; a false bottom is constructed in the apparatus employed, the dry ore laid on this, and the washes and solutions added.

Before proceeding with the erection of a plant, it is usual to confirm the small tests referred to above by experiments on larger quantities, with charges of from ½ to 3 tons; but laboratory tests, if carefully and intelligently made, form a most useful and reliable guide to actual practice, so much so that it is not unusual for large contracts to be made and purchases of tailings to be effected on the treatment of samples of less than 5 tons. One factor, however, requires to be borne in mind; it is that the consumption of cyanide is invariably greater in test experiments than in practice. An estimate of only ⅓ of the amount indicated by laboratory tests has been shown in practice to afford a safe margin when the laboratory consumption was not due to "cyanicides" or soluble salts.

On the other hand, it is not at all unusual for extractions to be obtained regularly in actual practice of a higher percentage than those indicated in the laboratory. As a general rule, one is, therefore, quite safe in estimating on the extractions yielded in experimental tests.

It is very hard to define the limitation of ores which may be profitably treated by cyanide; actual tests are necessary in each case, but ores with heavy coarse gold are best treated by separating this by mechanical means, prior or subsequent to lixiviation. Ores with a friable gangue not unusually yield better extractions than those in which gold or pyrites are in very hard quartz; whilst very decomposed concentrates, and ores with incrustations of soluble metallic salts, or with copper pyrites or antimony sulphides, do not lend themselves readily to commercially successful treatment. In the case of concentrates and tailings containing amalgam, means are usually provided for collecting the amalgam, which readily settles out after lixiviation.

The consumption of cyanide is ascertained by titration with the standard $\frac{N}{10}$ solution of $AgNO_3$. If this solution is used in a works, the foreman measures out 13 c.c. of solution to be tested instead of 10; and the number of c.c. of standard solution consumed divided by 10 gives the percentage strength in KCy of the solution. More usually, however, a special solution is made, containing 13·08 grm. of pure triple crystallised $AgNO_3$ per litre; 10 c.c. are then tested instead of 13, as with the $\frac{N}{10}$ solution, and the strength read off from the burette as before.

Thoughout this volume cyanide is referred to in terms of 100 °/₀ KCy. In actual practice, cyanide is used containing from 98 °/₀ to 126 °/₀ of cyanide estimated as the potassium salt, but the high grades usually contain a considerable amount of cyanide of sodium, which, when pure, contains 53 °/₀ of cyanogen, as against 40 °/₀ only in cyanide of potassium. Hence a sample of the pure sodium salt reported in potassium terms would appear to contain 132·5 °/₀ KCy.

Having now determined the best method for the treatment of the auriferous material under examination, it will be necessary to consider the type of plant in which this method may be carried out.

CHAPTER II.

Plant.

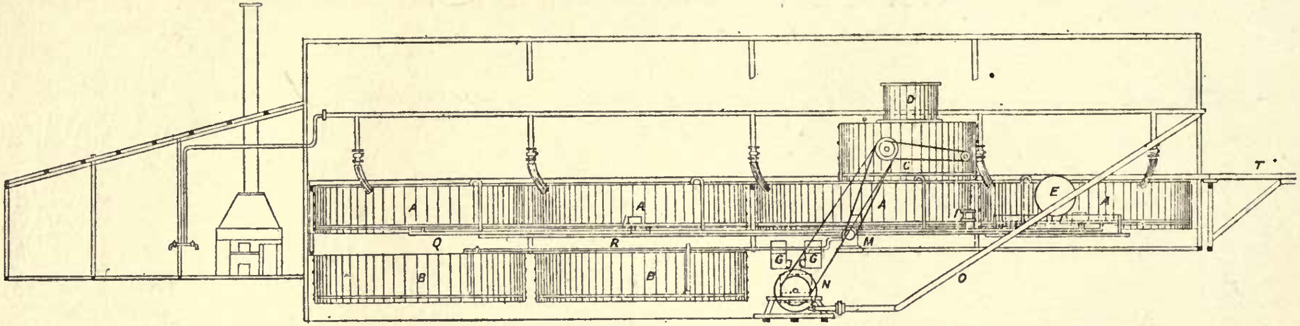
(I) Small Tailings Plant for treating from 500 to 4,000 tons a month.

A PLANT usually consists of percolators, reservoirs, and extractors, to which, if necessary, agitators may be added; but the last-named, except in the slimes process, are so seldom required that it is not necessary to here consider the details of their construction at any length.

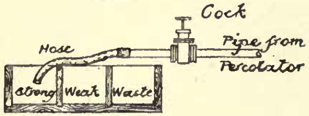
In Africa, buildings are usually erected to cover the extractors, suction apparatus, furnaces, assay offices, engine, pumps, and stores; but the percolators and reservoirs are in the open. In less dry climates it is usual to protect the whole of the plant by a galvanised iron and wood shed, the sides of which are left as much uncovered as wind and rain will permit, for the sake of increased light and improved ventilation. In very cold climates special provision must be made for the continuance, if possible, of operations during the winter months.

In considering the general design of a plant, it will be necessary to provide sufficient percolators—and agitators also, if these are to be used—to carry out operations in accordance with the method laid down as the result of the preliminary investigations. Thus, if it is found necessary to percolate and wash for two and a half days, and half a day is occupied in filling and emptying the percolators, it is obvious that one vat can have a fresh charge once in every three days, and there must therefore be three sets of vats, each capable of taking the whole of the daily output, with sump accommodation in proportion. Thus a 100-ton-a-day plant will require three percolators, each capable of holding 100 tons, or six capable of holding 50 tons, and to

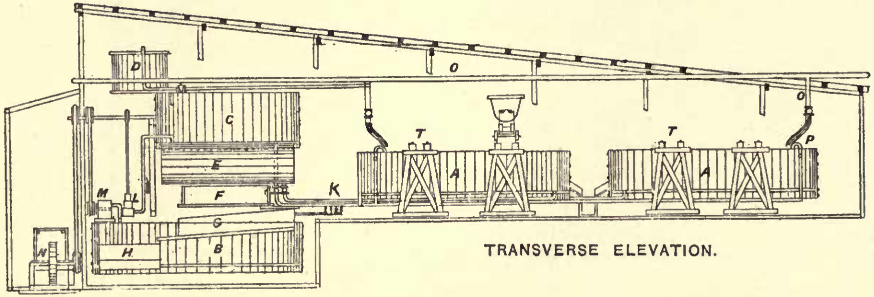
THE ORIGINAL WAIHI TAILINGS WORKS.



LONGITUDINAL ELEVATION.

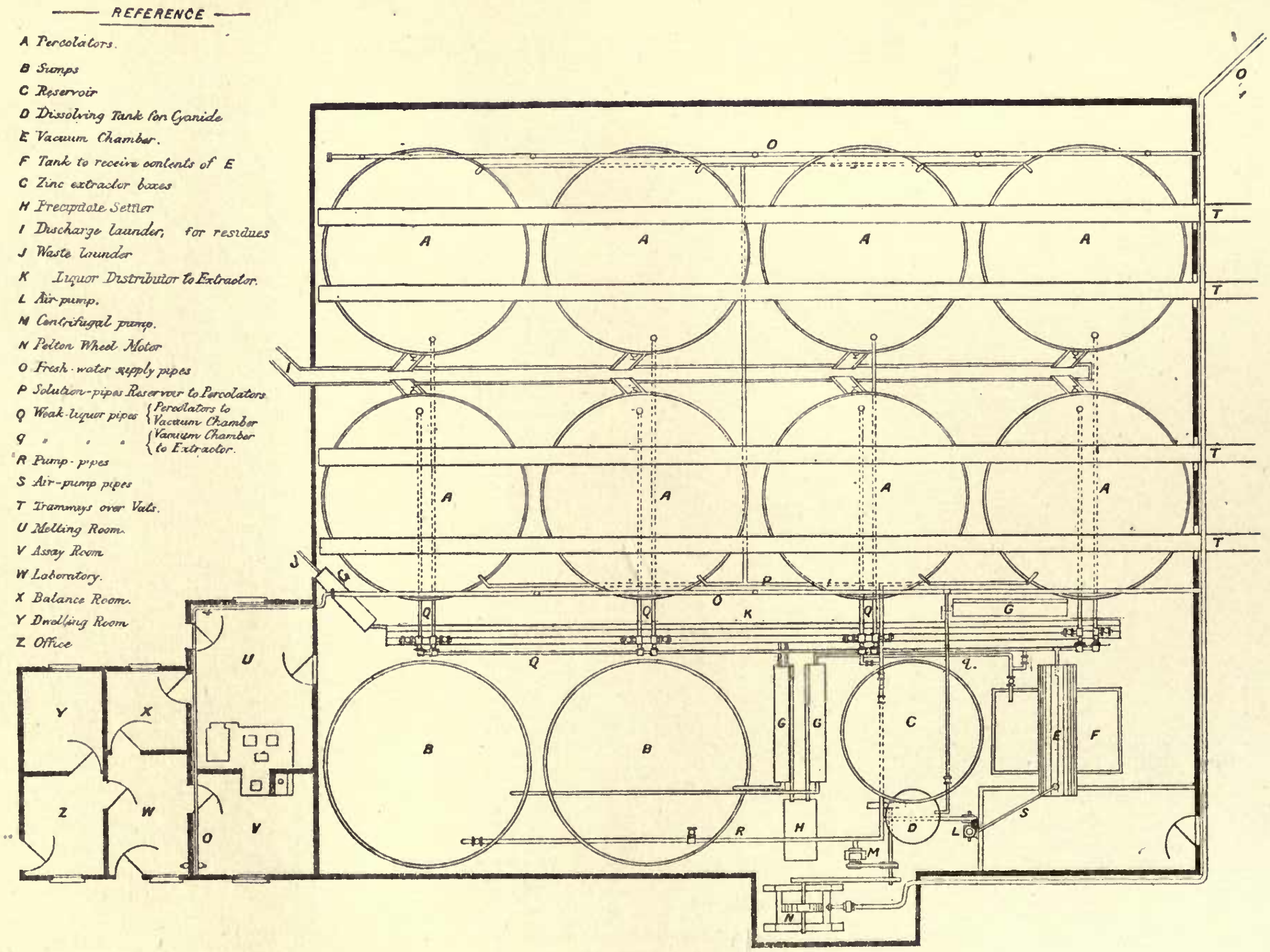


Detail of Distributor



TRANSVERSE ELEVATION.

ELEVATION OF THE WAIHI TAILINGS PLANT.



GROUND PLAN OF THE WAIHI TAILINGS PLANT.

this is added an additional vat for contingencies—renewing filter bottoms or general repairs, without interfering with regular operations. So that, in practice, a 100-ton plant would consist of four 100-ton or seven 50-ton percolators, the larger size vats being preferred from their greater convenience, their lessened cost in working, and the fewer pipe connections required.

There has been some difference of opinion as to the advisability of having very deep vats. In Africa, formerly, vats were made as deep as 14-ft. A few feet additional length of staves adds but comparatively little to the cost of a vat, but may readily double the capacity; on the other hand, the best extractions appear to be obtained in shallower vats. It is found that in all vats, except those of comparatively little depth, the lower layer of residues almost invariably contains more gold than the general average of the contents of the vat. This is so marked, when the tailings contain clay, that it was thought at first that the solutions were decomposed and the gold deposited by it, as lumps were found to be actually richer after treatment than before they were charged into the vats. This is now considered to be due to the comparatively dry clay absorbing the cyanide solution, and to the increased pressure on the lower layers rendering permeation of the solutions more difficult; it has been found almost impracticable to thoroughly wash these lower layers in practice, even with a vacuum.

Agitators.—These, if used, are worked on day and night shifts, and may thus be capable of treating 3 charges per diem; to lessen the large number of small percolators required to treat the small agitator charges, arrangements may be made whereby the whole of the agitators can be discharged simultaneously into one vat only, of increased dimensions. In all these plants vacuum filtration is applied to the percolators, otherwise the pulp is filter pressed.

The agitators are usually cylinders or barrels; the original form was a wooden vat 4-ft. high and 3-ft. 9-in. in diameter, made of 2-in. staves and bottoms. The staves are planed to the radius of the vat, so as to make a true joint without tongues and grooves or packing of any kind; the bottoms are dowelled. An upright iron shaft, with agitator blades of the same metal, revolves within the cylinder, supported on a pivot fixed in the centre of the bottom of the vat. Half a ton of solution was run into the cylinder, the agitators set in motion, and a ton of the ore was added gradually, so as not to cake and clog the apparatus. Six to twelve hours' treatment was usually sufficient to yield a good extraction. A cock at the bottom of the cylinder was then opened, and the contents discharged, whilst the blades were still running, into a percolator or filter. The subsequent process was identical with that of the present percolation method. When concentrates or tailings were being treated, a notable quantity of amalgam usually settled round the pivot under the agitator blades, whence it was easily collected.

This apparatus gave good extractions, but with an increased consumption of cyanide. Then, also, the power consumed, the constant watching to prevent the ore settling and thus clogging the agitator, the liability to accidents, the wear and tear of the working parts, and the effect of the cyanide on the iron were all so serious as to cause this method to be superseded, wherever practicable, by simple percolation.

The most recent type of agitator, however, and the most successful, is that employed for slimes treatment in West Australia in vats of upwards of 30-ft. diameter and 7-ft. deep. This form consists of a centre vertical shaft, free at the lower end, or working in a cement instep. To the shaft two pairs of horizontal arms are fitted in sets of four, each arm braced to its neighbour on the same plane and also to the nearest arm of the other tier. The arms may be conveniently made of angle iron, and usually extend to within 1-ft. of the side of the vat.

The agitator is driven at about six revolutions per minute by the bevel wheel gearing. Some of the agitators are fitted with apparatus for raising the stirrers out of the sludge, but this is seldom used.

A cheap and effective agitator is made by hanging pieces of old iron rope from horizontal arms radiating from a revolving vertical shaft.

Barrels have been tried as agitators, but are not in general use; and settling pans were found to be too small and to consume too great an amount of power. But all agitators, apart from other demerits, have the disadvantage of requiring power, and this alone is sufficient to account for the decided preference given to the percolation method of treatment.

Percolators.—These, the main feature of the plant, are usually arranged in one or two rows, in such a manner as to afford the greatest convenience for filling and discharging, and for controlling the flow of the solutions. They have been made of masonry (brick and cement or concrete), timber or steel, and may be round, oval, or rectangular. In permanent works of considerable magnitude, brick and cement pits were at one time regarded with favour, but have fallen into disrepute owing to the difficulty of detecting leakage, and the liability to cracks resulting from earthquakes, settlement or other causes, whilst in smaller works, and those for temporary purposes, round or rectangular wooden vats are in use. Steel vats, protected with asphalte varnish, are now, however, most usually employed. They are light and strong, and when coated with asphalte varnish have the great advantage of being less liable to decompose cyanide solutions than wooden vats, and permit of the location of leakage—which is almost impossible with cement vats.

For small syndicates vats of sheet iron, and even galvanised iron (painted internally), have been employed, but these are of course not recommended for permanent installations. The life of such vats would be too short, to say nothing of the leakage caused by the stress of filling and emptying them. For experimental purposes vats have even been made of filter-cloth, with satisfactory results.

Wooden Vats.—In new mining districts, or in those in which lixiviation is being attempted for the first time, rectangular vats—"tanks"—are usually constructed for the first plants, until sufficient skilled labour and experience is available to undertake the building of the round type—"vats."

We will first consider the construction of the rectangular tank. This is made of 9-in. × 3-in. white or red pine deals, well-seasoned; the joints are tongued and grooved; no packing of any kind is permitted, but white lead may be used. The bottoms are bolted together by ⅞-in. bolts running through the deals at every 13-in. or less, and the sides are let into the bottom, which is carefully gained ¾-in. for this purpose. The bottom projects 6-in. beyond the sides at the ends of the deals, but only 4½-in. at the sides of the deals, otherwise the 3-in. gain would reach exactly to the edge of the first plank where it makes a joint with the second.

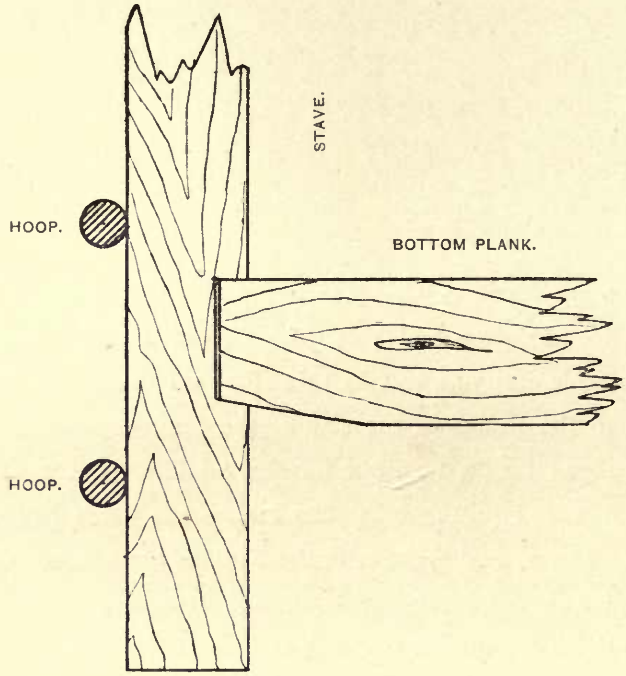
The sides and end bolts are ¾-in. in diameter, and run through the sides and bolt them to the bottom, and are at equi-distant spaces with the bottom transverse bolts. The bolts at the corners of the sides are as close as possible, and are connected by a triangular iron cap to keep the corners well down. The projecting ends of the sides of the tanks are drawn together by three ¾-in. bolts to keep them tight on to the ends. At every 30-in. or less, struts 4½-in. × 3-in. and the height of the tanks are morticed into the bottom and fixed against the outer side by coach screws to afford additional rigidity. Three 9-in. × 3-in. planks run across the top of the tanks at equal spaces, and act as distance-pieces as well as tie-rods for two of the sides, whilst the other two sides are supported by distance-pieces between them and the adjacent tanks. The vertical side-bolts shall be continued through the 9-in. × 6-in. foundation-pieces to afford additional rigidity to the bottom. These tanks are of any size from 11-ft. × 9-ft. × 3¾-ft. to 24-ft. × 16-ft. × 5-ft. All the first working plants in Africa and New Zealand—demonstration plants excepted—including the Robinson, Sheba, Mascotte, Caledonian, Nigel, Crown Reef, Ferreira; New Zealand Crown, Try Fluke, Great Mercury, Te Aroha, and Welcome were constructed of this shape.

These "tanks" have the advantage of requiring less skilled labour, are generally less costly, and more easily built, and take up less room, and where the tailings have to be discharged by hand into trucks at the sides, they permit of more trucks being filled at once, and the tanks are thus emptied in less time. But circular vats are stronger, stand wear better, shrinkage can be taken up more easily, they may be built of larger dimensions, and altogether they make a better job. In the majority of permanent works, therefore, the percolators are round.

Whatever the shape of the vats, it is usual in designing to allow about 30 cub. feet of space per long ton of ore contents, with an additional depth of 9 to 12-in. to allow space for the filter bottom, and for solutions to be run on to the top of the ore; thus, a 30-ton vat might be 19-ft. 7-in. in diameter, by 3-ft. in height, which gives 900 cub. feet, plus an additional 9-in. for filter bottom and solution space, making total height 3-ft. 9-in. The space per ton, however, must first be accurately ascertained, as certain clayey ores and tailings require a greater allowance than this. Concentrates occupy less space.

Circular vats are made of well-seasoned wood, free from knots. The staves are 4½-in. to 5-in. wide, and 3-in. thick for vats up to 25-ft. diameter, and 5-in. to 6-in. wide and 3-in. to 4-in. thick for vats above this size. Each stave is shaped to the radius of the size of the tank—not quite truly, but slightly feather-edged inside, to allow for the swelling of the skin of the timber in contact with liquids, which thus makes the joints perfectly tight. No tongues and grooves, or dowels, or bruised joints, are allowed in making the stave joints, which must be planed perfectly true and accurately fitted. The staves are at least 9-in. longer than the inside depth of the tank, and this allows for a 3-in. gain ¾-in. deep to receive the bottom of the tank, and for a projecting horn of 6-in. The sides are built vertical, there is no batter. The bottom is made of 12 × 3-in. planks planed to make true joints; tongues and grooves, bruised joints, or packing of any kind are not allowed, but white lead may be used, and the bottoms are dowelled, occasionally bolted together, and in some cases screwed down on to the bearers. The bottom planks are chamfered at their *lower* edge where they are recessed into the side staves; in New Zealand it was the practice to place round rubber, say $\frac{3}{16}$-in. diameter, around the bottom to make a tight joint at the staves.

The sides are held together and to the bottoms by round iron hoops, one for every foot of internal height of the tank, and one extra at the bottom; for a 25-ft. tank the two lowest hoops are 1-in. in diameter and the others $\frac{7}{8}$-in. They are made in about 20-ft. lengths, each length having both ends threaded and connected to the next by tightening boxes or lugs.



This enables slackness in any particular part to be at once taken up by tightening the adjacent nuts, and is also of advantage in fitting the hoops to the vat.

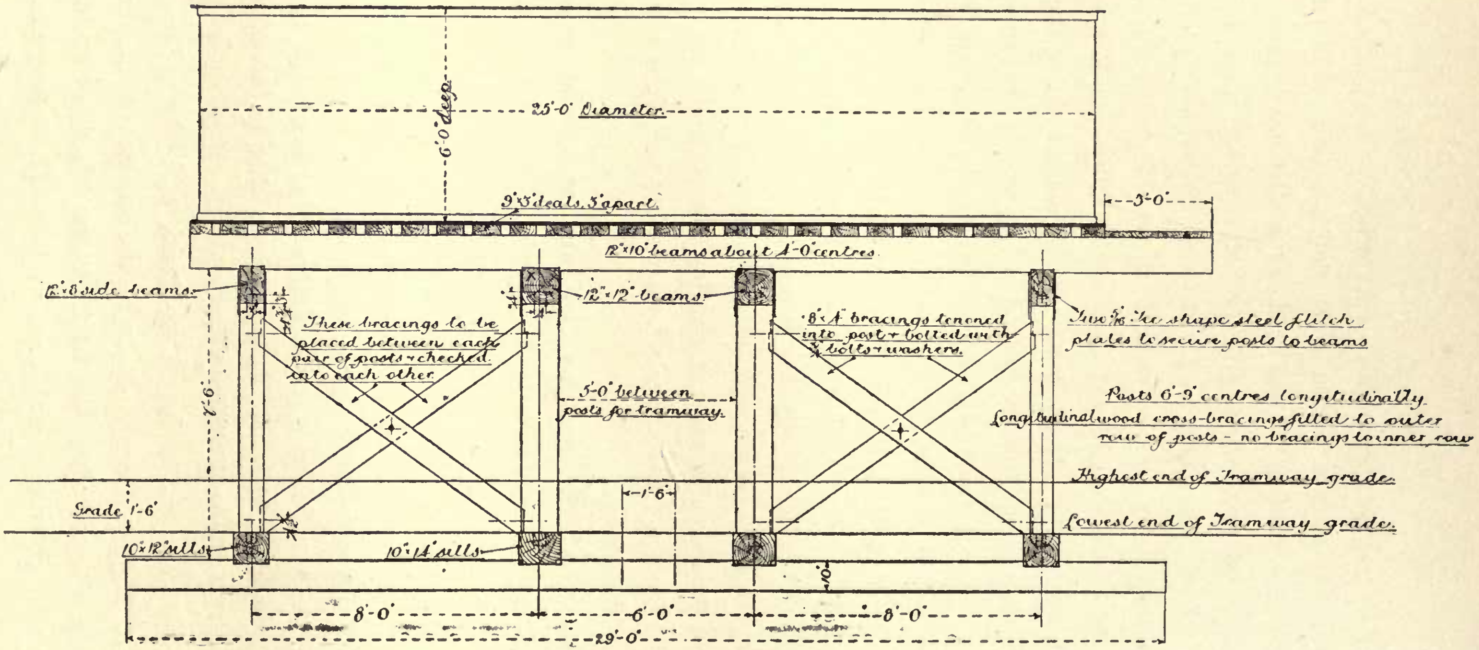
Both vats and tanks should be dressed smooth inside, and a treatment with solid paraffin is advantageous, though this is not usually given.

Filter bottoms are constructed of a ring of wood 3-in. high and $1\frac{1}{2}$-in. thick around the bottom of the vat, $\frac{1}{2}$-in. distant from the inside of the staves. The ring is fixed to the bottom of the tank by screws or wooden pins, and in the space between it and the staves are packed tightly the ends of the filter-cloth and the caulking rope. Within the circle formed by this ring, or in the case of rectangular vats within the parallelogram formed by the bounding slats, 2 × 2-in. filter slats are laid parallel across the planks forming the bottom of the tank, at a distance of 6-in. from each other. In vats not exceeding 5-ft. in height, these slats may be 10-in. from each other. At every 12-in. they are screwed or pinned to the bottom through wooden squares 2 × 2 × 1-in. high, which act as distance-pieces to keep the slats off the bottom, and allow ample room for the solutions to flow beneath. If the vat is more than 8-ft. high, the filter slats must be of 3 × 2-in. timber instead of 2 × 2-in. to stand the increased pressure. Across these slats 1-in. semi-circular ordinary ceiling moulding may be nailed in parallel lines with rounded side uppermost at intervals of 2-in. from centre to centre—this gives 1-in. spaces.

The filter may be of any of the materials mentioned above, but cocoa-nut matting or jute is usually employed for percolation unassisted by vacuum suction, otherwise cotton twill or canvas is preferred. The actual filter is protected by a layer of cocoa-nut matting over it, and then by slats of wood arranged to prevent the shovels from cutting the cloth when the residues are being discharged.

Where water is available provision is made for sluicing away the residues through a door in the side of the tank, placed at such a level that the bottom edge of the opening of the door is about ½-in. below the level of the top of the filter. These doors are usually 12 × 6-in., though in some cases they are made of larger sizes, a plate with a rubber ring on its flange being pressed tightly by a screw on to a frame, bolted on to the opening in the vats. The screw and thrust bar are made to come away from the lugs when the door is opened, so that no damage may be caused to them by the rush of pulp.

When the residues are discharged by hand, and the sides of the vats are over 4-ft. high, a door is built in the centre of the bottom, and the residues are shovelled into the trucks below. In large tanks there are as many as six or eight of these doors. Where this bottom discharge



End Elevation

Timber supports for small single treatment plant.

is not convenient, doors have been placed in the sides at a suitable height through which the residues are shovelled, but this practice is practically obsolete. In masonry vats tram lines are laid over the filter, and trucks run into the vats through iron doors built into the sides.

The foundations for large tanks are usually constructed of masonry, built in walls of the required height—say 5-ft. The outer walls are rounded to the shape of the vats, and with the inner walls are arranged to admit of a tram service under the discharge doors, and also for access to any part of the bottom of the vats to examine for leakage. The vats do not rest directly upon the masonry walls, but upon intermediate bearers of 9 × 6-in. timber, running in a direction at right angles to the planks in the bottom of the vat. The bottom of the staves must be quite free from the foundations ; they must have no strain, and leakages being most apt to occur at the point where the bottom is gained into the staves, freedom of access must be provided for.

In vats of less size, up to 25-ft. diameter, and not exceeding 6-ft. high, wooden foundations answer perfectly well. The vats are supported on 8 × 6-in. bearers, about 3-ft. apart from centre to centre, resting on posts 6-in. square, braced by diagonal struts of 4½ × 3-in. timber. The sills are 8 × 6-in. planks, and these again rest on mud sills, which are intended to prevent any settling of the foundations. The dimensions above given are for the 25-ton vat and 2-ft. 6-in. length of posts ; larger vats or higher framework will require stronger posts. The foundations are octagonal in shape, the radius of the octagon being at least 3-in. less than that of the internal diameter of the vats ; this allows the ends of the staves to be quite free.

On the outside of the sluicing-out door an apron is fixed to direct the residues into the launder, which conveys the treated pulp from the works to the dump.

Reservoirs.—These should usually be of sufficient capacity to contain at one time all the solutions—strong and weak—in circulation in the works ; the dimensions thus depend on length of treatment and proportion of solution and washings to ore treated, but, as a general rule, reservoir accommodation in small plants is provided equal to one-third of the cubical contents of the tanks. Thus, for six percolators there would be two reservoirs of the same dimensions, the one for strong and the other for weak solutions; but where the weak solutions greatly exceed the strong in amount, the comparative sizes of the respective reservoirs must be altered accordingly.

These reservoirs are constructed in precisely the same manner as percolators, except that they have no doors and no filter bottoms.

When reservoirs are placed at a less elevation than the percolators, so that solutions may flow into them direct from the extractors, they are termed "sumps," and it is a convenience in this case to have at an elevation above the percolators an additional smaller reservoir of such capacity as to contain the amount of solution required for any one charge. Thus, if a plant treats 100 tons a day, and uses an amount of strong solution equal to one-third of the weight of the ore, the smaller reservoir would have a capacity of about 35 tons of solution; or if the plant treated 200 tons a day, charging two 100-ton vats a day, one in the morning and one in the evening, the reservoir would still be of 35 tons capacity.

The advantage of having this upper reservoir is that the limited amount of solution required may there be made up to the required strength, instead of having to keep all the solution in the sump always up to the normal strength; and also that one does not require to be running the pumps so frequently, as the solution once pumped up to the reservoir may be allowed to run on to the percolators at such a rate and in such quantities as desired; this in practice is of the greatest convenience, and minimises the inconvenience consequent on a breakdown of the pumps. The reservoir is fitted with a tell-tale, consisting of a float actuating an ordinary variable level indicator, which, working against a scale of tons, thus shows the contents at a glance.

Another plan is to have the two reservoirs at a higher level than the percolators; the solutions flow from the latter into a small sump, whence they are continuously pumped up into the reservoirs. This is not such good practice, as it necessitates the constant employment of power for pumping, and the total suspension of operations in case of a breakdown to the pumps, unless a duplicate set is provided. It also requires the whole of the solution in the upper strong reservoir to be kept up to normal strength, and this is inconvenient when one requires to treat exceptional charges with different strengths of solution. It has the advantage of lessening the elevation required for the percolators, or of the alternative excavation for the sumps.

Where alkaline washes are required, an additional reservoir must be provided for the storage of these.

If vacuum suction is used two small tanks will be required, each capable of containing the contents of the receiver, and connected with the strong and weak extractors respectively. This permits of continuous working. These tanks are usually made square, as occupying less space, and shallow, so as to keep the receiver as low as possible to prevent loss of suction power or increased elevation of percolators.

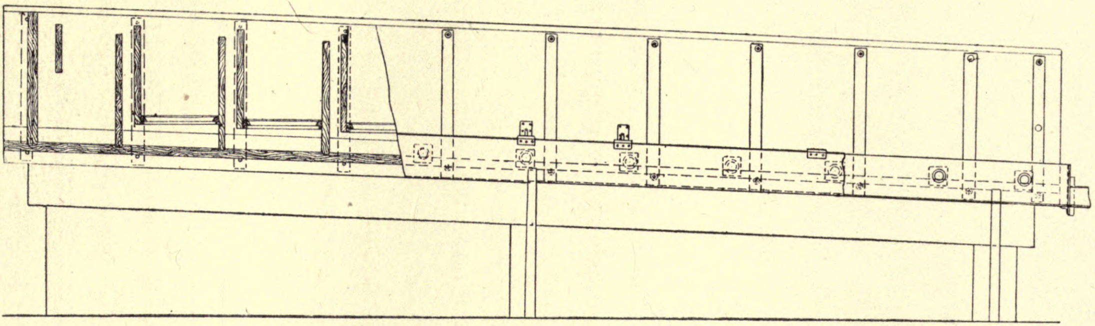
There is also a small reservoir, usually made of iron or wood, though cement or earthenware would be preferable, in which the solid cyanide is dissolved. This is effected by placing the cyanide in lumps on a movable suspended tray kept near to, but below, the top of the solution contained in the tank; by raising or lowering the tray, the contents of the tank are agitated and the rate of solution increased.

Extractors.—These are long boxes or troughs containing the prepared zinc, and so arranged that the cyanide gold solution is forced by baffle boards to pass through a considerable amount of zinc before it leaves the extractor. They are usually constructed with ten zinc compartments, each of such dimensions that the amount of solution to be treated may flow up through the zinc easily and steadily, without eddies and without disturbing the precipitated gold. In practice, each compartment is designed to hold about one cubic inch of zinc for every ton per month treated. Thus a 2,000-ton-a-month plant would have extractors with compartments of such size as to contain about 2,000 cubic inches of prepared zinc, and this zinc space would thus be 15-in. wide by 15-in. deep by 9-in. long (in the direction of the length of the extractor). The baffle boards are so arranged that the solutions always flow through the zinc in an upward direction. This ensures greater regularity of motion, assists in the disengagement of the hydrogen gas given off, and also increases the precipitation of the gold in the lower portion of the extractor.

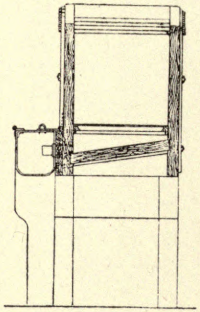
The box may be built with a slight fall in the direction of its length. Though there are ten zinc compartments, usually six only are filled with zinc—the first being used as a settler to allow of the disposition of any matter coming away from the percolators with the solutions. The next six compartments are filled, and the three remaining act as settlers for any fine gold precipitate accidentally carried over from the last zinc compartment; or when the solutions are made up to their normal strength in the sumps, lumps of cyanide are placed in the last compartment of the "strong" extractor box, which is thus converted into a dissolver. After treatment in the extractors, the solutions are conducted into their respective sumps.

Each compartment is fitted with a tray, usually of iron wire gauze of about $\frac{1}{8}$-in. mesh, on which the zinc rests, and through which the gold precipitate passes into the space provided for it below. At the lowest part of the space is fitted a plug, by the withdrawal of which the gold slimes flow out of the extractor into the cleaning-up launder (as will be described under "cleaning-up"). These plugs are protected from interference by an iron launder, the lid of which is fastened with Brahma locks by the Works Manager, and the top of the extractor is covered with a locked lid, the keys of which are kept by the Works Foreman, whose duty it is to replace fresh zinc each morning.

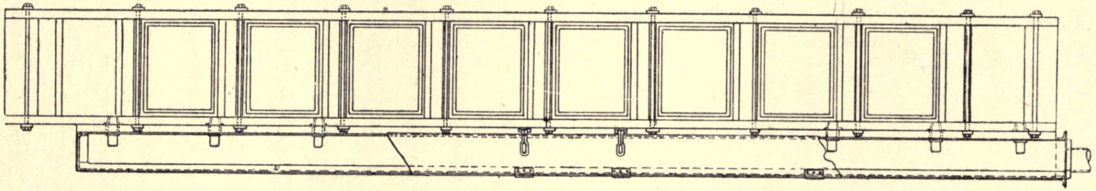
Unprotected iron should be most carefully avoided in extractors,



ELEVATION. (PART SEC.)



SEC^{L} END ELEVATION.



PLAN. (PART SEC.)

EXTRACTOR BOX.

These drawings are to scale, but as dimensions vary with the capacity of the box required, these must be obtained from the text.

especially when it can come into contact with the zinc, as the two metals form a strong galvanic couple, the effect of which is to send the zinc into solution and to gold-plate the iron. Serious losses have been caused by this. Porcelain trays have been suggested instead of iron wire gauze, but are too fragile and clumsy, and at present the iron wire still holds the field.

Recently, however, boxes have been placed on the market made of enamelled iron, and as they have all the advantages of metal boxes without its serious disadvantage, due to galvanic action, they are strongly recommended.

An extractor as above for a 2,000-ton plant would be about 12-ft. long, 18-in. wide, and 2-ft. 6-in. high outside measurements. The sides and ends and bottoms are made of clear yellow pine boards, 1½-in. thick, the baffle boards being 1-in. The centre and two intermediate baffle boards are usually made of 1½-in. wood, to give greater stability to the sides. All baffles are gained ¼-in. into the sides, which are kept together, and to the ends and bottom, by bolts.

There are usually three extractors in each plant, one for strong one for weak solutions, and one for very dilute washings, or surplus solutions leaving the works. It should be a rule that no solutions should leave the works—sluiced out residues excepted—without first passing through an extractor. Attention to this precaution has saved many ounces of gold which would otherwise have been wasted.

In addition to agitators, percolators, reservoirs and extractors, there are sundry accessories and fittings which require explanation.

Vacuum Apparatus.—This is most useful in dealing with agitator products, dry ore, concentrates, and tailings that do not percolate freely. It comprises: (1) The vacuum producer, which may be a plunger vacuum pump or a Meldrum steam ejector. The latter is an extremely useful appliance, and will give a good vacuum with great rapidity without any other machinery whatever, but has the demerit of consuming a considerable amount of steam. Where power can be obtained from a shaft, a vacuum pump will probably give greater satisfaction. It can be regulated to any speed up to 150 strokes a minute. A 6-in. vacuum pump requires about 2½ horse-power. (2) The receiver, which is boiler-shape, is made of iron or steel plate, and should be varnished inside. Such a chamber, 3-ft. in diameter by 13-ft. long, holding about 4 tons, is of sufficient capacity for a 2,000-ton tailing plant, where the strong solutions are allowed to percolate unaided; otherwise two may be used, being one each for strong and weak solutions respectively. It is fitted with vacuum and water gauges, and with a discharge into the two vacuum tanks previously described under "reservoirs." It is connected with the exhauster by 2-in. air pipe and by 3-in. solution pipe to the main line of piping leading to the percolators.

Pump.—This is required to elevate the solutions from the sumps to the upper reservoir, or to the percolators. A centrifugal works well, and is very convenient; whilst the special steam pump, with steel pistons and rods, has the advantage of requiring no shafting or pulley connections.

Lathe.—This, in the earlier plants, was an ordinary 6-in. wood-turner's lathe, on which the zinc shavings were turned by hand. Machine tools have been now successfully applied for this purpose. One of the best known of these is Liddell's Patent, by which as much as 130 lbs. of shavings may be cut per day of 10 hours, for 2 horse-power.

Water Supply.—This should be led to the cyanide dissolver, upper reservoir, to each percolator and agitator (if any), to the extractors for cleaning-up purposes, and to the assay laboratory and melting department. The main service supplying each percolator and the upper reservoir should be not less than 3-in. piping, except with small plants, while the supply to the extractors, dissolver, and laboratory may be branch services with 1-in. pipes and cocks.

Pipe Lines for Solutions.—There are three systems of these—

1. Solution pipes—a 3-in. service from the upper reservoir to the agitators and percolators.

2. A direct service from each percolator to the distributor, which is

connected to the extractors and also to vacuum receiver, if this is employed. This service is of from 1-in. to 2-in. pipe, according to the size of the vats employed ; the smaller size pipe is suitable for vats holding up to 40 tons, the larger for vats of upwards of 400 tons capacity.

3. This system is that connected with the pumps, and runs from the sumps to the upper reservoir, or direct to the percolators, and may also be connected to the other two systems, so that all systems communicate at will. Frequently the fresh water service and the solution service have their main pipe line in common, but this method cannot well be adopted in plants where the residues are sluiced.

Distributors.—These are made in two forms, the longitudinal and the compartment. The former, which was introduced at the "Waihi" tailings works (*see* p. 15), is much more simple and easy to manage. The latter is more compact and is not liable to the loss of solution caused by the misplacing or springing out of the rubber hose used with the longitudinal form.

The *longitudinal form* is practically three or more wooden launders side by side, built perfectly tight—all leakage is loss of *gold*—and communicating each with a different extractor box. The solution mains from the vats end in a hose which may be set to discharge into either of the three launders, and thus into the proper extractor box.

The *compartment form* has as many compartments as there are vats, and the mains from each vat discharge each into a separate compartment. Connections stopped by plugs lead to the mains to the various extractor boxes. By merely pulling out a plug, communication is at once made with any extractor required.

With this form the loss caused by the tendency of the hose to spring out of the compartment, and perhaps to discharge over the side, is obviated, as all solution must fall by gravitation into the compartment reserved for that vat. But it is difficult to prevent leakage in the plugs and pipework, and the longitudinal form will usually be found more convenient in practice.

These distributors have done away with the network of pipe connections and valves formerly employed to enable the operator to divert the flow of solutions into one or other of the extractor boxes, and they have the great further advantage of enabling the work of each vat to be always under the eye of the operator, who can thus tell at a moment if the filters are behaving properly, if the leaching rate is satisfactory, or if the vat is choked. And the operator is also able to instantly take a test of the solution without the trouble formerly occasioned to detach the flow of one particular vat from the rest of the pipe system in order to obtain a sample.

Residues Launder.—This conveys the sluiced residues from the percolators. It should be strongly made, have a fall of from $\frac{1}{2}$-in. to 1$\frac{1}{2}$-in. in a yard, varying with the fineness of the material treated ; thus, pan tailings, or fine dry-crushed ore, take $\frac{1}{2}$-in. in a yard ; ordinary 1,000-mesh tailings take 1-in. in a yard ; concentrates from spitzlutten 1$\frac{1}{2}$-in. in a yard. For a 2,000 to a 4,000-ton plant where one man can do all the sluicing required, this launder may be 18-in. wide by 12-in. deep. Wooden riffles are sometimes placed at the end of the residues launder to catch any amalgam—the product of previous treatment with mercury—or even coarse gold, which may have escaped treatment by cyanide.

A Melting Furnace will be required capable of taking a No. 50 or 70 plumbago crucible, and an assay laboratory is also a necessity. Carr's double melting furnace, to take two No. 60 crucibles, is largely used, but reverberatory patterns are preferable for large installations.

Power.—For an ordinary percolation plant the only power required is that for pumping solutions, cutting zinc, and producing a vacuum. If water is available, a pelton or turbine will be most useful, and always ready for immediate service : 5 horse-power should be sufficient for a plant up to 3,000 tons a month.

Filter-Press.—A special filter-press, with plates of 12-in. to 24-in. square, is now used in most of the large works for cleaning-up purposes. This press has its plates machined at the edges to prevent leakage of gold slimes. By the use of this apparatus the clean-up is greatly simplified, as by leading the suction pipe of flexible hose into a compartment of the extractor box all the precipitate is collected in the filter-press without any handling whatever, and by placing thin filter papers *over* the cloths a dry cake is obtained, which can at once be placed on the drying or roasting tray, or into the acid-treatment vat, without the precipitate being in any way handled, and there is also no loss through the filter-cloths becoming impregnated with the fine gold slime; the thin filter papers are of course burnt, and the ashes added to the melting charge.

(II) Large Plants for Double Treatment and for Slimes.

The usual practice is to divide the percolators into two sets, the one superimposed on the other, as shown in the illustration. The pulp from the mill is elevated in a tailings wheel to spitzlutten, in which the concentrates are separated for special treatment; it then passes through spitzkasten, which eliminate the desired proportion of the slimes. The pulp then passes through a revolving radial arm distributor, or through a hose, into the upper set of vats; these are provided with sluice gates of adjustable level (*see* page 40), or an overflow launder, through which the water carries off the remaining slimes, leaving the sands in the vat.

As soon as the vat is filled to within about 6 inches of the rim, the contents are allowed to drain dry, and the first treatment with cyanide solution is given, and after this has been drawn off, the discharge doors in the bottom of the vat are opened and the contents discharged into the lower vat, in which the final cyanide treatment is given. By this means the ore is broken up,—and any settled layers of slimy material prevented from interfering with the extraction,—better contact between the particles of ore, solution and air is ensured, and the extraction appreciably improved. Solutions from sulphide ores are also oxygenated by this handling.

The residues after treatment are discharged through the discharge doors into trucks, which run below the vats, and are thus conveyed to the tailings heap.

The concentrates are similarly treated, but for a longer time.

The slimes are settled in spitzkasten, agitated in vats fitted with agitation and circulation apparatus, and washed by decantation.

Tailings Wheel.—This may be built either of wood, steel, or wood and steel. Wooden wheels are very serviceable, but are very expensive. Steel wheels are very light, but are liable to excessive wear. Composite wheels, with steel or iron centres and with wooden buckets and arms, have been put on the market, and are most used.

These wheels are useful for lifts of from 10 to 40 feet. For a greater height than this, pumps or a series of wheels should be employed, as the tendency of the tailings to pack in the buckets becomes very marked as the height increases.

Each wheel must be accurately designed for the speed of revolution and height of lift required, as if the pitch of the bucket is incorrect the tailings will pack, or the delivery may be too early or too late. A well designed wheel should give an effective lift of over 80 °/₀ of its diameter. These wheels are run at very slow speeds, the larger sizes making only 2½ to 4 revolutions per minute, and are run by belting in preference to spur wheel or sprocket gearing.

Spitzkasten and Spitzlutten.—These have to be designed for each particular ore, theoretical rules having been discarded as useless in practice.

Vats.—These are built of $\frac{3}{16}$-in. to $\frac{5}{16}$-in. mild steel plate, single riveted, with angle irons at rim and base. A not unusual size is 40-ft. diameter by 10-ft. high. Such a vat would hold 400 "short" tons of tailings at a charge.

The upper angle irons are almost invariably placed outside the rim, but in the latest Johannesburg plants they are being placed inside. The lower angle irons are inside in sumps and reservoirs, and frequently outside in leaching vats. Placing angles inside enables supports to be considerably nearer each other, and thus reduces the weight of the steel joists necessary to carry the upper tanks; on the outer line of joists the saving in weight may be very considerable.

Owing to the inconvenience caused by the presence of the supports within the lower vats, necessitating an extra strain on the bottom plates, or the cutting away of these and the building-up of a tube or other water-tight joint around the support, with consequent loss of space in the lower vat, attempts have been made to support the upper vats by girders without intermediate pillars, and also by using spherical bottom upper tanks supported from the periphery only. The former method, however, requires heavier supports and wastes height, an important matter where tailings wheels are concerned, and the latter was not found to be an economical method of procedure.

Conical vats are built for slimes treatment to assist in directing the flow of settled slimes to the central exit, and in these the bottom plates are usually thicker than in the flat sands vats.

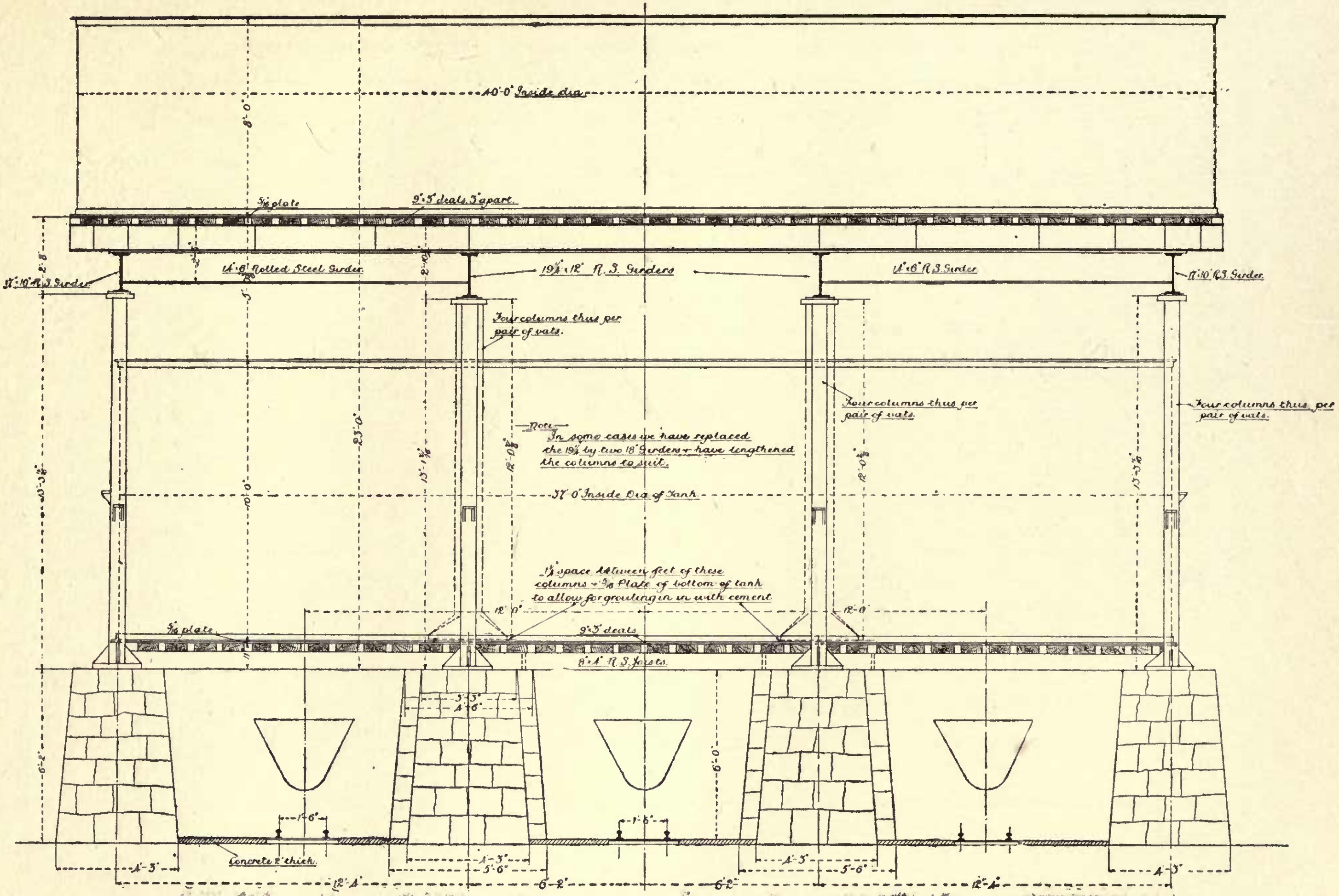
The author has been recently engaged on the largest slimes plant in the world for a Johannesburg Company, and the vats are 30-ft. × 8-ft. deep at the side and 9¼-ft. at the centre. The centre bottom plate is ½-in. thick, as against ¼-in. for the sides.

The upper vats usually rest on a staging built up of iron or steel columns and steel joists, but wood is employed in many instances for this purpose. The lower vats rest on masonry or cast iron pillars, with wood or steel joists intervening. It is best to have 9 × 3 deals placed immediately under the vats as intermediate support.

Direct Treatment.—When ores are treated direct, drying furnaces are usually necessary, and in special cases roasting furnaces may be employed.

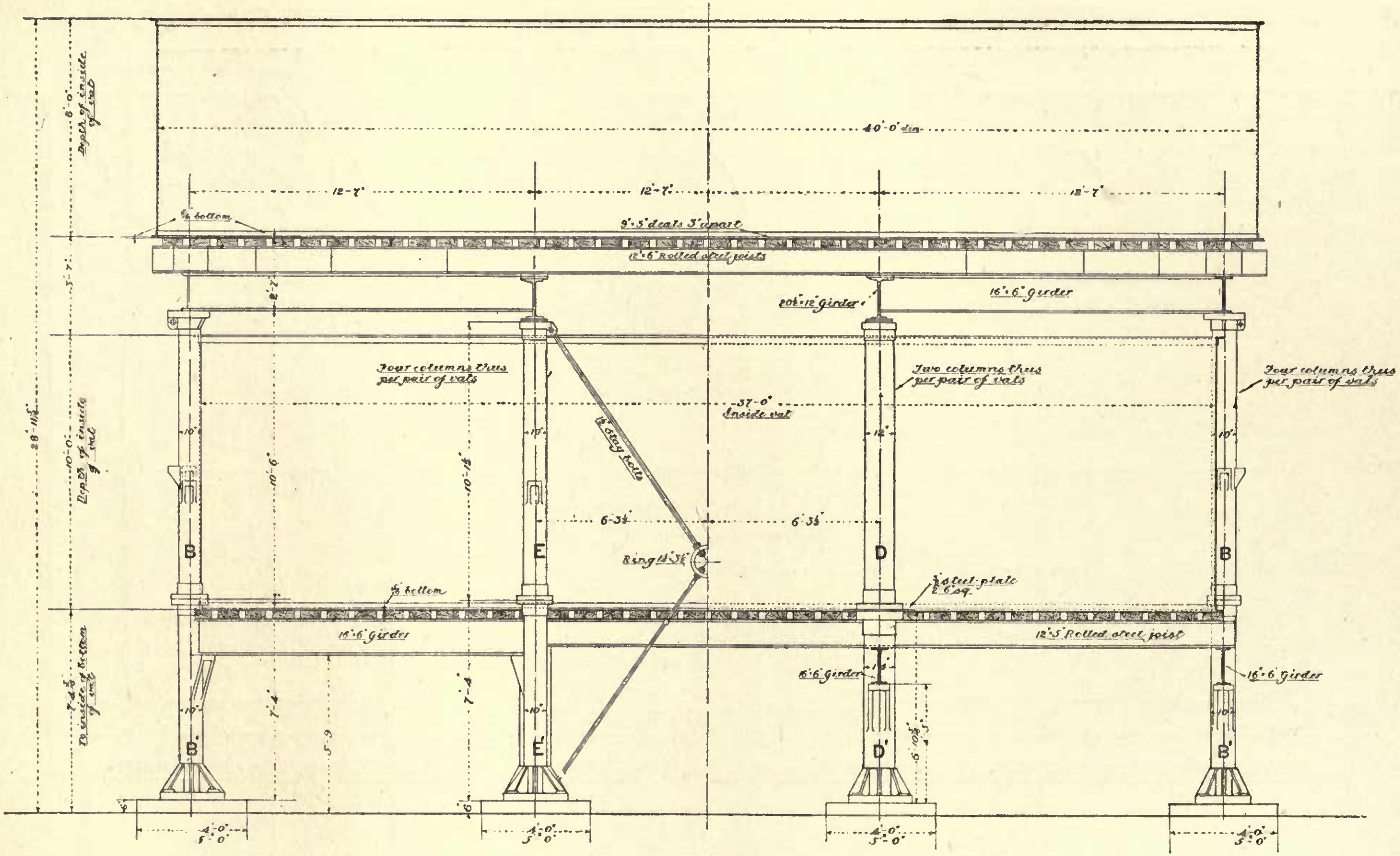
Drying Furnaces.—Kilns were formerly employed. These are apt to overheat the ore, and to thus deleteriously affect the extraction of the precious metals. "Step" roasters have been used in New Zealand, but are expensive, and are now superseded.

Rotary dryers are much used, and in a well-designed furnace the heat and admission of air are perfectly under control, and a maximum of work



End Elevation of iron & steel supports on masonry walls of large double treatment plant

South African Practice, 1,000-ton-a-day Plant.



—Half Elevation—

—Half Sectional Elevation—

—Steel & iron supports without masonry walls for large, double treatment plant.—

Most recent South African Practice—now in course of erection—1,000-ton-a-day Plant.

done at a low cost. The *Argall Dryer* has proved itself a most economical furnace, whether for drying or calcining ores.

Roasting Furnaces.—These are frequently employed when the ore contains tellurides, sulphides of silver, or copper in an attackable condition. An ill-designed furnace will lose from 5 to 10 °/₀ of the value of the ore, and this may be largely increased in chlorodising roasting. The furnace should be long, should permit of the heat being gradually applied to the ore, and should not be liable to make dust. There are furnaces on the market which succeed in roasting rich telluride ores for a loss of less than 2½ °/₀ of the value.

SPECIAL NOTES ON THE ARRANGEMENT OF CYANIDE PLANTS.

Percolators.—If the site will permit of the percolator vats being placed almost directly on the ground, instead of on raised foundations, a saving in erection costs of possibly some hundreds of pounds will result. This method is possible where the slope of the ground enables the tramways to be excavated under the vats, and to come out into the open at a convenient distance not too far from the percolators.

Filter Bottoms.—In arranging the filter bottoms, when jute and cocoanut mats are both used, it will be noted that the jute are placed beneath the cocoa circles, which serve to protect the jute from wear and from being cut by the shovels. The jute circles are made twelve inches larger than the diameter of the vat, with unbound edges, and are to be caulked with rope between the wooden ring and the side of the vats. Holes for discharge doors are cut in the jute circles to a radius about six inches less than that of the combing of the door. This allows for caulking-in between the combing and the filter ring. The same procedure is carried out where columns are placed inside the vats.

Side Sluice Gates.—When revolving distributors are supplied, the vats have either overflow launders or side-discharge sluice gates. The latter have wooden slats placed in slots as the level of the sands rises in the vat. A roller blind ot waterproof cloth is frequently affixed to the inside of the slats. This blind is fastened at its under edge at the bottom of the vat, and hangs temporarily on the upper slat for the time being. As the level of the slats rises, so the blind is unrolled, and thus acts as a water-tight covering during the percolation, and prevents cyanide solutions from finding their way to the discharge gate. The same object is also obtained by having rubber sheeting fixed at the lower inner edge of each slat, so that the pressure of the sands makes a tight joint at the horizontal junction of the two slats. The most recent form of side sluice is a roller blind working against a wooden open trellis work; the necessity for the addition of slats as the level rises is thus obviated. It is sufficient to unroll the blind.

Discharge Doors.—In emptying the vats, sheet iron flaps are sometimes placed below the discharge doors, so that the men in the vats may keep on shovelling during the shifting of the trucks. On opening the flaps, a truck load is at once shot into the car below. The doors should be hooked up, out of the way of the discharge.

Vacuum Pump.—This has usually a double connection to the vacuum chamber, the upper to exhaust the air, and the lower to empty the chamber of solutions and to send them direct to the distributor box.

GENERAL INSTRUCTIONS AND SUGGESTIONS.

Centrifugal Pumps.—The pump should be firmly bolted down to a wood or brick foundation, perfectly level with the pump spindle, and in a line with the driving pulley. The driving belt should be of good length and width, as a tight one will throw extra strain on the spindle and entail loss of power in friction, besides wearing out much sooner. In most cases a foot valve is necessary to retain the water, as these pumps require charging before starting. This, when once done, seldom requires repeating. Belts should not be vertical, but as near horizontal as possible, and not exceeding 45°.

After having fixed the pump in the most convenient position, the suction pipes and foot valve should be connected with indiarubber rings between the flanges, it being of the utmost importance to have all joints *air-tight*, as a small leak will impair the efficiency of the pump. The whole of the foot valve and grating should be well immersed in the liquid so as to ensure a good supply. The pump and pipes must then be charged with water and allowed to remain some time to detect any leakage. If satisfactory, the pipes can be connected, care being taken to avoid any unnecessary bends, as owing to the great velocity of the water they entail great loss in friction, but, if absolutely required, bends of a large radius should be used.

For lifts up to 40 feet it is desirable to place the pumps mid-way between the suction and delivery, but the larger sizes can, if required, be made to draw 28 feet vertical from centre of water in well to centre of pump spindle.

If the suction pipes should run any great distance horizontally, they must be placed with a gentle rise towards the pump, *so as to prevent any accumulation of air.* If this is not possible, an air cock must be placed in the highest point to allow it to escape.

Before starting, the glands should be attended to, care being taken that they are not screwed too tight, that the lubricators are charged, and the bearings free from dirt, and well oiled. If this is done they will give no trouble.

The power should be applied gradually.

If the centrifugal pump does not start well, water may be run in by turning on the cock from a percolator, solution tank, or fresh water main. The pump, when full, will at once draw.

Extractor Boxes.—Fresh zinc shavings should be added to the extractor boxes every morning, and carefully packed at all corners by tamping down with a stick, to prevent channels being formed through which solutions may pass without coming into effective contact with the zinc shavings. The compartments should be kept full of zinc.

INSTRUCTIONS FOR THE ERECTION OF STEEL VATS.

The bottom plates should first be raised on the framework supporting the vats, packing (old oil drums are very convenient) being first placed underneath them so as to raise the bottom of the vat about 2 feet clear of the framework, and thus allow room to put in and hold up rivets in the bottom of the vat. The curb angle should then be bolted on round the edge, and the whole of the bottom riveted up, the rivets being passed through from the outside. Do not start riveting until the whole of the bottom plates have been properly bolted together in position. After riveting up the bottom, lower it on to its permanent position on the framework.

Then the side plates, with the upper angle iron and stiffening piece (if there is one), must be bolted in place and riveted up, the rivets being passed from the outside and riveted over on the inside.

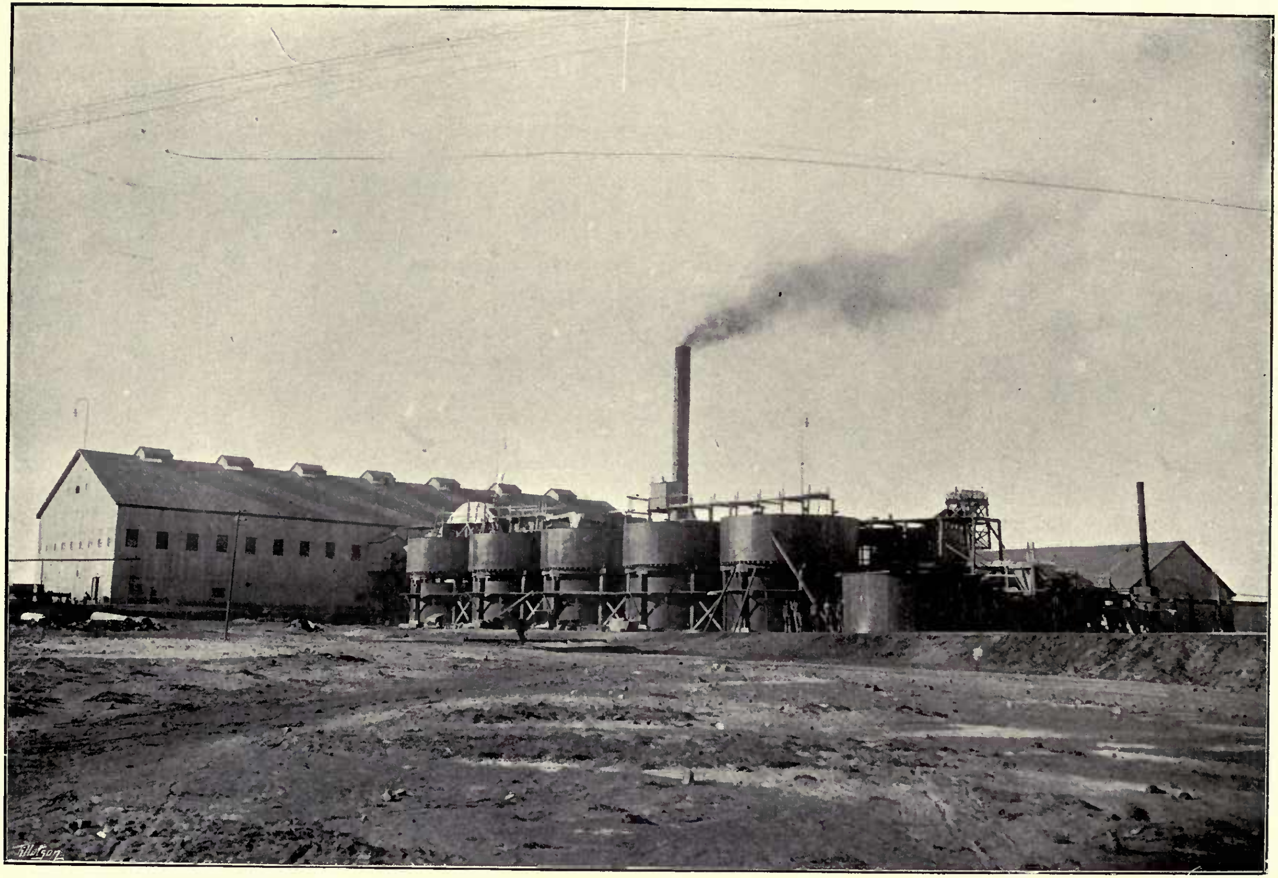
Then the edges of the bottom angles and the cross laps of the plate seams must be chipped, fullered (or trimmed) and caulked inside if necessary.

The tightness of the vat should be tested by charging it with water, and any leak taken up by caulking.

Care must be taken that the surfaces of the plates at the joints are scraped quite clean ; a coat of white lead may be given to the joints, but no packing should be used.

The service bolts used to tack the plates together before riveting should be about 3 feet apart and well tightened.

The riveting must be done hot, and the rivet heads should be hammered over and not snapped, as this method spreads the body of the rivet more, and ensures a much tighter joint.



THE FIRST STEEL DOUBLE-TREATMENT PLANT.
The Geldenhuis Deep G. M. Co., Ltd., Johannesburg.

CHAPTER III.

Extraction.

NOTE.—*Special attention is invited to "Pointers" on page* 60.

IN the earlier history of the process it was necessary to take material for treatment in the condition in which it was delivered to the works, but now that complete equipment is being laid down at mines, in the designing of which lixiviation by cyanide has been kept in view as the ultimate treatment, it may not be out of place to discuss the effect of the present systems of milling on the material to be leached.

For hard ores, containing free gold:—Where water is available, wet stamping with amalgamation and cyanide treatment of tailings is the usual practice; if sufficient water is not available, dry crushing by stamps or ball mills, not by rolls, cyaniding the pulverised material direct. The portion which contains the coarse gold is retained by the screens, and may be amalgamated in a pan, or the heavier gold may be separated by an automatic gravity separator.

On the other hand, in the case of another typical ore, soft, liable to make much slime, oxidised, and containing fine gold, such as those occurring in parts of Western Australia—this ore would probably be best treated dry. Stamps are discarded in favour of ball mills, any coarse gold present is retained by the screens, or mechanically separated, and the dry material is leached by cyanide. Even if an abundance of water is available, this ore should still be crushed dry to avoid the floating away of slimes, and consequent loss of gold; but in this case the residues may be run over amalgamated plates or mercury riffles after leaving the vats, if it is found that any very coarse gold is present.

Dry Crushing, as applied to gold ores for cyanide lixiviation, has come into prominent notice and the results gained have attracted considerable attention. Extractions are usually very high, whilst working costs are low. With a very hard ore, 93 °/₀ to 96 °/₀ of the gold contents of the ore are being actually obtained as a result of "dry" treatment, as against an extraction of under 90 °/₀ for the *combined* results of pan amalgamation, and subsequent cyanide treatment, of the tailings as previously carried out, whilst the cyanide consumption in both methods is practically the same.

As to the relative advantages for this purpose of stamps, rolls and ball mills, it is found that on hard ores stamps or ball mills are necessary; for softer ones, rolls and ball mills deliver much more granular products with less slime. On a hard ore it was found that material crushed by stamps to 30-mesh screens gave a better extraction than a 60-mesh ball mill product.

If wet crushing and preliminary amalgamation are employed, means have to be adopted to get the pulp into the percolators in a leachable condition. If the tailings are run into the vats direct, they either pack, or, if slimes are present, they cake, and render percolation uneven and ineffective. This difficulty does not arise when the tailings have been already settled in dams. Two methods are in use—one is to run the pulp into an upper vat with revolving distributing feeders of unequal lengths: this is kept full of water, so that the slimes may overflow, whilst the coarser pulp separates out and settles until the buddle or collecting vat is filled

The other method is to send the pulp through spitzkasten or spitzlutten, and thus to get rid of the slimes, the pulp running into upper vats, where it receives preliminary treatment, and is then discharged into the lower percolators for subsequent treatment.

These "slimes" are laid aside for after-treatment by decantation (*see* Chap. IV), or other method. Further concentration has been tried on them with successful results, but experiments prove that a very considerable amount of gold still remains in the very finest portions. This difficulty is obviated by dry crushing.

Before dealing with the details of lixiviation practice, it will, perhaps, be advisable to glance at the general routine of the process as practised at small plants. The material to be treated having been sampled, it is charged into the percolators, and is leached if necessary by water, which is allowed to rapidly drain off; then an alkaline washing if necessary, then two or more washings of various strengths of cyanide solution, and finally a water washing to get rid of the last portion of the cyanide. The residues are now sampled and discharged. The solutions pass from the percolators to the respective extractors, where they are freed from their gold contents and flow into the sumps; thence the strong solutions, fortified up to their normal strength by addition of strong cyanide solution, are pumped on to fresh charges, preceded or followed by weak solutions or washings, as arranged. The gold is cleaned up from the extractors fortnightly or monthly, dried, and melted.

Sampling.—This, in the treatment of tailings dumps, is effected by drawing with a sampling iron a small portion of the contents from each truck as it enters or leaves the building. The portion thus withdrawn is deposited in a locked box, and when the percolator is charged or emptied the assayer comes and quarters down his sample. If the residues after treatment are to be sluiced they are sampled as they lie in the percolators by probing the mass with the sampling iron, care being taken to prevent an undue proportion of the upper layers from being included. It is also usual to sample the lowest 6-in. to 18-in. separately.

In double-treatment plants the sampling is effected by an automatic sampler working in the launder feeding the upper vats.

Charging the Percolators.—If dry crushed ore is to be treated, this is usually made to travel on a belt conveyor from the battery to the ore hopper, in the leaching house, and is run from thence by another belt, or is discharged into trucks, which take it on to a "traveller" working over the vats. By this means a truck can be emptied on to any portion of the vat, which is thus charged evenly. The truck is covered with an old filter-cloth, and is dumped on to a "spreader" beneath the traveller, so that all shock and consequent dust caused by the impact of a body of dry ore on the contents of the vat is avoided.

Tailings are usually dumped into the percolators from side-tipping trucks, running on two or more lines of rails overhead. It is important to arrange that no part of the tramway comes into contact with the vats, as the vibration would impair the joints of the latter and thus cause leakage.

Where large quantities of tailings are to be treated, wire ropeways, or belt conveyors can be utilised to convey the material from the dump to the vats, or a travelling rope can be readily arranged to deliver loaded trucks continuously and to return empties. This is the method usually employed at Johannesburg for dumping residues. Where labour is scarce, excavators or steam shovels are employed in large works for filling the trucks.

As soon as the requisite number of truck loads has been dumped into the percolator, the material is levelled down and the first wash run on.

Fresh Water and Alkaline Washes.—It is important not to attempt to run solutions into an empty percolator from the bottom, unless weights have been distributed over the filter bottom to prevent it from "ballooning."

It is necessary, when the preliminary tests indicate the need for such treatment, to run on a water wash to the charge to free it from acid salts as fully as is economically possible. This water wash should be added from *below* the filter bottom; it thus rises through the charge and passes over the side of the vat into a launder arranged for this purpose. It is then succeeded by an alkaline wash of either soda or lime, also added from below. By this means any acid liquor is prevented from contaminating the pipes and bottoms of the vats, which are thus always kept alkaline or neutral, and the formation of Prussian blue, for instance, is avoided.

As to the relative advantages of caustic soda and lime, it will be found that a judicious discrimination in the use of these two alkaline agents will enable the solution to be preserved in an effective condition. A certain amount of alkalinity is advisable for successful economical working, as will be shown more especially in considering the reactions occurring in the extractor boxes; whilst, on the other hand, an excess is to be carefully avoided. Lime forms insoluble compounds, and by employing this reagent, when the solutions are too alkaline, and soda, when increased alkalinity would be of advantage, the balance may be thus assisted, and the solvent power of solutions maintained. Dilute solutions of lime do not appear to be more destructive of cyanide than those of soda.

The alkaline washing is now completed, and cyanide solution may be run on, unless an intermediate water wash is found advisable to carry off the residue of alkali and alkaline products resulting from the previous wash.

Cyanide Solution Treatment.—In some works sufficient "weak" solution—about 10 °/₀ of the amount of tailings in the percolator—is now pumped up from the sump on to the tailings, with the object of displacing the water left from the previous washing, and thus to prepare the way for the strong solution; this preliminary wash of weak solution is frequently omitted, but as a method of reducing cyanide consumption it is to be recommended.

The strong solution may contain from 0·1 to 0·5 °/₀ KCy, and is usually about 33 °/₀ of the amount of ore in the charge. To prepare it, the foreman ascertains the strength of the solution in the strong sump, and also that of the liquor in the cyanide dissolver. He then runs in a calculated amount of liquor from the dissolver: this liquor is thus intimately mixed with the bulk by the agitation consequent on the pumping.

The amount of cyanide liquor required is calculated as follows:—

"Multiply the tons of solution taken by the difference between its present strength in cyanide and the normal strength required, and divide by the strength of the strong cyanide liquor less the strength of the normal solution required."

Thus, let T equal tons of solution in the sump, or reservoirs.

A „ present strength in cyanide of the sump solution.

B „ the normal or required strength to which T is to be brought.

C „ strength of liquor in cyanide dissolver.

Then $\frac{T\,(B-A)}{C-B}$ amount in tons of liquor to be added.

If it is required to make a solution of 30 tons of ·5 % KCN, the strength of solution in the strong sump being ·35 %, and of the liquor in the cyanide dissolver 20 %, then—

$\frac{30\,(\cdot 5-\cdot 35)}{20-\cdot 5}=\frac{30\,(\cdot 15)}{19\cdot 5}=\frac{4\cdot 5}{19\cdot 5}=\cdot 231$ of a ton, or 46·2 gallons (2,000 lbs. per ton).

The solution now in the reservoir will be 46·2 gallons more than the 30 tons, but this slight excess is immaterial, the strength in cyanide of the total amount being correct.

The correct formula is :—

x = amount of strong solution required.
y = amount of strong liquor.

$$x+y=30 \text{ tons and } \frac{\cdot 35x+20y}{x+y}=\cdot 5 \;\therefore\; \begin{array}{l} x = 29\cdot 771- \\ y = \quad \cdot 229+ \\ \hline \quad\;\; 30\cdot 000 \end{array} \text{tons.}$$

The strong solution is now run on to the top of the charge, and the cocks or clamps on the exit pipe are so adjusted that the solution may flow evenly during the whole 24 hours or so of contact. Usually it is found to be of advantage to allow solutions to percolate more rapidly, and to pump them back on to the top of the charge, and thus to "circulate" them. Examples of this practice are given below.

When the ores contain much silver, or if concentrates are being treated, it is occasionally found advantageous to increase the amount of solution from a 1/3 to 1/2, or even 1/1 of the ore; but generally it is found that increased proportions of solution show an increased consumption of cyanide.

Vacuum suction is not applied to the strong solution percolation, except in very stubborn cases and also where agitators have been used, as it is found the percolation rate has usually to be retarded rather than accelerated In wet weather, however, when tailings are charged into the vats in a sloppy condition, percolation is retarded, and suction may be required. Under these conditions it is found to be beneficial, where practicable, to allow longer time for percolation and for the treatment generally, or the extraction percentage may be temporarily lowered.

The "weak" solution is run on as soon as the top of the charge appears to be dry. Sometimes the ore is allowed to drain quite dry, to permit of access of air. In percolating concentrates, the turning over the charge by hand, due to the discharge from the upper to the lower vat, has a distinctly beneficial effect.

Various patents have been taken out for the addition of oxydising agents to the solution, with the object of accelerating the rate of solubility of the gold; but these additions are usually ineffective in practice. The solution itself contains more than sufficient oxygen for the reaction, as does also the charge to be treated, and, moreover, ferricyanide of potassium is also frequently present, so that any such addition is quite unnecessary.

Some years since the use of haloid salts of cyanogen—the bromide chloride, and iodide—was proposed by Dr. Gaze, and more recently the suggestion has been warmly propounded by Messrs. Sulman and Teed, who evolved a practical process utilising the reactions between bromo-cyanide, gold, and potassium cyanide. This method is successful on metallic gold and is the basis of a most successful method for the treatment of auriferous tellurides without roasting (*see* chapter on the Treatment of Sulpho-Telluride Ores). But with ordinary ores there is a general consensus of experience that there is an increased expenditure on account of greater consumption of cyanogen with no corresponding advantage in extraction.

A great advantage of the process, however, is that with certain ores, such as marcasite and arsenical sulphides, the more rapid extraction may even show a lessened consumption of cyanide owing to the briefer contact of the alkaline solution with the sulphides and the consequent lessened formation of alkaline sulphide, but this is a matter for expert determination, and as a general principle it may be taken that with ordinary tailings the simpler cyanide process will give the most profitable results.

Strong solutions are of very complex nature, and become more so with each additional treatment. This is not wholly of disadvantage, as will be shown hereafter; but they should be occasionally tested for alkalinity, and test extractions should also be made from time to time, using "works" solution against freshly made cyanide solution, and the difference in extractions and consumptions noted (*see Pointer No.* 4).

The "weak" solution is usually from 0·02 to 0·2 °/₀, and the quantity taken varies from one-half to double the amount of the strong solution used. The amount required is determined by preliminary investigation; formerly, it was seldom more than the strong solution, and was regarded rather as a washing to free the charge of as much strong solution as possible before the final water washing, but it is now considered to play an important part in the solution of the gold, and receives more attention accordingly.

The strengths of strong and weak solutions given here are those most used in actual practice; it must, however, be clearly understood that "strong" solutions of under 0·1 °/₀ may be effectively employed, followed by corresponding dilute "weak" solutions, but the increased time required to obtain an efficient extraction by this means has resulted in the general adoption of the more rapidly effectual strengths given above.

An increased liability to decomposition, and consequent reprecipitation, of the gold has been noticed in one case where extremely dilute solutions have been used. This, however, has not been generally proved, and the subject is one which should receive increased attention and investigation.

Final Water Washings.—These are added to the top of the charge and allowed to percolate as rapidly as is possible by the aid of the vacuum apparatus. The amount of water taken varies with the quantity of solutions circulating in the works. If the works are full, the last washing is usually given with weak solution. This practice is not to be followed, as it causes both a loss of gold and of cyanide, which leaves the works in the "moisture," amounting to 15 °/₀ to 20 °/₀, contained in the residues when discharged. It is desirable to have an extra extractor through which the water washings leaving the works may be run. This saves the gold and permits the stronger portion of the cyanide solutions coming off to be also retained.

A water washing should be about one-third of the weight of the charge to do its work effectively, but in practice, in order to maintain the "balance" of solutions, it is frequently only $\frac{1}{20}$th or $\frac{1}{10}$th.

The following are some of the solution charges employed in South Africa on tailings and concentrates :—

(1)—Charge—450 tons of tailings.

First Treatment	15 tons	0·06 % sol.	24 hours.
	33 ,,	0·2 % ,,	
	33 ,,	0·1 % ,,	

The tank is allowed to drain dry, and is then discharged into another tank.

Second Treatment	100 tons	0·15 % sol.	4 days.
	250/350 ,,	0·07 % ,,	
	20 ,,	water ,,	

The tank is allowed to drain dry, and is then discharged.

Total time of treatment, 5 days. Cyanide consumption under $\frac{1}{2}$ lb. per ton treated. Extraction 75 %.

(2)—Charge—135 tons, tailings.

Solution :—	25 tons	at 0·02 %	18 hours.
	80 ,,	,, 0·1 %	3 days.
	25 ,,	,, 0·02 %	
	25 ,,	,, 0·01 %	

Vats drain and discharge.

Total time of treatment, 8 to 10 days. Cyanide consumption 0·3 lb. per ton. Extraction about 70 %.

(3)—Charge—190 tons, tailings.

Solution :—	30 tons	at 0·1 %	12 hours.
	60 ,,	,, 0·26 %	48 ,,
	30 ,,	,, 0·17 %	15 ,,
	30 ,,	,, 0·1 %	12 ,,
	30 ,,	,, water	12 ,,

Total time up to 6 days. Cyanide consumption, $\frac{1}{3}$ lb. per ton treated.

The average consumption of zinc shavings with these tailings is $\frac{1}{4}$ lb. per ton treated.

(4)—Charge—200 tons, concentrates.

Treatment for 25 to 30 days with solutions as in (1). Extraction, 90 %, Cyanide consumption, estimated at $1\frac{1}{2}$ lb. per ton.

Residues are now either sluiced out by means of a hose and nozzle with water, under a head of about 60 feet (though a pressure less than this will work satisfactorily), or they are discharged by hand into trucks, and dumped at any convenient site by the aid of a system of rope haulage.

Extractor Work.—The solutions coming away from the percolators are tested both for gold and for cyanide, and conducted by the distributor into the respective extractors. No. 1 is for strong solutions, No. 2 is for weak solutions, and No. 3 for washings running to waste. The first liquor coming off from moist charges, or when sluicing is carried out, is run through No. 3, as it consists practically of the displaced water, but gradually the solution shows small amounts of cyanide. It is now usually very strong in gold, and is run first into No. 2 extractor, and then into No. 1. Generally, solutions above ·15 % would pass into No. 1 extractor, from ·15 to ·05 % into No. 2 extractor; below this, through No. 3. At the start of operations the fresh zinc does not seem to act as effectively as is the case in regular work, and a white hydrated precipitate of cyanide of zinc is frequently formed when the solutions are very weak in cyanide. This precipitate disappears, usually, after a day or two, and may be prevented entirely by using stronger solutions at the start of operations. At first, there is not the same facility for that regeneration of the cyanide in the extractor which takes place in practice, but as the solutions become more complex, matters speedily readjust themselves; the zinc, too, becomes coated with gold in the form of fine black powder, thus forming a galvanic couple of such energy as to decompose water. This galvanic couple is apparently responsible for a large amount of the reactions taking place in the extractors.

Skey has given the following series as the electromotive order of metals in a solution of cyanide of potassium—strength not specified—proceeding from negative to positive:—

Carbon.
Platinum.
Iron.
Arsenic.
Antimony.
Mercury.
Lead.
Gold.
Silver.
Tin.
Copper.
Zinc.

This shows the necessity for avoiding the use of iron in extractors, as a zinc-iron couple is one of the strongest combinations possible, and would result in great decomposition of zinc and the precipitation of the gold on the ironwork.

Elsner's equation shows that:—

$$4KCy + 2Au + H_2O + O = 2KAuCy_2 + 2KHO \text{ ; and}$$
$$2KAuCy_2 + Zn = K_2ZnCy_4 + Au_2.$$

It is found in practice that for every ounce of gold recovered, about $\frac{1}{2}$ lb. of zinc goes into solution; and it has been generally accepted that this is due to the action of cyanide on zinc: thus—

$$4KCy + Zn + H_2O + O = K_2ZnCy_4 + 2KHO$$

or, in strong solutions, probably

$$4KCy + Zn + 2H_2O = K_2ZnCy_4 + 2KHO + H_2$$

and much has been written about the defects of zinc precipitation from the tremendous consumption of cyanide involved thereby. The writer, however, has not been able in practice to discover this enormous consumption of cyanide. On the contrary, many careful experiments with solutions of varying strengths have been made by placing continuous syphons in the first and last compartments of the extractors and then testing every few hours the samples thus continuously withdrawn, and the results show no appreciable difference between the strength of solutions in active cyanide when entering and leaving the extractors; the latter solutions (those treated) are not the weaker. He suggests that whilst there is probably a certain amount of reaction between the aurocyanide and zinc, the following are the reactions taking place:—

$$Zn + 2KHO = K_2ZnO_2 + H_2$$

and then as Feldtmann points out—

$$AuKCy_2 + H = KCy + HCy + Au,$$

the nascent hydrogen reacting on the aurocyanide to regenerate the cyanide and precipitate the gold. Moreover, if any gold is present, as auricyanic acid, it is precipitated by the same means:—

$$AuHCy_4 + KHO = AuKCy_4 + H_2O$$

and—

$$AuKCy_4 + 3H = KCy + 3HCy + Au,$$

so that the precipitation is actually more perfect than that afforded by zinc alone, which will not effectually precipitate the gold from a solution of auripotassic cyanide.

That the reaction is really one between the caustic soda and the zinc rather than between the cyanide and zinc is corroborated in a very striking manner by a consideration of the behaviour in the extractors of copper when this metal is present in the ordinary dilute works solution. It is found that the copper is precipitated at such a rate as to coat the zinc and to hinder the precipitation thereon of the gold, but that an addition of cyanide to the solution so as to strengthen it *before* it enters the extractors quite overcomes this difficulty. It is then found that the copper remains in solution in consequence of the addition of cyanide, the gold only being precipitated, and this action may be accounted for by a consideration of the electro-motive questions involved.

The difference in the E.M.F. of a zinc-copper couple in solutions of

KHO and KCy is very marked; in KHO, copper and gold are very negative to zinc and to almost the same extent, and therefore the two metals would be precipitated together from the solution roughly in proportion to the comparative amounts present, and the copper-coated zinc would have practically no action on any gold in solution. Whereas in cyanide solutions copper is only slightly negative to zinc, and exceedingly positive to gold, which would therefore be precipitated practically copper-free unless the latter were present in considerable excess; and even copper-coated zinc would still be effective.

The simplicity of the reactions necessary for the whole process is now apparent:—

$$2Au + 4KCy + H_2O + O = 2KAuCy_2 + 2KHO$$

$$\text{and } 2KAuCy_2 + 2KHO + Zn = Au_2 + 2KCy + 2HCy + ZnK_2O_2$$

It does not, however, follow from this equation that 6 oz. of gold ought to be extracted at a cost of 1 oz. of zinc, as the ZnK_2O_2 combines to a certain extent with the HCy and KCy to form zinco-potassic cyanide—

$$ZnK_2O_2 + 2KCy + 2HCy = ZnK_2Cy_4 + 2KHO,$$

so that an ultimate product of the reaction is zinco-potassic cyanide and caustic potash. (*See* Chap. IX.)

This zinco-potassic cyanide has itself a solvent action on gold, and extractions have been made from ores by employing it as the dissolving agent. It has also been pointed out by Feldtmann that it may be converted into cyanide of potassium and sulphide of zinc, by reaction with potassium bi-sulphide produced during the leaching of the ore—

$$K_2ZnCy_4 + K_2S = ZnS + 4KCy.$$

On iron sulphide the action would probably be—

$$K_2ZnCy_4 + FeS + 2KCy = ZnS + K_4FeCy_6$$

any zincate of potash would also be similarly converted into sulphide of zinc and potassium hydroxide—

$$K_2ZnO_2 + K_2S + 2H_2O = ZnS + 4KHO.$$

The zinc is thus prevented from accumulating in solution.

The zinc used in the extractor box is turned down from discs or sheets on a lathe into the form of fine zinc shavings. A test of its condition is to apply a light to a mass of the shavings: if these ignite freely, they have been turned sufficiently fine. The zinc should not contain more than 1½ °/₀ of lead, which contaminates the gold slimes; "V.M." brand, Nos. 10 to 14, is generally used for this purpose. A brand of lead-free zinc is now on the market which gives very good results in the extractors, and produces bullion of a higher grade, not liable to the fine imposed by refiners on that carrying lead. The consumption of zinc is about ¼ lb. per ton of ore treated; in some cases it is claimed to be only 2 oz. It is usual to add fresh zinc to the extractors daily, to replace that consumed during the previous 24 hours.

Except in special cases, the solutions flowing from the extractors in good working order should not assay more than ½ dwt. gold per ton. Under favourable conditions, with machine-prepared zinc, all solutions from strong to extremely dilute (under ·05 °/₀) have been continuously reduced to under 2 gr. per ton in ordinary regular monthly runs, solutions being tested daily.

It was formerly suggested that zinc precipitation was not effective with extremely dilute solutions. The figures given above show that this supposition is not necessarily confirmed in practice; but it is not unusual at Johannesburg to immerse the zinc in a 10 °/₀ solution of acetate of lead, to allow the precipitated lead to form a galvanic couple with the zinc, and to thus act with more energy on the gold in solution than the zinc alone would do. In Mexico, also, the lead-zinc couple is used to precipitate the copper in solution after the gold has been removed in a previous set of boxes containing zinc only. (*See* also Chap. IX.)

Various precipitants other than zinc shavings have been from time to time suggested, such as aluminium, zinc dust or fume, sodium amalgam, and the application of electricity; but, with the possible exception of the latter, none of these suggestions appear to have made any headway in practice.

Cleaning-up.—Before commencing to collect the slimes, fresh water is allowed to flow through the extractors until the effluent is no longer decidedly alkaline. The zinc shavings in the upper compartment are now rubbed between the fingers to facilitate the disentanglement of the gold slimes, which settle in the bottom of the compartment. The tray is withdrawn and the supernatant liquor is pumped out with the gold slimes by a special filter-press, which greatly facilitates cleaning-up (*see* page 34). If no filter-press is available the plug of the compartment is withdrawn and the gold slimes flow into the launder, and down to a 40-mesh sieve about 2-ft. square.

This sieve is suspended over the settler, and has attached to it a twill bag like a pillow-case, with a 3-in. slit just under the sieve. The fine gold slimes are carried through the 40-mesh sieve, whilst the disintegrated zinc remains behind. This is rubbed with a piece of flat rubber to still further disengage the slimes, and the retained zinc is put back on to the tray, which has meanwhile been cleaned and replaced in the compartment, which has itself been washed out thoroughly with fresh water. The second compartment is now treated similarly, and the zinc, after being freed from gold slimes, is moved up into the first compartment until this contains its normal amount of shavings. This operation is continued similarly with the other compartments. Throughout the process the cleaned zinc is moved up, so that fresh shavings are put into the lower compartment only. The filter bag, under the sieve, becomes full of slimes, and the liquid overflows through the slit into the settler, carrying a portion of the slimes with it. As soon as the extractors have been cleaned up, the filter bag is hauled up ; it is next squeezed, to press out any liquid, and the contents dumped out on to a heated iron plate, with turned-up edges, on which the slimes are dried and mixed with fluxes in readiness for melting down.

The liquid in the settler, when clear, is syphoned off, and tested for gold ; the slimes are collected from the pyramidical-shaped bottom, filtered and dried as above. In other works a false bottom is fitted into the settlers, and the liquid drawn off by vacuum apparatus, which leaves the slimes in a cake convenient for handling.

The above method has the great advantage that all the cleaning-up operations are performed in the extractors, and the possible loss of slimes consequent on the removal of the zinc for treatment elsewhere, is avoided, but the use of a filter-press will obviate the necessity for the use of settlers, twill bags, &c.

The fineness of the sieve through which the slimes pass makes a great difference in the purity of the bullion obtained. When 30-mesh only is used a considerable amount of zinc passes through, and the resulting bullion is very base. With a 40-mesh sieve and well-turned zinc, bullion has been obtained above 960 fine—gold and silver—without any acid treatment, roasting, or special fluxes. It will be obvious that this mechanical method of eliminating base metal has great advantages over the expensive acid treatment, or the loss attendant on fusion with nitre. In many works the slimes are roasted before fusion, but where they have been treated as above there may be as little as 1 °/₀ of zinc present, and roasting affords no advantage, and does not increase the yield.

If acid treatment is used, the filter, with its contained cake of slimes, is taken to the acid vat and emptied into it. The acid vat may be of wood, lead lined, or of steel, enamelled or coated with special protective varnish, or of aluminium, which answers perfectly, and is clean, and is not attacked by sulphuric or even nitric acid. The dissolved slimes are washed with hot water and filter-pressed, or run into a detachable filter vat with a vacuum connection, washed, vacuum dried, mixed with fluxes, and melted. A vacuum vat is preferred by some in place of a filter-press.

Fusion of Slimes.—This operation is conducted in plumbago crucibles Nos. 35 to 70. The furnace employed resembles an assay furnace with a flat top, but is of larger dimensions, and is built specially for this purpose. A reverberatory furnace taking twelve or more No. 60 crucibles is now frequently employed in large works, and is to be recommended.

The fluxes used vary at each works in accordance with the condition of the slimes. When these are fairly pure, as above, they are melted with one-third to one-half their weight of fused borax, and about $\frac{1}{10}$th sodium carbonate; to this a little sand is added if necessary. If much sand is present the proportion of soda is increased, until it is about 2½ times as much as the sand. Zinc, copper, or other metallic impurities require the addition of more borax; old slags from previous fusions are added with advantage. Fluor spar is sometimes used. The borax used should be ground borax glass: this permits of much larger charges in the crucible, and does not swell or spirt.

The slimes and fluxes must be quite dry before being charged into the crucible, and as they are very bulky, unless acid treatment is employed, it is possible to melt only about 250 oz. of bullion in each fusion. In large works two or more crucibles are kept going continuously at a good heat, more flux and slimes being added from time to time as the fusion proceeds. Should the contents at any time threaten to boil over, the addition of a little salt causes immediate subsidence. When the crucible is nearly full it is hoisted up by means of a block and tackle running over the furnace, and emptied into a mould. The life of one of these crucibles is usually about eight fusions.

The moulds are of iron, conical, and preferably of sufficient size to hold the entire contents of the crucible. The lumps of metal collecting in the bottom of each mould are subsequently run into bars, and the accumulated slags are crushed and panned off, or run over blankets, the concentrates are smelted, and the residues sold to the smelters.

Base bullion may be improved in appearance and quality without risk of loss of gold, by granulating the metal and giving it a sweating roasting in a muffle. When re-melted it will be bright and ductile.

Staff.—A plant to treat 2,000 to 3,000 tons a month, where the power for pumps, &c., is supplied by water or from the main battery, would require, in addition to the men employed in emptying and filling tanks, which is usually done by contract, one chemist and assayer, one works foreman, a handy man for putting on cyanide, turning zinc shavings, attending to samples and odd jobs, and a boy in the laboratory.

A good man can shovel into trucks about 20 tons of tailings per day of 8 hours, though 15 tons is nearer the usual tailings pit practice; he can discharge from 40 to 60 tons per day through bottom discharge doors. One man with 60-ft. head of water and 1-in. nozzle will discharge from a vat 20 tons or more dry weight of tailings per hour.

There is usually an abnormal consumption of cyanide at the start of operations with a new plant. This is due to the action of the cyanide solutions on the fresh surfaces of the timber and metal; and also, and perhaps mainly, to the comparatively simple nature of the solutions at the outset.

POINTERS.

1. Have the material to be cyanided thoroughly tested first by a reliable man, and be guided by his results.
2. Have your plant designed on most recent lines—don't be content to merely copy your neighbours, and repeat their mistakes. Well designed plants are treating 1-dwt. sands at a profit.
3. Have a good man to run the plant, who will see that the amount of *gold sent to the Bank* is close on to the theoretical extraction. A loss of 10 °/₀ of the gold by imperfect washing and precipitation, or careless clean up, may mean the difference between running the works at a decent profit or at a serious loss.
4. Test your sump solutions from time to time against solutions made up of fresh water. If they do not give as good extractions, find the reason for this, or get rid of the solution. (*See* Chapter IX.) A little lead acetate or chloride added to the sump solutions

will often improve their work, by taking up the soluble sulphides sometimes formed by the pyrites, and lime cures many troubles caused by soluble sulphates, carbonates, and metallic salts. See that solutions are well aërated.

5. If your ores or tailings are free milling—contain no sulphides—see that your solutions are slightly alkaline, as slight alkalinity tends to improve the extraction. Add soda or lime, a little at a time, till the solution flowing out from the extractor boxes does not rise in cyanide test.
6. If your charges contain sulphides, avoid excess of alkali, which forms soluble sulphides and lessens the extraction. A little soda is necessary, and helps the zinc box reaction, but use lime generally for neutralising purposes, and keep your solutions so that they rise apparently in cyanide test if soda is added.
7. Keep an eye on the assay of the bottom layer (12-in.) of tailings in the vat.
8. To test acidity :—

   Add 44·6 grains of commercial caustic soda to a litre of water. Take 100 grains of ore and stir up with an equal amount of water, run in the soda solution until blue litmus is no longer turned red. Every c.c. of soda solution taken means $\frac{1}{10}$ lb. of soda for each ton of ore.

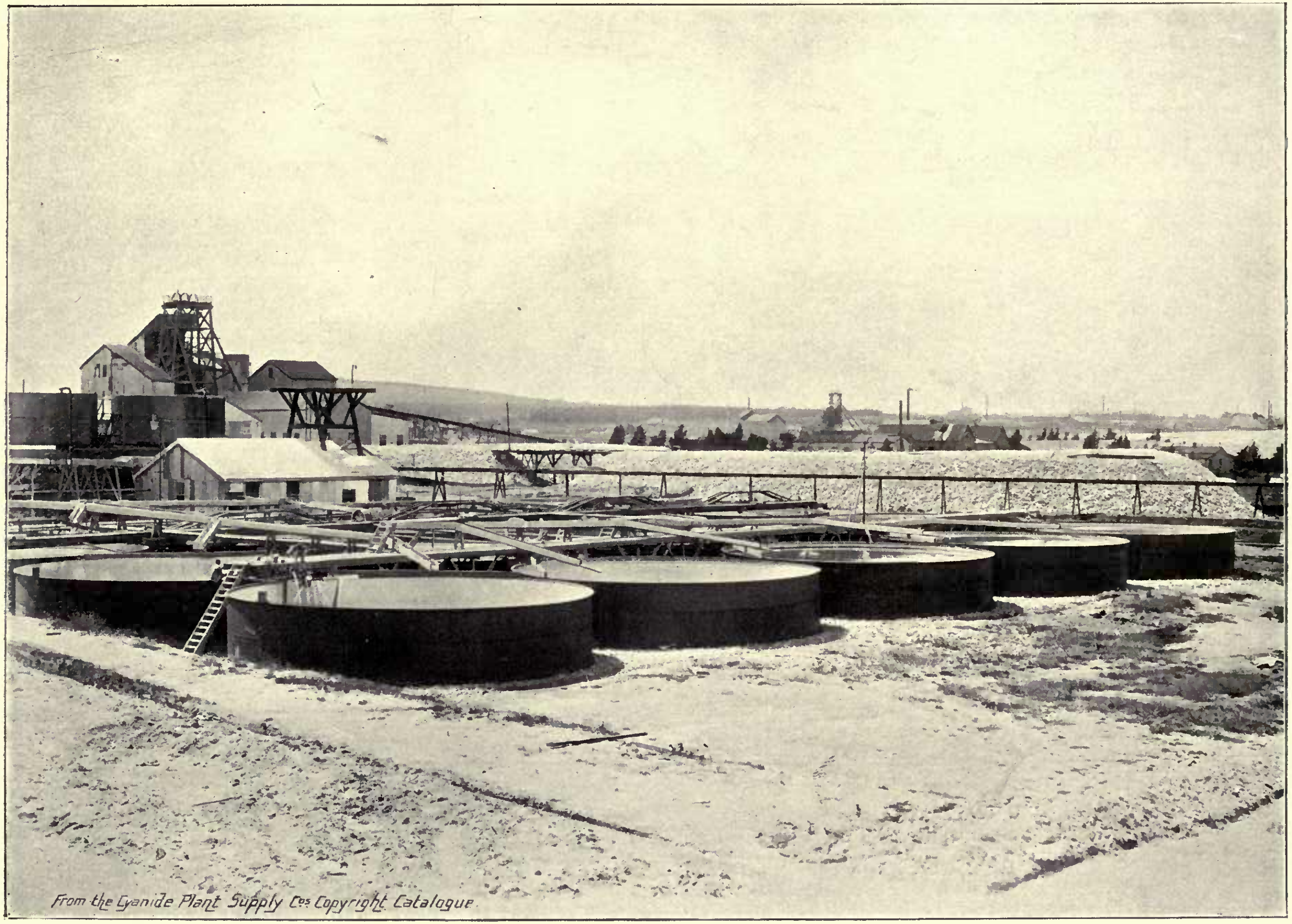
   If the ore contains soluble salts of iron or copper, give water washes before adding the lime or soda.
9. To test strength of cyanide solutions :—

   Dissolve 130·8 grains of pure triple crystal nitrate of silver in a litre of distilled water. Each c.c. taken by 100 grains of cyanide solution—0·1 $^{c}/_{o}$ KCy.
10. In cleaning up, handle the gold slimes as little as possible. *Avoid the use of nitre.* If you wish pure bullion, use lead-free zinc,

and treat slimes with sulphuric acid and hot water, or roast as shown on page 61.

11. When adding fresh lots of mixed slimes and fluxes to the melting pot, do this before the upper portion of the last charge is fused; this prevents violent ebullition and loss of gold slimes. It is better to add a little borax first, and then the slimes and fluxes may be carefully added by a special scoop or wrapped into a parcel in a piece of filter paper. Always add a cover of borax to the charge in the crucible.

12. To lessen cyanide consumption keep comparatively small quantities of stock solution on hand, and constantly circulate by pumping small quantities on to the charge every three or four hours. Thus, a 60-ton charge may be worked by pumping up 600 gallons every three or four hours.



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A SLIMES PLANT AT JOHANNESBURG (DECANTATION METHOD).

CHAPTER IV.

Treatment of Slimes.

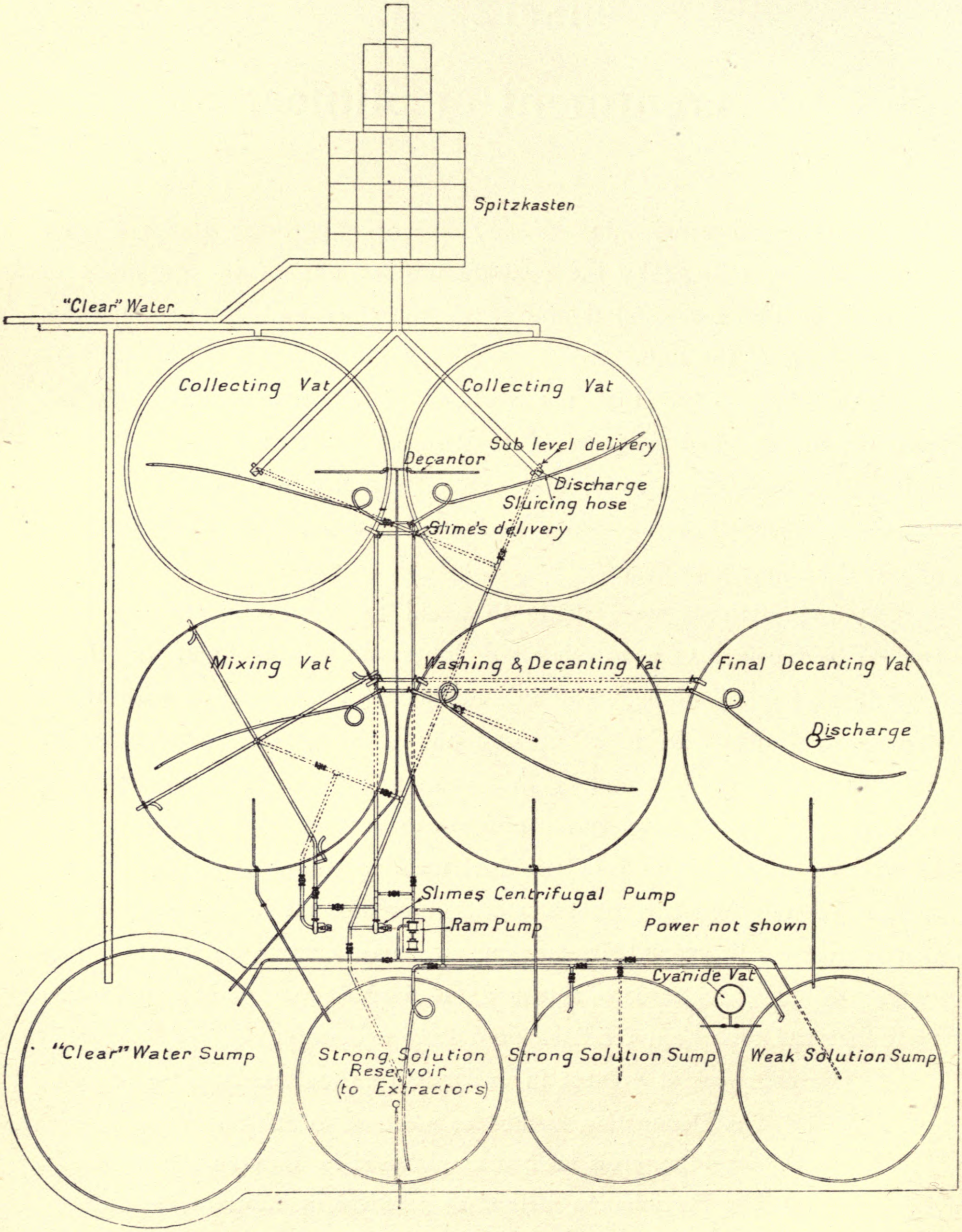
NOW that almost every mine of any size or importance makes a point of treating its tailings by the cyanide process, one of the questions of the hour is the successful treatment of the finer and almost impermeable products of the mill.

It has been found that the portions of the tailings dump farthest from the intake yield a mass of matter practically untreatable by the ordinary percolation process, and much ingenuity and skill has been exercised, at very heavy expense, in finding a practicable solution of an admittedly difficult problem.

At first attempts were made to avoid the necessity of a separate process by recourse to sun-drying the slimes and mixing them with large quantities of sandy tailings, and this method is still employed with much success in India. In New Zealand the difficulty was overcome by recourse to dry-crushing, and by this process ores containing over 25 °/₀ of clay are reported to have proved themselves perfectly amenable to ordinary lixiviation in vats, the final washing being aided by vacuum suction. But in Western Australia and other districts it has been found that even coarsely ground dry-crushed ores may contain so great a percentage of clayey calcareous slimes as to impede percolation, and other treatment has thus been rendered absolutely necessary.

The processes at present in use may be thus tabulated :—

(1) **The Decantation method as practised in Africa.**
(2) **The Filter-press method as practised in Western Australia.**
(3) **The Precipitation method as practised in America.**
(4) **The Sun-drying method as practised in India.**



Outline of a "Decantation" Slimes Plant.

DECANTATION.

In Africa the first large scale plant for the systematic treatment of the accumulated slimes resulting from the wet-crushing mills was erected by the Rand Central Company near the Robinson Mine. The plant was of elaborate design, and comprised numerous spitzlutten for separating sands of the very finest grade, and vats provided with agitators working over filter bottoms under which suction could be applied. This plant, as designed, was not altogether a success, as the slimes tended to form thin tough cakes on the filter bottoms, and to thus stop the whole operation.

The next attempt was made by Mr. J. R. Williams at the Crown Reef Works, and depended on the agitation of the slimes with extremely dilute cyanide solution and the decantation of the clear solution as the slimes settled. It was found that the addition of a definite proportion of lime made this process quite feasible, and the plant was so successful that almost every large plant subsequently erected was designed on these lines. At first there was a good deal of difficulty caused by choking and clogging, but subsequent modifications enabled the method of agitation by pumps to be adhered to, although in some plants mechanical stirrers were introduced for this purpose. At the outset the solutions were run off from holes in the sides of the vat, but the introduction of decanting pipes with swivel or flexible joints, permitting motion in a vertical plane, speedily superseded the original contrivance, and enabled the solutions to be run off with greater regularity and regard for the settling rate of the slimes. The following is a brief description of the routine of the decantation process, and is based largely on African practice.

The pulp flowing from the spitzlutten and the overflow of the sands vats receives automatically by an ingenious contrivance a regulated amount of lime, varying from $5\frac{1}{2}$ to 20 lbs. of lime to 1 ton of dry slimes, or from $\frac{1}{2}$ to 1 lb. per ton of pulp. The amount of dry slimes present in the pulp is found by filling a bottle with the mixture, weighing the bottle, and a simple equation gives the amount of dry slimes. Thus, if we assume that—

a = the weight of a bottle full of water.
b = the weight of the bottle full of slimes pulp.
c = the dry weight of slimes in the bottle.
d = the specific gravity of the dry slimes.

Then if we place a known weight of dry slimes into a bottle previously filled with water, and weigh, we can obtain d thus:—

$$\frac{c}{a + c - b} = d \text{ and } c = (a + c - b)\, d.$$

Thus, if the bottle full of water weighs 15,000 grains (a), and the same bottle filled with slimes pulp weighs 16,500 grains (b), and we have added 2,500 grains of dry slimes (c), the specific gravity

$$d = \frac{2{,}500}{15{,}000 + 2{,}500 - 16{,}500} \text{ or } \frac{2{,}500}{1{,}000} \text{ or } 2{\cdot}5\,;$$

and conversely

$$c = (15{,}000 + 2{,}500 - 16{,}500)\, 2{\cdot}5 = 2{,}500 \text{ grains of dry slimes}$$

in the bottle.

If we know the weight of water in the bottle, and divide 100 c by this, we obtain the percentage weight of dry slimes present in the pulp. This calculation is simplified by using a bottle holding a known weight of water—say 10,000 grains—when c would only require to be divided by 100 to obtain the percentage required. Of course, when the specific gravity of the slimes is 2·5, it is merely necessary to add 2/3 to the increase in weight noted to obtain the weight of dry slimes present.

Thus, if the water and bottle weigh 15,000 grains, and the bottle and pulp 16,500, the addition to the difference (1,500) of 2/3 of this, viz., 1,000, gives 2,500 as the correct amount of slimes present; and similarly if the bottle weighs 21,000 grains, 2/3 of the 6,000 grains of difference, viz., 4,000 added to the 6,000 gives 10,000 as the correct amount of slimes present. This is simply allowing for the displaced water, which is 2/3 of the weight of the slimes of a specific gravity of 2·5 or $\frac{5}{2}$.

Similarly a table can be easily arranged to show at a glance the percentage contents, as readily as a bullion table shows the value of an ore from the weight of the assay bead.

It is advisable to boil the water in the bottle after the addition of the dry slimes for the first experiment given above, and, of course, allow it to cool before re-weighing, as otherwise the air entangled in the dry slimes is apt to give an erroneous result.

Prior to its arrival at the slimes plant the pulp has passed through a small spitzlutten, which eliminates any coarse particles of sand which may have escaped from the vats. A pipe or launder returns these sands to the tailings wheel or pump.

The pulp as received at the slimes plant contains from 2½ to 5 °/₀ of dry slimes, and this is passed through large spitzkasten, which deliver a product containing from 7½ to 12 °/₀ of slimes, the balance of the water flowing off practically clear to a "muddy water" dam. The slimes pass into the first vat, which has a sub-level delivery and overflow rim launder. By this ingenious contrivance the almost clear water runs to the surface and overflows into the rim launders, while the slimes subside to the bottom of the conical-shaped vat until this has received its calculated charge of, say, six or twelve hours, or one day's output —this of course depends on the size of the mill. In large plants more than one receiver vat per diem must be allowed for.

The pulp is now turned on to No. 2 receiving vat, and the contents of No. 1 allowed to subside. This takes from 2½ to 12 hours, and even longer, depending on the size of the vat and the nature of the material to be treated. The vat thus holds about 1 ton of slimes to 4 or 5 of water, and as the level of the settled slimes subsides the supernatant liquor passes through the swivel pipe, which is adjusted (usually by a boy) to continuously decant the liquor until there remains only a slimey mass containing about 50 % of moisture.

It is scarcely necessary to remark that each plant requires to be designed specially for the quality of material to be treated. The size of some of the most recent vats is 40 feet diameter by 14 feet deep at the sides and 17 feet deep at the centre. They are, of course, made of steel, and are usually of 5/16-inch steel plate, with the exception of the conical plate at the bottom, which is ½-inch thick.

Cyanide solution containing from 0·01 % to 0·02 % is now pumped at great pressure through a hose with a nozzle to the amount of about 4 tons of solution for every ton of slimes, and this washes the viscous mass through the opening at the conical bottom of the vat into the intake of a special slimes centrifugal pump. This method of dealing with the slimes is preferred to the agitation of mechanical stirrers, as it ensures every particle of slimes being broken up and thoroughly mixed with the cyanide solution, and the whole of the contents of the vat are sluiced out into the pump with its swiftly revolving vanes. It is found that most effective aëration can also be obtained in this manner, particularly if a valve for this purpose is introduced into the suction of the pump. The agitation vat, into which the slimes are pumped, may be fitted, if necessary, with a circulating arrangement, by which the slimes may be kept by pumping in constant agitation for as many hours as may be found necessary for the effective solution of the gold.

After agitation the slimes are settled, and the gold-containing solution decanted off for precipitation, after subsidence or clarification, either by zinc or by electrical deposition. The settled mass is now sluiced into another vat with weaker cyanide solution, to go through a further decantation process; and if the slimes are rich, the washing may be repeated for a further decantation of the auriferous solution—with a proportionately higher extraction.

The slimes may be settled in deep tanks until they contain less than 40 % of water, and are finally disposed of by hydraulicing, or by washing into the intake of a pump, or by agitating with sufficient water to make a pulp sufficiently liquid to flow to the residues dam, from which any overflow of water is returned to the works. It is this final disposing of the slimes which is the chief cause of the loss of water in this process.

As far as practicable, the solutions are used in advancing series. Thus, the final washing water becomes the weak solution of the next charge before it passes through the precipitation boxes; and the weak solution of a previous charge, in like manner, is made up in strength, so as to become the strong solution of the next charge. By this means the necessity of precipitating the gold from a huge bulk of extremely poor solutions is obviated, and the amount to be dealt with kept within practicable bounds.

If the solutions are to be precipitated by zinc, it will be advisable to allow a space in the extractor box of about 2 cubic feet of zinc shavings for each ton of solution to be treated per diem. If electrical precipitation is to be employed, the boxes must have at least three times this capacity.

The following are, perhaps, the best results which have yet been obtained by the use of this process. (*See Journal of the Chemical and Metallurgical Society of Johannesburg.*)

Crown Deep Mine.

Tonnage of slimes treated (in 3 weeks)	11,280.
Assay value—dwts.	3·157.
Actual extraction	87·49 %.
Tonnage solution, measured by meter	21,842 tons.
Assay value	36 grains.
Gold in total solution	1,645·679 oz. fine gold.
Gold recovered	1,557·847 „ „
	or 94·6 % of solution treated.

Working costs, 2*s*. 9*d*. per ton. Profit, 9*s*. per ton.

It is anticipated that these working costs will be speedily reduced to 1/9 per ton.

At another plant an actual extraction was shown of 89 °/o on a six months run, but it may be taken that the average extraction is considerably less than this. Such low extraction is, however, due to the low value of the slimes treated, as a 2 to 3 dwt. proposition is naturally not worth the increase of plant and slight expense necessary for a further transfer and washing to recover an additional 3 to 5 °/o.

The water consumed by the method is stated to amount to about 50 °/o of the slimes treated; but it is probably safer to estimate that 1 ton of water is thrown away for every ton of dry slime treated.

A remarkable result of the adoption of this process in Africa, and the successful working of the large and expensive plants installed for this purpose, has been the discovery that the methods adopted for facilitating the treatment of the slimes give greater profits than the actual treatment of the slimes themselves. Thus, the addition of lime to the mortar boxes causes coagulation of the finer gold-bearing particles of the pulp, and improves the amalgamation results by 5 or 6 °/o, with a like impoverishment in the value of the slimes, to treat which the plants were erected. The settling of the slimes and decantation of the water, with direct return of the latter to the mill, has reduced the consumption of water by as much as 50 °/o, and caused a heavy saving in pumping charges. The absence of slimes from the tailings permits better percolation and draining, and consequently of a higher extraction by about 5 °/o, and there is also an increased quantity of tailings caught in the vats and return spitz.

The unexpected result, therefore, of the erection of a plant has been to diminish the value of the material on which it has to work, and the mines in Africa are consequently called to seriously consider the proposition whether it would be better to add lime to their boxes, erect return spitz for separating the sand carried over with the slimes, and separate the water in large spitzkasten, all as is done at present for the slimes plants, *but to omit* *the installation of the actual slimes plant*, as the foregoing accessories are reducing the value of the slimes to under 2 dwt., and this scarcely pays for separate treatment.

To discover if this process was suitable for the type of slimes met with in Western Australia, experiments were undertaken with the slimes obtained from, amongst others, the following Companies:—Hannan's Brownhill, Sons of Gwalia, Bayley's United, and North Boulder. It was found that these slimes subsided quite practicably without the addition of lime, owing, probably, to the calcareous nature of the gangue, and that the addition of lime tended rather to make the subsidence less rapid and less complete.

There was no difficulty in obtaining a settlement of the slime which would permit the solution to be drawn off until the sediment contained 60 to 50 % of dry slimes to 40 to 50 % of solution; and the gold was, as a rule, so readily soluble that 96½ % was found to be in solution at the close of one of the trials. The amount of gold actually recovered depended on the number of decantation washings, but from the slimes of one of the above-mentioned Kalgurli Companies 91 % was recovered as a result of three cycles of agitation and settling. This repetition of agitations and settlings may appear expensive, but in Africa the cost of this dealing with the slimes, which is effected by pumping, amounts to less than 4*d.* per ton, and the total labour cost is under 1*s.*

The following figures show the capacity to be allowed for per long ton of slimes treated:—

20 of water to 1 of slimes takes 732½ cubic feet of space per ton.
4 „ „ 1 „ „ 157 „ „ „
1 „ „ 1 „ „ 52 „ „ „

Experiments also showed that the slimes settled quite well in saline water, similar to that obtained at the Great Boulder, and containing 6·2 % of salt, 0·45 % of magnesium chloride, and 0·73 % of sulphate of lime. This should not, therefore, be in the way of the adoption of the decantation process at Kalgoorlie; and though at first sight the decantation process appears to require much more water than the filter-press process, yet in the former process the sands might be separated from the slimes to a much greater extent than at present, and thus there would be a smaller quantity of the latter to be finally thrown away containing an equal weight of water. It may be assumed that the actual consumption of water by the decantation process would be from two to three times the amount required by the filter-press method, but whether the cost of the number of decantation washings necessary for the treatment of high-grade slimes, and the value of the gold lost by the poorer extractions of the decantation method would not considerably exceed the expense of filter-press treatment—now universally adopted for material of this nature—is a matter for investigation and determination.

FILTER-PRESSING.

(*See* also Chapter V.)

A consideration of West Australian practice shows that a higher average extraction is being obtained on higher grade slimes, at a higher working cost, but with a lower consumption of water. The assay values of the residues discharged appears, however, to be higher than in Africa, where this has been reduced to as low as 9½ grains of gold per ton ; and the heavy percentage extractions obtained at Kalgoorlie are thus due to the grade of the slimes treated.

A considerable amount of experiment has been necessary to determine the best method of press treatment for this material, but the following is that most generally adopted at present :—

The slimes are agitated in mixing vats for a sufficient time to dissolve the gold, and aërations (and even dilute chlorine solutions) are employed to accelerate the rate of solution. The slimes are then run into a monteju, and forced by compressed air through the presses, and 70 °/₀ of water is immediately obtained quite clear and ready for return to the battery dam or reservoir. Water washing is carried out *in situ* with the cakes in the presses; it has been found to increase the extraction if air is blown through the cakes in the press after the first cyanide solution has been expressed and before washing. The presses have capacities varying from 1½ tons to close on 6 tons per charge, the latter being the estimated capacity of the new presses for the sulphide plants. The tendency of the experience of the last two years has been towards increasing the size of the filter-presses; this has necessitated attention to the difficulties arising from the handling of such heavy masses of metal—as, for example, the head pieces, which have required to be specially reinforced to stand the strain. An arrangement is now included in some of the best presses whereby the screw is eased of the heavy strain due to the weight of the head piece, and a system of rollers or wheel bearings has been introduced to facilitate the moving of the plates and distance frames.

It has been found essential to have apparatus which shall fill the presses rapidly without intermission or shock, and pumps have therefore been superseded by montejus, to which an air jet has been added to agitate the contents and prevent the settling of the coarser particles.

At present the cycle of charging, pressing, washing, and discharging the contents of a press may be completed in from 1 hour to 1¼ hours, and the discharging itself takes only about 20 minutes, but the average number of charges put through a press does not exceed ten per day of 24 hours. The pressed cakes are sufficiently dry to be handled and trucked to the dump——this is a great advantage over the slimey African pulp, which can only be handled by diluting and pumping, sluicing or flowing.

At the Lake View plant, which was one of the pioneers of this system of treatment, solution took place *in the presses themselves*, but in the most recent plants—the Hannan's Brownhill and the Ivanhoe—provision is made for the solution of the gold before it enters the presses, and this method is undoubtedly the correct course to follow.

The extraction obtained by the press method varies from about 82 °/₀ to 92 °/₀, and published costs (Lake View) are as follows :—

	Average per ton.	
Superintendance and settling solutions - -		6·832
Discharging and filling presses - - -	2·	10·353
Compressed air - - - - - -		9·663
Turning zinc - - - - - -		·196
General repairs - - - - - -		3·037
Assaying, retorting, and melting - - -		2·956
Zinc - - - - - - - -		·528
Filter cloth - - - - - - -		3·121
Cyanide of potassium - - - - -		10·748
General stores and charges - - - -		4·403
Electric light maintenance - - - -		3·488
	6*s.*	7·325*d.*

But the actual cost of filter-pressing, excluding cyanide and zinc, &c., has now been reduced to 4*s.* per ton.

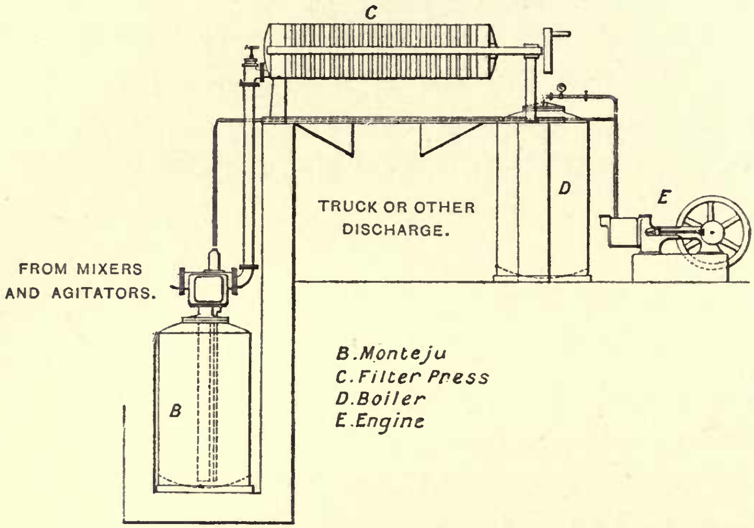
From the above it will be seen that the one item of "Discharging and filling presses" costs more than the whole of the African method, including cyanide. An installation of twelve 1½-ton filter-presses has a capacity of about 120 tons of slimes per diem. In practice one press is found to treat from 10 to 12 charges per day of 24 hours.

It is unlikely that filter-pressing at Kalgoorlie will ever be replaced by decantation, at any rate, so long as the ores treated are of such high grade as at present. This is a question which can only be decided by the metallurgists on the spot, who have by their enterprise and ability already converted one of the most dreaded contingencies—the slimes question—into such a success that the increase of the percentage of fines in the total yield from the mill is now a desideratum merely as a sure prelude of resultant high extraction. This matter will doubtless receive attention from those most interested on the spot, but it cannot be doubted that the high results obtained by filter-pressing must be regarded as exceedingly satisfactory, whereas by decantation serious discrepancies not infrequently occur between the theoretical results and those actually obtained, thus making the actual recovery considerably lower than the calculated results. The high first cost of a decantation installation is also a very serious matter, though in mines with large working capitals the low working cost is undoubtedly a great attraction, and has resulted in a very general adoption of the decantation process when once results have been proved to be remunerative.

It is suggested that in place of agitating the slimes in the manner now proposed in West Australia, these should be circulated by centrifugal slimes pumps in and from conical tanks, as is done in decantation practice. By this method much more thorough agitation and aëration is possible, at less expense than with an arm stirrer, and the liability of the latter to breakage and to jamming in the settled slime is avoided. The agitated slimes, in place of being decanted (as in Africa) with subsequent washings by the same process, would be passed direct to a filter-press and washed therein, and loss of water thus avoided. Two conical collecting vats and a mixing vat would consequently suffice for the whole of the agitation operation.

It is also suggested that in place of the double pressing described by Mr. Wilson in Chapter V., the surplus water in the pump would be more cheaply eliminated by the use of a collecting vat or spitzkasten, delivering clear water at the top, and at the bottom a pulp concentrated to about 50 °/₀ of moisture. One set of filter-presses and single filter-pressing would thus suffice for all cases.

PLANT FOR FILTER-PRESS SLIMES TREATMENT.



This usually consists of leaching filter-presses, dissolving vats with agitators, montejus, with special automatic head pieces arranged for rapid working; air compresser, air receiver and fittings, wash-water pump with engine, together with pipe work and fittings for connecting the different pieces of apparatus together between themselves. The capacities of the mixing vats are such that they will each hold from six to twelve hours' working of the leaching presses, so that the quantity of slimes will be under the action of the cyanide for this period.

It should be distinctly understood that the quantity which can be turned out from a given filter-press plant will depend on the character of the slimes, and in some cases the chambers are required to be 1½ inches deep, in others they may be 3 inches, or more, according to the special character of the slimes in question; the object being to fill the chambers as quickly as possible so as to avoid the deposition of fine sand in the chambers during the formation of the cakes, and so produce uniformly permeable charges. It should be distinctly understood that the quantities specified by makers are not always to be relied on, as may be seen from the following paragraph.

In estimating the capacity of any filter-press plant, it should be noted that the usual average output per press at the mines is 10 charges per diem of 24 hours—certain makers estimate on a basis of 18 to 24—and that these charges are of *wet pressed cake* containing about 30 °/₀ of moisture by weight. The size of the press should be correspondingly increased if *dry slimes* are taken as the basis of calculation. Thus, a 4-ton press should turn out 40 tons of slimes (press cake) per diem, but this is only 28 tons of dry slimes—it would require a 6-ton press to deal with 40 tons of dry slimes per day.

PRECIPITATION.

The precipitation process referred to as having been in use in America is based on the direct precipitation on to mercury of the gold contents of the viscous or muddy solution. There is thus no necessity for filtration at all. The pulp is charged into pans, barrels, or other apparatus with the required amount of cyanide solution. Mechanical agitation takes place until the gold has been dissolved, and means are then employed to precipitate the gold from the mass of slimes and solution on to mercury to form an amalgam, which settles in the bottom of the pan or other apparatus, and is collected in the usual manner. After the solution has been freed from its auriferous contents, the whole mass of slimes and solution is discharged and replaced by a fresh charge. The means adopted to effect precipitation are electrical or chemical.

Thus the Pelatan-Clerici process, a large installation of which (50-ton plant) was erected at Delamar, had a tank with agitators suspended over a mercury bath resting on a copper plate. The agitator was the anode, and the mercury bath the cathode. Salt was added to lessen the electrical resistance of the solution, and after the gold had been dissolved by the action of cyanide it was precipitated on to the mercury by the electric current.

This process was not sufficiently successful on the readily percolating ores at Delamar, and was superseded by the ordinary cyanide lixiviation process; but it is reported in operation in Canada. A somewhat similar process—the Riecken process—has recently been installed at Kalgoorlie (*see* chapter on the Treatment of Sulpho-Tellurides). As in precipitation processes the gold-bearing solutions are supposed to be thrown away with the residues, it will be seen that such processes depend for their efficiency on the complete precipitation of the dissolved gold. This has always proved a most difficult matter to effect electrolytically, and thus one can understand these processes not surviving in face of the filter-press or decantation methods.

Greater advantages, however, appear to be possessed by the simpler alternative process of precipitation by chemical means rather than by an electrical current. Thus the addition of sodium amalgam, or of copper or zinc amalgam, is effective for the purpose; and the latter method (the Gilmour-Young process) has been in use at Nicaragua for a considerable period.

It is claimed that the process is extracting at the Santa Francisca Mine upwards of 90 °/₀ of the bullion from the slimes, at a cost of about 3*s*. per ton for chemicals and 3*d*. per ton for labour. A considerable amount of horse-power is required, the cost of which, and of superintendence, assaying, &c., is not given.

The ore or slimes is charged into a pan in 2-ton lots, with sufficient water to form a thick pulp. From two to six bottles of mercury are added until the globules can be seen circulating in the pulp, and then the required amount of cyanide. After running this for about two hours, 10 lbs. weight of zinc or copper amalgam, or both mixed, is added, and the pan run for four hours longer. The contents of the solution are reduced to about 10 grains per ton of solution. The pulp and solution are then discharged into a settler, and the mercury recovered in the ordinary manner. The copper amalgam is made by treating copper sulphate with cast-iron turnings and mercury in a separate pan. The zinc amalgam is easily made by pouring mercury into molten zinc; but when made, it should be kept under water to prevent rapid decomposition.

As the process successfully treated the output of the mine for considerably over twelve months, it has claims to be regarded as practicable; and although much more expensive than the decantation method, it may be of considerable service to mines which have a disused pan plant available, and have not a sufficient output to warrant the expense of the erection of a special slimes plant of the more expensive type; but filter pressing is undoubtedly a superior process when circumstances permit of the use of presses.

Working Details of Process as in use at the Santa Francisca Mines, Nicaragua (as furnished by the Secretary).

Cyanide solution is made up as follows:—

KCy - - - - 0·2 °/o
NaHO - - - - 0·1 °/o

The pans (5-ft. Boss) are charged as thickly as is consistent with a good circulation of the pulp, the cyanide solution being run in at the same time as dry ore is charged into the pan, until the charge is sufficiently watered down to circulate well.

Two or three bottles of mercury are added, and the pan run without further addition for about two hours, when 30 lbs. of amalgam are added, and at the end of another four hours the precipitation is complete, and the pan is discharged into a 7-ft. 6-in. settler, in which the mercury separates out.

The amalgam is used over and over again with fresh charges of pulp until it becomes rich enough to retort, and by passing the amalgam ten times through the pans the bullion melts to bars showing 700 fine. The precipitation is quite as complete with rich amalgam as with poor, but if the amalgam becomes very rich the quantity of such rich and consequently less active amalgam must be increased.

The point in this process over which most care must be taken is that the mercury shall rise well with the pulp. In the 5-ft. Boss pan the speed must not fall below 68 revolutions per minute. The mullers must be kept well down, but need not be set to grind.

The dry charge can easily be sampled, but to take a fair average of the discharge is very difficult, and for this reason all results have been calculated on bullion recovered. For experimental purposes, and as a check on working, the easiest method is to take a sample from the pan just before the discharge and wash out the mercury and dry the clay and sand (now free from mercury) together: in this way the right proportion of sand to slime is kept.

In case of the extraction not being satisfactory, find whether the gold remains in the sandy portion of ore (A), or the clay portion (B), or in solution (C).

(*A*) If remaining in sand, the precious metal can only be extracted by re-treatment in a leaching plant.

(*B*) If in the clay portion, the want of extraction is due to any of the reasons which affect extraction in the ordinary cyanide process, and should be rectified by experiments in the laboratory.

(*C*) If the gold remains in solution, and the precipitation is not complete, the cause will be found to be faulty circulation of the mercury and precipitation due to one of the following reasons:—

(1) Consistence of the pulp is too thin.

(2) Speed of agitation is too slow.

(3) The mullers of the pan are not kept nearly touching the bottom of the pan.

The copper amalgam can be made in one of the Boss amalgamating pans, but, owing to the action of the copper sulphate on the iron portions it is better to make it in a special pan.

A convenient charge is:—

100 lbs. copper sulphate
30 lbs. iron filings.
6 bottles of mercury.

The reaction is complete in from two to four hours, depending on the condition of the filings.

The above charge produces about 150 lbs. of squeezed amalgam containing 14 °/₀ to 15 °/₀ of copper.

It may be noted as a point of interest that as the copper amalgam gets richer, so does the percentage of metallic residue left by retorting rise. To instance this—a bulk of amalgam well squeezed was found to yield on retorting metallic residue equivalent to 25 °/₀ of its original weight. This metal went 750 fine in gold and silver.

Extraction from Air-separated Ore Slimes.

These slimes are separated out from dry ground ore by means of a small fan. They consist of pure clay, and contain hardly any sand at all. The time and detail of working are the same as for the Santa Francisca ore treated by this process.

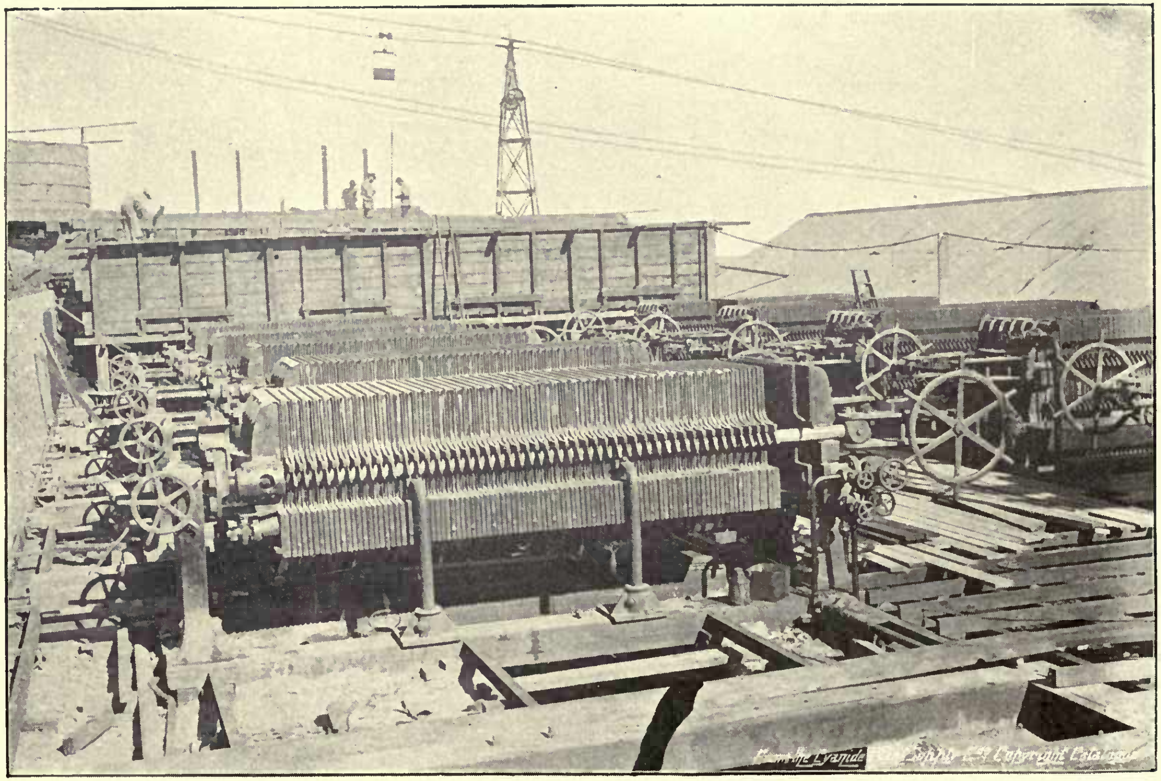
The following are the results of treating this material, and are calculated from assays of the slimes before and after treatment:—

DATE.	TONS.	CHARGING ASSAY.		TAILINGS ASSAY.		EXTRACTION.	
		Gold.	Silver.	Gold.	Silver.	Gold.	Silver.
		Oz. Dwt. Gr.	Oz. Dwt. Gr.	Oz. Dwt. Gr.	Oz. Dwt. Gr.	Per cent.	Per cent.
August, 98,	8	1 10 0	4 10 0	0 3 0	1 6 16	90·0	70·3
October, 98,	18	1 12 0	4 8 12	0 1 22	0 15 18	94·0	82·2
November, 98,	13	1 0 0	2 11 0	0 1 20	0 14 0	90·8	72·5
December, 98,	20	1 4 20	3 4 12	0 2 18	0 13 20	88·9	78·5
January, 99,	8	1 8 0	3 13 8	0 6 0	1 15 8	78·5	51·8
February, 99,	21	1 9 2	3 19 2	0 5 21	1 3 14	79·8	70·2
March, 99,	35	1 3 18	3 7 18	0 2 20	0 16 8	88·0	75·9
Average ...	123	1 6 8	3 12 5	0 2 19	0 17 12	89·3	75·8

It is not necessary to use much zinc amalgam with the copper amalgam. With some ore none at all is used. The best way to use the zinc amalgam (made by pouring ten parts of mercury into one part of molten zinc) is to add about 50 lbs. to 300 lbs. of copper amalgam and mix the two well together. The chief advantage of the small proportion of zinc is that it preserves the amalgam from being reduced in weight to the same extent as pure copper amalgam would be.

SUN-DRYING.

The treatment of slimes by sun-drying is of course only applicable to highly silicious ores. In India it has been found quite practicable to treat in this manner accumulated slimes resulting from the pan treatment of tailings; in other places the dried slimes are mixed with a percentage of coarse clean sands, and the whole operation is then carried on in the ordinary manner, with the addition of a vacuum suction arrangement for accelerating and improving the final washings of the charges treated. But this method is usually only a temporary expedient pending erection of a suitable treatment plant on the lines of one of the methods previously described.



FILTER PRESS SLIMES PLANT AT LAKE VIEW CONSOLS (W. AUSTRALIA).

CHAPTER V.

The Filter-Press Treatment of Slime in Western Australia.

(By J. K. WILSON.)

FILTER-PRESSING was first introduced by the Lake View Company in the early part of 1898, and although subsequent decantation experiments were carried out on a large scale by several of the leading companies, it was found that the time occupied in thoroughly washing the large bodies of rich slime by decantation was too great as compared with the rapid manner in which this could be effected by means of filter-presses, and although the working cost of pressing is considerably higher than decanting, it has now been universally adopted on the fields.

The design and arrangement of the various plants at Kalgoorlie naturally vary according to the nature of the ore, or the ideas of the engineer or metallurgist in charge. Broadly speaking, however, they may be divided into two classes, viz., single pressing and double pressing plants.

Single Pressing.—This method is applicable for the treatment of (*a*) slime containing gold so readily soluble that it may be dissolved in the press, and (*b*) slime from a dry-crushed ore, or ore that has been lying exposed to the sun and the weather, and contains but little moisture. At Kalgoorlie single pressing is carried out in two ways. (1) Where the gold is brought into solution in the presses themselves, as at the Lake View and Golden Horseshoe Mines. (2) Where it is dissolved in agitation vats prior to entering the presses—the press being used merely as a filtering medium—as at the Brownhill, Ivanhoe, and Great Boulder Mines.

Under the first sub-division,—where the gold is dissolved in the presses themselves,—this method was first introduced by the Lake View Consols, and has been worked by them for some two years with satisfactory results. The plant consists of 16 complete-extraction frame-presses—the frames of which range from 2·25-in. to 3-in. in width; 2 montejus pressure tanks, 2 Excelsior (gravitation) filters for clearing the liquors before entering the precipitation boxes, and a 4-drill air compressor, together with necessary pumps, pipes, &c. The plant is capable of treating about 4,000 tons of slime a month, at a cost of treatment of about 6*s.* per ton.

The *modus operandi* is as follows:—The slime on leaving the distributing vats, where the bulk of the sand has been settled, is passed through a series of pointed boxes, or spitzkasten, and any sand which may have escaped the distributors is caught. The true slime, 98 °/₀ of which will pass a 150-mesh screen, flows into settling tanks, and the excess of muddy water is drawn off and pumped through two presses set apart for this purpose. When these presses become full of slime, they are dealt with in the same manner as the others, the water being returned to the mill. As the slime settles in the tanks it is run out from the bottom into the montejus tank and forced into presses by means of compressed air—the pressure required being from 30-lb. to 60-lb. per square inch, according to the thickness of the cake. The clear water is returned to the battery, and a cyanide solution containing about 2 °/₀ KCy is pumped through the press, compressed air also being forced through the charge at intervals of five minutes or so until the gold is dissolved. The wash, either of weaker cyanide solution or water, then follows, and the charge is dried to a certain extent with compressed air, after which the press is opened and discharged. It is found that all the gold passes into solution in about twenty minutes, and the whole operation of treatment is completed in about two hours.

It is a remarkable fact that with certain of the Kalgoorlie ores no better extraction is obtained by a prior agitation for three or four hours in vats, than that effected in the press with aërations as described. It is also a noticeable feature of this method, that whenever air has been forced through the charge, the exuding liquors are of increased gold content, showing that air under pressure has a remarkably beneficial effect in aiding the solution of the gold. Although this system of treatment has the disadvantage of requiring a larger number of presses to treat a given quantity of slime than the agitation method, still the agitation vats and other appliances required for plants of the latter class counterbalance to some extent the first cost of the plants for the two systems. Moreover, this method has the advantages of return of water to the mill direct, no accumulation of liquors, and lower working costs.

Under the second sub-division, where the gold is brought into solution in agitation vats, presses are used as filtering medium only. The method is suitable for the treatment of slime from an ore which has been dry-crushed, or which contains but little water, and consequently requires a longer contact with cyanide solution in order to extract the gold. It cannot, however, be recommended for the treatment of slimes coming direct from a wet-crushing mill and carrying much water, as in this case the cyanide liquors accumulate to a large extent and have constantly to be run to waste, causing loss of cyanide, water, and to a certain extent, gold. This latter class of slime will be referred to under the head of double pressing.

As already stated, this method is in vogue at the Brownhill, Ivanhoe, and Great Boulder Company's Mines. At the former the dry-crushed material from the ball mills is divided into three grades by means of fans—coarse, medium, and fine. The first is ground and amalgamated in pans, the slime thus produced being agitated with cyanide and pressed. The second is leached in vats, and the third or slime portion is agitated in vats and pressed. By this means, very excellent results are obtained, and indeed no mine on the field obtains a closer extraction than the Brownhill, although the working costs are said to be high and the wear and tear great.

At the Ivanhoe much the same course is pursued, except that the ore is wet-crushed. The coarser sands are saved by means of a spitzlutte, and are amalgamated in pans. The medium sands are treated by cyanide in vats, and the fine slimes agitated and pressed. This method gives good results, and the only exception which can be taken to it is in the treatment of the slime, which should be double pressed instead of single. The slime as it comes from the mill, after being settled to a certain extent in tanks, contains about 50 °/₀ of water, while that leaving the presses contains only 30 °/₀; as a result the liquors naturally accumulate to the extent of about 20 °/₀, and have to be run to waste, causing a loss as already described.

At the Great Boulder slimes plant the material dealt with has been lying exposed to the air and the weather for some years, and has consequently lost much of its moisture, so that here they have no difficulty in agitating and pressing the slime in the ordinary way.

Double Pressing.—This method, as already indicated, is the one employed for the treatment of slime which has been wet-crushed, and consequently requires a long contact with cyanide in order to dissolve the gold. Although slightly more costly as regards working expenses than single pressing, it is the one most likely to give the highest extraction.

The method of procedure is as follows:—The finely pulverised slime is forced by means of compressed air or by pumps alternately into one of two presses set above two large agitation vats (which serve, to a great extent, also as sumps), where the water is separated from the slime and returned to the mill or the pans, as the case may be. The press being now full of slime containing about 30 °/₀ moisture, is now opened, and the slime discharged into the vat below, where it is quickly disintegrated by means of agitation arms kept in continuous slow motion, and becomes thoroughly mixed with the cyanide solution in the vat, causing the gold to be dissolved. From these vats the pulp, which now consists of slime and solution in about equal parts, flows into one of two montejus tanks, from which it is forced into a second series of four presses, where the cyanide solution is separated from the slime. The charge is washed and dried in the press, and discharged into trucks and dumped, while the auriferous cyanide liquor and washes are conveyed through the zinc precipitation boxes back to the agitation vats, or to a small sump which is attached to the plant for washing purposes.

It will be noticed that the extra working cost of double pressing over single pressing is in connection with the filling and discharging of the two first presses for the separation of the water. It is estimated, however, that this cost will not exceed 1*s.* per ton at the outside, so that, taking into consideration the conservation of water, less loss of cyanide, and the probable saving of at any rate a few grains more of gold per ton, this cost is soon defrayed.

The great beauty of the filter-press process lies in its capacity for handling large bodies of slime in a neat and clean manner, and in the absolute control which can be exercised over the substance under treatment.

The chief objection to the process is its high working cost, and, on this account, it seems doubtful whether it will ever vie with decantation for the treatment of low-grade slime. No doubt, however, in a country where labour is cheap and water plentiful, the cost of filter-pressing could be considerably reduced.

CHAPTER VI.

The Treatment of Sulpho-Telluride Ores.

THE KALGOORLIE sulphide ores, which form the subject of this chapter, contain considerable percentages of tellurium minerals, and of lime, alumina, and magnesia salts, in addition to the usual constituents of gold-bearing reef matter; and to this characteristic feature is due the trouble which has arisen in the successful treatment of the same.

Typical analyses of Kalgoorlie sulphide ores give—

Silica, about	50 per cent.	
Iron	10 „	
Alumina	5 „	to 20 per cent. or more.
Magnesia	1 „	„ 5 „
Sulphur	3 „	„ 7 „
Copper	0·1 „	„ 0·3 „
Lead	Traces.	
Zinc	0·02 per cent.	
Arsenic	Traces.	
Antimony, about	0·02 per cent.	
Tellurium	0·03 „	to 0·1 per cent.
Calcite	6 „	„ 17 „

in addition to variable amounts of sodium salts, oxygen, moisture, &c. The sulphur present as sulphide seldom exceeds 4 °/o. A small percentage (about 2 °/o) of soluble salts, such as the sulphates of sodium, aluminium and magnesium, is also present. The iron pyrites varies from 3 to 7 °/o.

It will thus be seen that the presence of so large a percentage of calcite puts chlorination out of the question, except for concentrates, and that the ore is peculiarly liable to form slimes, and that these factors have to be very carefully considered in arranging for the necessary treatment. The ore when ground in a ball mill to pass a 30-mesh sieve gave 22 °/₀ retained on a 40-mesh, and 49 °/₀ passed a 90-mesh; when ground to pass through a 60-mesh sieve, 24 °/₀ was retained on a 90-mesh, the balance passing through. The finest portions assayed highest.

A note of some of the tests carried out on these ores will possibly be of interest, as showing how the methods adopted for actual treatment have been determined.

An amalgamation test on a raw finely ground "telluride" sample, one of the first sent to this country (it was really a slightly oxidised sulphide ore, containing about 12 °/₀ of pyrites, and 0·03 °/₀ of tellurium), showed that 20 °/₀ only of the gold was recoverable by ordinary methods. By roasting the ore and adding treatment in pans, the extraction was increased to 44 °/₀. Preliminary treatment of the ore with strong alkali, and also with nitric acid, to dissolve any tellurous acid coating from the gold particles, failed to improve this extraction. It may be noted here that the result of many experiments showed that roasting could be carried out in spite of the very volatile nature of the tellurium minerals, with practically no loss of gold beyond that carried away mechanically in the dust.

Ordinary cyanide bottle tests on the raw ore showed an extraction of from 60 to 77 °/₀ for 16½ hours' agitation. The residues of the 60 °/₀ experiment when roasted yielded a further extraction of 33 °/₀ (total 93 °/₀) to cyanide treatment.

Concentration did not give good results. Good separations of the sulphide were made from the gangue, but the latter still contained a considerable percentage of finely ground, almost impalpable, tellurides, which carried heavy values, necessitating further treatment. The removal of the concentrates, 16 °/₀ of which carried 23 °/₀ of the total gold, did not render the tailings any more rapidly amenable to ordinary cyanide treatment.

By percolation of the raw ore with cyanide solution and double treatment an extraction of 81 °/o was obtained in the two treatments, the first period being for ten days and the second for seven days.

In all these trials there was a rather heavy consumption of cyanide, mainly due to the large percentage of copper (0·029 °/o) and of iron (0·04 °/o) taken up by the solution.

The result of all the cyanide tests was to show that fine grinding (to pass a sieve of 8,100 holes to the sq. in.) was absolutely essential to success, and that agitation or intimate contact with the solution was also necessary.

On roasting the ore there was a loss in weight of about 4 °/o, varying with the analysis of the ore. An ordinary agitation bottle test on the roasted ore gave an extraction of 87 °/o in 16½ hours.

From the above test it appeared that agitation of the finely ground raw ores with cyanide solution should be effective for ores containing even large quantities of sulphides, but not exceeding a small percentage of tellurium, and of a value of 1 oz. of gold or under; as for this value, very high percentage extractions are not of such moment as low working cost; and that ores of lower grade could be profitably treated by percolation with re-treatment, provided it is found feasible in practice to keep the cyanide consumption due to the copper and iron at a low point—as for instance by the use of dilute solutions. Richer ores could be treated by roasting, with or without amalgamation—the use of the latter process depending on the nature of the gold in the roasted material.

A further set of experiments was carried out on various ores carrying a greater percentage of tellurium, in which the results due to the employment of aëration, of bromocyanide, and of roasting were investigated, and an attempt made to obtain a total absolute extraction for the sake of the information to be thus secured, even at the cost of the disregard of commercial considerations. These showed conclusively that the results obtained depended on the amount of gold-carrying telluride present as contrasted with iron pyrites, and also on the hardness of the vein material; and that the conclusions deduced from the first set of trials required to be modified by the need shown for still finer grinding, for the employment of an accelerator with agitation, or for the roasting of the ores.

These experiments were as follows, the first ore used being the sulphide (0·03 % tellurium) ore referred to above:—

(*a*) Percolated for 37 days with eight changes of 0·5 to 0·03 % of simple cyanide solution. Extraction obtained 94 %, the final residues being reduced to 7 dwts. 4 grns. of gold per ton. During the progress of this test the charge was constantly aërated, and air was introduced under pressure under the filter bottoms. Freshly precipitated lead salt was also added to counteract the retarding action of the small amount of soluble sulphides formed by treatment of such length and duration.

This is practically the method—without the air—by which the African Spitz concentrates containing about 7 % of sulphides are successfully treated at the majority of the mines at Johannesburg.

(*b*) Another portion was percolated for 24 days with cyanide solutions of from 0·5 % to 0·04 % KCy, with the addition of bromocyanide equal to about one-sixth of the cyanide present, and the solutions were circulated through the charges as rapidly as possible. Fresh charges of bromocyanide were usually added every 6 to 10 hours. In all, nine charges of solution were used. The total extraction obtained by the bromocyanide treatment was 93·4 %, but the speed of the earlier extractions was very marked, an extraction of 82 % being obtained from the unroasted ore in four days, as against under 50 % by plain cyaniding in six days.

Similar tests were also carried out on the ore after roasting for an extraction of 93 % by plain cyanide in 29 days, and 85 % by bromocyanide in 24 days. In this case, on the roasted ore, the accelerating advantages of bromocyanide were not so marked.

A further test with plain cyanide, the strongest solution used being

0·3 % of KCy, on the same ore ground to pass a 60-mesh sieve, yielded an extraction of 80 % in 25 days, as against 93 % for a similar trial—that is, with plain cyanide—for the same period on the roasted ore.

In these tests it was noticed that the roasted ores contained a large percentage of soluble salts, and that there was a *heavier consumption of cyanide on the roasted ore*, and the necessity was thus deduced for some intermediate treatment between roasting and cyaniding to wash out, or get rid of, the soluble salts. The blowing of air into vats between the filter bottoms and into the solutions did not accelerate or improve the extraction on either the raw or roasted ore, provided the solutions were circulated with the ordinary rapidity.

On attempting to obtain these results with other ores, however, carrying a heavier percentage of tellurium, it was at once seen that special treatment would be necessary.

Thus a rich ore from one of the Kalgoorlie mines crushed to pass through a 60-mesh sieve yielded 66 % of its value to the first solution, and then gradually less and less, until after a treatment extending over two and a-half months, an extraction of 90 % only had been obtained, though the residues still assayed over 2¾ oz., and were yielding up their gold at the rate of about 1 dwt. per ton per diem.

In this case bromocyanide was constantly added after the treatment had lasted six weeks, and was responsible for a considerable spurt in the extraction for the first few days, but it failed to improve the extraction beyond that shown above. The same ore ground to pass 90-mesh, yielded an extraction of 90 % for six weeks' treatment.

These trials showed that it was not possible on these rich ores to get a final extraction within a reasonable time, except by roasting, or other special means.

An investigation was then made as to the cause of the lengthened percolation period required by the roasted ore, and it was found that practically all the extraction freely obtained was obtained in from three to seven days, the balance of the time being required to slowly bring down the residues to the required value. On treating a similar roasted sample for three days only, and grading and concentrating the residues, it was found that practically the whole of the gold was in the *coarse sand residue*, and that the slimes of a 6-oz. ore had been reduced to under 2 dwt., and even after concentrating these slimes to an oxide of iron slime, the latter was found to be practically gold free. The necessity for, and the value of, fine grinding was thus doubly evidenced.

It was thus clearly shown that, given roasting and sliming, even rich ores could be readily and rapidly treated, provided mechanical means were adopted to overcome the difficulty of leaching a mass of slimes.

To avoid roasting, however, other methods were tested, and will be referred to later, including that of agitation, which, although yielding an extraction of from 60 to 77 °/o on the above-mentioned ore (the "sulphide" ore), gave considerably lower results on the more difficult ores subsequently tested, the extraction falling to from 40 to 45 °/o, even after the ore was ground to pass 90-mesh, and to 46 °/o for agitation and amalgamation combined. On roasting this ore, however, an extraction of 88 °/o was obtained with 16 hours' agitation, and 96 °/o when the treatment was prolonged for 41 hours. The latter result was also obtained by roasting the residues from the previous raw agitation and grinding them to pass through a 90-mesh. Amalgamation extracted from these residues 19 °/o, and subsequent cyanide treatment of the pan tailings 71 °/o, thus bringing the final residues of an ore originally assaying 7 oz. down to 2¼ dwt.

These figures are given with some detail, as the ore referred to has now become historical, and is daily yielding extractions actually confirming those obtained in the above experimental treatment.

It was noticed in these tests that all the way through the agitation method showed much higher results than that of percolation.

In view of the more rapid results of bromocyanide on unroasted ores experiments were carried out on samples ground to almost impalpable powder. These results will be referred to at the latter end of the paper when considering the Diehl process, but in sequence of the tests given above, the method of treatment to be first noticed is that of—

1. The Roasting and Sliming Process.

In putting into practice this roasting-sliming method of treating the ores difficulties have arisen due to the bad working of the furnaces, to the abnormal amount of soluble sulphates formed in the ores, to the swelling of the ore in the vats, and to the necessity of having the ore exceedingly finely ground, and of making provision for the amalgamation of any gold possibly fused during the furnacing.

These difficulties, one by one, have been overcome as they arose, and extractions of 93 to 94 °/₀ are now being daily obtained from the treatment of sulphide ores by this method at the Great Boulder Main Reef and other mines.

The inefficient working of furnaces was a case for the adaptation of the designs and *modus operandi* to the peculiarities of the ore to be treated; the abnormal amount of soluble sulphates which caused so heavy a consumption of cyanide has been got rid of by passing the pulp from the grinding pans through filter-presses, or spitzlutten, before cyanide treatment. Pans have been installed to grind the ore to the required fineness, and although these pans also serve to amalgamate coarse gold, it must not be overlooked that their function in this case is one of grinding rather than of amalgamation—in fact, some of the gold refuses to amalgamate even in a pan, and it may be found advisable with rich ores to concentrate or settle this out of the pulp. The difficulty of the percolation and filtration of solutions through ore so finely ground has been obviated by the introduction of agitators and of filter-presses, which have for some time past been so successfully employed in the treatment of oxidised slimes.

This process has now been adopted by a large number of the best known mines at Kalgoorlie, and the whole scheme is an example of the successful results attending the investigation and rectification of difficulties as they occur.

Naturally in a new process treatment costs are not yet available; they are officially set down as from 20*s.* to 30*s.* per ton, of which 4*s.* to 8*s.* is for roasting and 6*s.* to 8*s.* for filter-pressing. Some time since the author, by the courtesy of Mr. H. C. Callahan, published details of the cost of filter-pressing at the Lake View Consols, but it must not be assumed that because the filter-pressing process—including superintendence, discharge and filling, compressed air, zinc, repairs, assaying, retorting and melting, filter-cloth, cyanide, general stores, and electric light—at that time cost 6*s.* 6*d.* per ton, double filter-pressing would correspondingly increase this figure; on the contrary, as the first operation is solely one of expressing superfluous moisture, laden with soluble salts, and does not include cyaniding and washing, it is stated by Mr. J. K. Wilson, formerly of the Golden Horseshoe, that two men per shift and two presses will readily deal with the moisture of 100 tons daily, and that the extra cost over that for ordinary filter-pressing should not exceed 1*s.* per ton; of course where spitzkasten are substituted for these presses, the cost of this operation is merely nominal. On the other hand, the saving in cyanide due to the pressing out of the soluble salts is enormous.

The cyanide consumption by this method is from 1 to 2 lb. per ton treated, and the residues assay from 1 to 2 dwt., varying with the value of the original ore.

The pioneer plant—that of the Great Boulder Main Reef—consisted of—

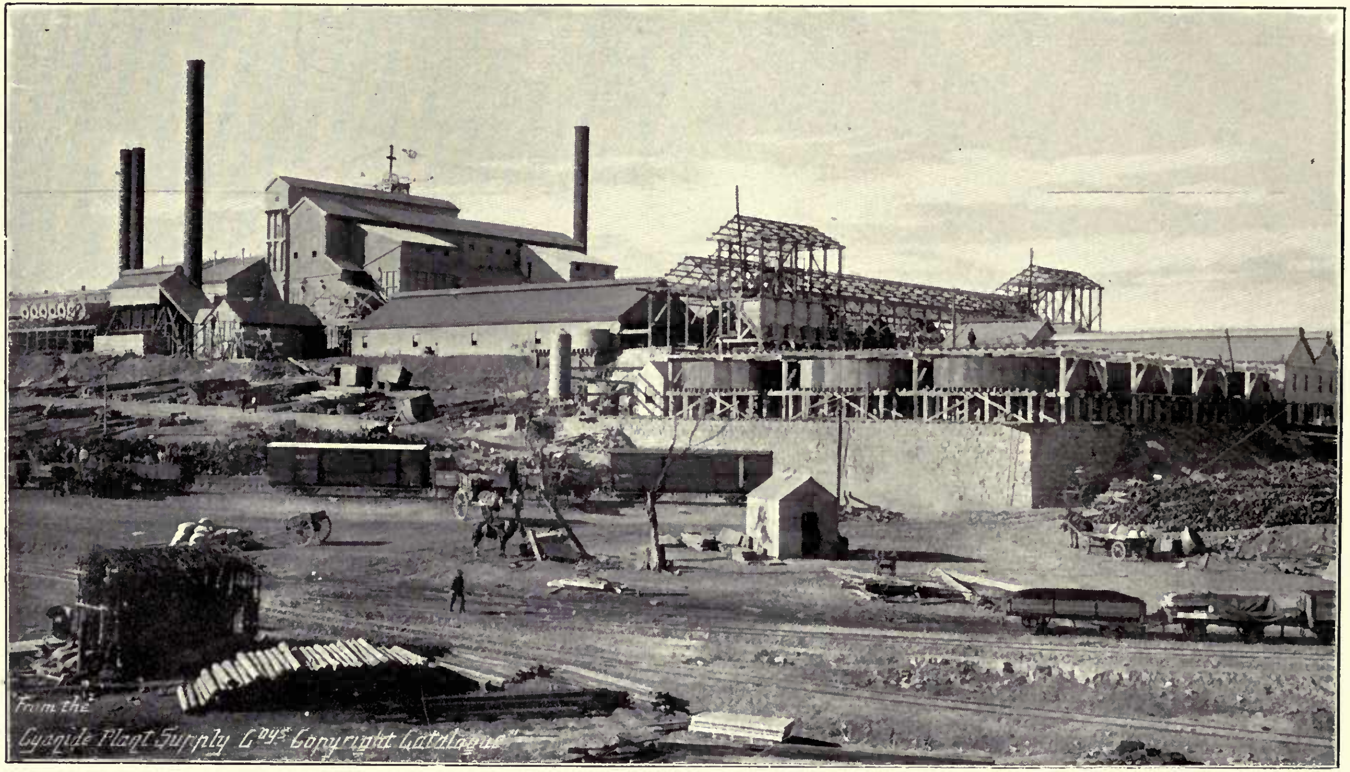
Ball mills.
Multiple hearth vertical furnace.
Tailings wheel.
Pans (5-ft. diameter).
Filter-presses.
Agitators.
Filter-presses.
Precipitation boxes.

But Edwards' furnaces have since been installed in preference to the vertical hearth type.

When old slimes are treated, these are broken up in a special tank called a breaker or mixer. It is about 4-ft. deep by 5-ft. diameter, with a pierced internal cone with a revolving screw or vane at the bottom. The pulp is thus drawn through the cone past the rapidly circulating vanes, and returned through the holes in the cone to the upper portion of the vat. The actual agitators are much larger than this, varying from 10-ft. to 21-ft. in diameter, and are from 5-ft. to 16-ft. deep. They are usually fitted with a vertical shaft revolving by a bevel gearing at the top, and carrying two sets of radial arms at the bottom. Other methods of agitation are a swiftly revolving pipe of, say, 18-in. diameter, fitted with internal helical-curved (or rifled) vanes, which force all the pulp against an iron plate on the bottom of the vat. Another most efficient method is to use a centrifugal pump.

As an instance of a recent type of plant for this method, details are given herewith of the installation being erected by the Great Boulder Proprietary Company, which consists of:—

1 No. 8 Gate's breaker.
4 H Gate's fine breakers.
10 Griffin mills.
12 Edwards' roasting furnaces.
12 Grinding pans.
6 Settlers.
1 Three-throw pump.
Filter-presses for eliminating moisture.
Cone mixer or breaker.
1 Three-throw pump.
Central agitation vat, 14-ft. diameter × 16-ft. deep, with radial arms.



From the
Cyanide Plant Supply Coy's Copyright Catalogue"

THE LARGEST CYANIDE PLANT IN AUSTRALASIA.
View of the Western half of the Sulphide Plant of the Associated Gold Mines of Western Australia.

12 Other agitation vats, of sizes varying from 15-ft. to 20-ft. diameter × 5-ft. deep.

Monteju.

Filter-presses.

Extractor boxes.

In working this process it is found that three-throw pumps for filling the presses save heavy cost for compressed air, but that montejus tend to give better cakes.

To distinguish the above process from that which follows, it may be conveniently named after its pioneer, and will therefore be subsequently referred to as the "Marriner" process.

2. The Concentration-sliming-bromocyanide Process.

The other process prominently before the metallurgical world at the present moment is named after Dr. Diehl, and involves the use of Sulman's bromocyanide, plus fine grinding, agitation, filter-pressing, and concentration. It avoids the roasting of the ores necessary to the Marriner process, wet crushes, amalgamates, concentrates, separates the sands and slimes, regrinding the former to slimes in tube mills, and then agitates and filter-presses. As the process has only recently been put into operation, detailed costs are not yet published, neither are precise details of the methods employed. The scheme at present appears to be incomplete and without provision, otherwise than by smelting or roasting (with subsequent chlorination or cyaniding), for the treatment of the 10 °/₀ or so of concentrates produced.

The difficulty of obtaining a high percentage of the value of the concentrates from telluride ores has been already referred to in the results given above. The author has, however, carried out the following experiments on the most refractory sample of telluride ore at hand—which happens by a coincidence to be that of the Hannan's Star Mine, at which Dr. Diehl first installed his process, and where the first run of 250 tons was reported to have yielded an extraction of 90 $^{\circ}/_{0}$, subsequent results being even better.

The object of these experiments was to test the effect on an unroasted telluride ore of agitation, excessively fine grinding or sliming, and bromocyanide treatment.

Test 1.—The ore ground to pass through a sieve of 14,400 holes per sq. in. (120-mesh) was agitated with 0·4 $^{\circ}/_{0}$ cyanide solution for 23 hours. The extraction obtained was 51 $^{\circ}/_{0}$. The residues were similarly re-treated for an extraction of 10 $^{\circ}/_{0}$. Further re-treatment yielded 4 $^{\circ}/_{0}$, and the final treatment gave 3 $^{\circ}/_{0}$, or a total extraction of 68 $^{\circ}/_{0}$ for four agitation treatments, each of 23 hours.

Test 2.—Ore prepared precisely as in the previous experiment was agitated for 23 hours with 0·4 $^{\circ}/_{0}$ cyanide solution, to which 0·04 $^{\circ}/_{0}$ of bromocyanide had been added. Two further similar additions of bromocyanide were made, each at intervals of two hours. The extraction obtained was 91·5 $^{\circ}/_{0}$, and as re-treatment of the residues failed to increase the extraction, although they still assayed 11 dwt., it is probable that the amount of bromocyanide used might have been reduced for the extraction obtained.

Test 3.—This was with bromocyanide, as in the previous experiment, but on the very finest portion of the ore only, viz., that which floated off by gently moving water. These levigated slimes assayed 4¾ oz., as against 6½ oz., the original value of the sample; but even with such extremely fine material the results obtained by bromocyanide treatment were in no way increased beyond those obtained in the previous experiment.

Test 4.—This was on the ore ground to pass 40-mesh only (1,600 holes per sq. in., as against 14,400 holes per sq. in. for the first experiment). The extraction obtained by repeating Test 1 on the ore was practically identical, being at the rate of 52, 9, and 4 $^{\circ}/_{0}$, or a total of 65 $^{\circ}/_{0}$ for three treatments which is the exact extraction obtained in the same time in the first experiment.

Test 5.—This was similar to No. 4, but with the addition of bromo-cyanide, as in No. 2. The extraction obtained was 78 % only, showing the necessity for very fine grinding to fully utilise the intense energy of the bromocyanide solutions. It is a remarkable fact that although the plain cyanide solution appeared unable to take advantage of the finer grinding, it made the greatest difference to the results when bromocyanide was used.

To resume, on comparative trials of the ore ground to pass a sieve of 14,400 holes to the square inch, bromocyanide extracted 91·5 % in a treatment of 23 hours with final residues of 11 dwt., which could not be lessened, as against an extraction of 68 % only for four treatments of 23 hours, each with plain cyanide with final residues of over 2 oz.! With ore ground to pass a sieve of 1,600 holes per square inch only, bromocyanide failed with similar treatment to extract more than 78 %. The consumption of cyanide is apparently rather less with bromocyanide than in the longer treatment with simple cyanide, but, on the other hand, there is an additional cost for the bromocyanide used, which may be a very considerable item.

It is evident from the above results, that even without the separate treatment of the concentrates, Dr. Diehl may be able with very fine grinding—120-mesh—and agitation with bromocyanide, to obtain the 90 % extraction on this ore, as claimed by him, but it is a moot point as to whether he will be able without concentration to reduce the tailings of high-grade ores to as low a point as those yielded by the Marriner process. On low-grade ores, and those of up to 1 oz. in value, this is not a matter of considerable moment, provided working costs are less, but on richer ores it becomes of importance, and this is probably the reason for the concentration problem to which Dr. Diehl has set himself—an elimination of a rich refractory or long treatment product for future roasting or smelting might simplify considerably the treatment of the remaining pulp.

It is too early as yet to contrast the difference in working cost of the "Marriner" and "Diehl" processes; the latter starts with an initial saving of from 2*s.* 6*d.* to 7*s.* a ton (depending on the type of furnace used) for roasting charges, but labours under the disadvantages of charges for royalty and for bromocyanide, the latter of which alone may readily amount to as much as the roasting; necessity for finer grinding; heavier cyanide consumption than with the roasted ores, due to the reaction of bromocyanide, and of the sulphides, copper, and metallic iron contained in the pulverised ore; smelting charges, or the probable loss of gold already experienced in roasting the concentrates charged with the salts found in the Kalgoorlie water, or higher charge for fresh water; and, finally, if concentration be not employed, higher residues than those given by the Marriner process. That both processes can be made to produce 90 % results need no longer be doubted; but the question is also one of cost, and it will be interesting to contrast the expenses of the bromocyanide process with the costs of the Great Boulder Main Reef Company.

Probably it will be found feasible in practice, when the number of plants at work tends to produce a spirit of healthy rivalry, to improve both processes and to lessen the costs, but there is no reason to doubt the problem of the successful treatment of the Kalgoorlie sulpho-telluride ores may be considered as already solved.

These notes are not intended to give an exposition of the details of the apparatus employed, but to show concisely the general methods involved in solving the sulphide problem. The furnaces which are at the present moment known to be doing good work on telluride ores are the "Edwards," the "Argall," the "Ropp," and probably the "Brown Straight-line." The original furnace at the Great Boulder Main Reef was of the well-known "Mount Morgan" multiple hearth type.

The roasting costs at Cripple Creek on telluride ores are under 2*s.* per ton, including fuel, cooling, conveying, labour, and lubricants. The chief item is for fuel, which is 160-lb. of slack coal (at 8*s.* per short ton), or say 7¾*d.* per short ton, or 8½*d.* per long ton of ore roasted. If we take Australian wood fuel at 9*s.* a cord, which is the price paid by the Great

Boulder Main Reef at the time of writing, we find that 1 cord of fuel weighing about 35 cwt. equals 1 ton of Collie coal, or 15 cwt. of Newcastle coal. This, on the above lines, should roast at least 9 long tons of Kalgoorlie ores, which gives a fuel cost of 1*s.* per ton as against the American cost of 8½*d.* The actual consumption of timber wood fuel at the Great Boulder Main Reef is stated to be 1 ton (or 20/35ths of 9*s.*) to 6 tons of ore roasted; this is less than 1*s.* per ton. But assuming that the new contracts for fuel will be at 22*s.* 6*d.* per cord, or 13*s.* per ton, this brings the fuel cost up to 2*s.* 2*d.* per ton of ore roasted.

In Colorado, men attending to furnaces are paid 12*s.* per shift, and the cost of labour amounts to less than 6*d.* per ton roasted with the best installations. Lubricants cost 1*d.* per ton. Repairs vary with the amount of ironwork used in the interior of the furnace; but in the best plants this item has been reduced to less than 1*d.* per ton. It is, therefore, not unreasonable to expect that roasting will be done at Kalgoorlie at 2*s.* 6*d.* per ton, as it is scarcely seriously claimed that the furnace originally erected at the Great Boulder Main Reef is the best or most economical for the purpose. The official statement of the results of this furnace places the roasting costs at 8*s.* a ton.

The following are the published official figures of the costs at the Great Boulder Main Reef:—

	£	*s.*	*d.*
Crusher			3·33
Ball Mill		1	6·45
Furnace		8	3·41
Grinding		1	3·67
Filter-Press		4	3·59
Cyanide		5	10·47
Steam		2	8·37
Smelting			3·62
Assays			2·33
Water and Condenser			10·11
Management		1	0·23
	£1	6	7·58

The result of the more recent working has been to reduce these costs to 21*s.* per ton, the consumption of cyanide now being only 1*s.* 3*d.* per ton!

Roasting charges, too, are reduced to under 3*s.* a ton—already a close approximation to the estimate of 2*s.* 6*d.* in the preceding paragraph.

It is more difficult to give the cost of the bromocyanide (Diehl) process. In the cheapest case, "Hannan's Star," on a low-grade ore it is stated by the owners of the process that the costs are as low as 23*s.* per ton. The consumption of bromocyanide is understood to be ½-lb., of potassium cyanide 2-lb., and of lime 5-lb. per ton of ore treated.

At the "Brownhill" and "Lake View" Companies the consumption of bromocyanide is much greater, amounting to as much as 2-lb. per ton —*or say 7s. 6d. per ton for bromocyanide only*—and royalty, lime, and cyanide have to be added to this, in addition to the crushing, sliming, concentration, agitation, and usual treatment costs.

Moreover, a factor against the Diehl process is that the concentrates, which at Lake View carry as much as ⅓ of the total gold recovered, though reckoned in the extraction percentage, have to be realised subsequently by special treatment, and some of the mines have laid down roasting furnaces for this purpose.

THE RIECKEN PROCESS.

A further process has been introduced at the "South Kalgurli" mine, called the Riecken process. This is similar to the Pelatan-Clerici mentioned in the chapter on the Treatment of Slimes under the heading of "Precipitation." It proposes to treat the roasted ore in a bath of cyanide solution, plus the salts already in the mine water, and the gold is then precipitated by an electric current on to a copper cathode over which mercury is continually flowing. In practice it is found, however, that filter-pressing and roasting are both necessary, and the process is thus more costly than either of the preceding.

The owners of the Riecken process are understood to claim to be able to treat the "South Kalgurli" ores at a cost of 28*s.* per ton, but there is no statement showing that this has yet been accomplished. At the present moment the employment of roasting, electric precipitation, and filter-pressing means one process more than either the Marriner or Diehl method of treatment, and it is probable that the next move of the Riecken patent owners will be to modify their process—possibly by avoiding roasting.

It thus appears that with its extraction of 93 % to 94 % at a cost of 21*s.* per ton, the Marriner process is showing better results than any of its competitors ; and a great point in its favour is that it requires less power. This at Kalgurli is a very heavy item in the total costs at present, 1 H.P. costing £50 to £70 per annum, or 3*s.* to 4*s.* per diem. Thus a plant treating 20 tons per diem, and requiring 60 H.P., would show a cost of 180*s.* to 240*s.* per diem, or 9*s.* to 12*s.* per ton for power alone.

It must not, however, be lost sight of that an inexpensive method of eliminating the long-treatment product—the tellurides—would considerably cheapen the cost of treatment of the Kalgoorlie ores, by doing away with the heavy expense of the roasting, sliming, or other special methods at present found necessary. Ordinary concentration has not been found effective for this purpose, but if a new method acting on other principles—the Elmore oil-separation process for instance—should successfully accomplish this object, treatment costs might at one step be reduced to ordinary cyanidation costs, plus concentration and realisation of concentrates.

This is an aspect of the question deserving the serious attention of directors and their advisers.

CHAPTER VII.

Dry Crushing.

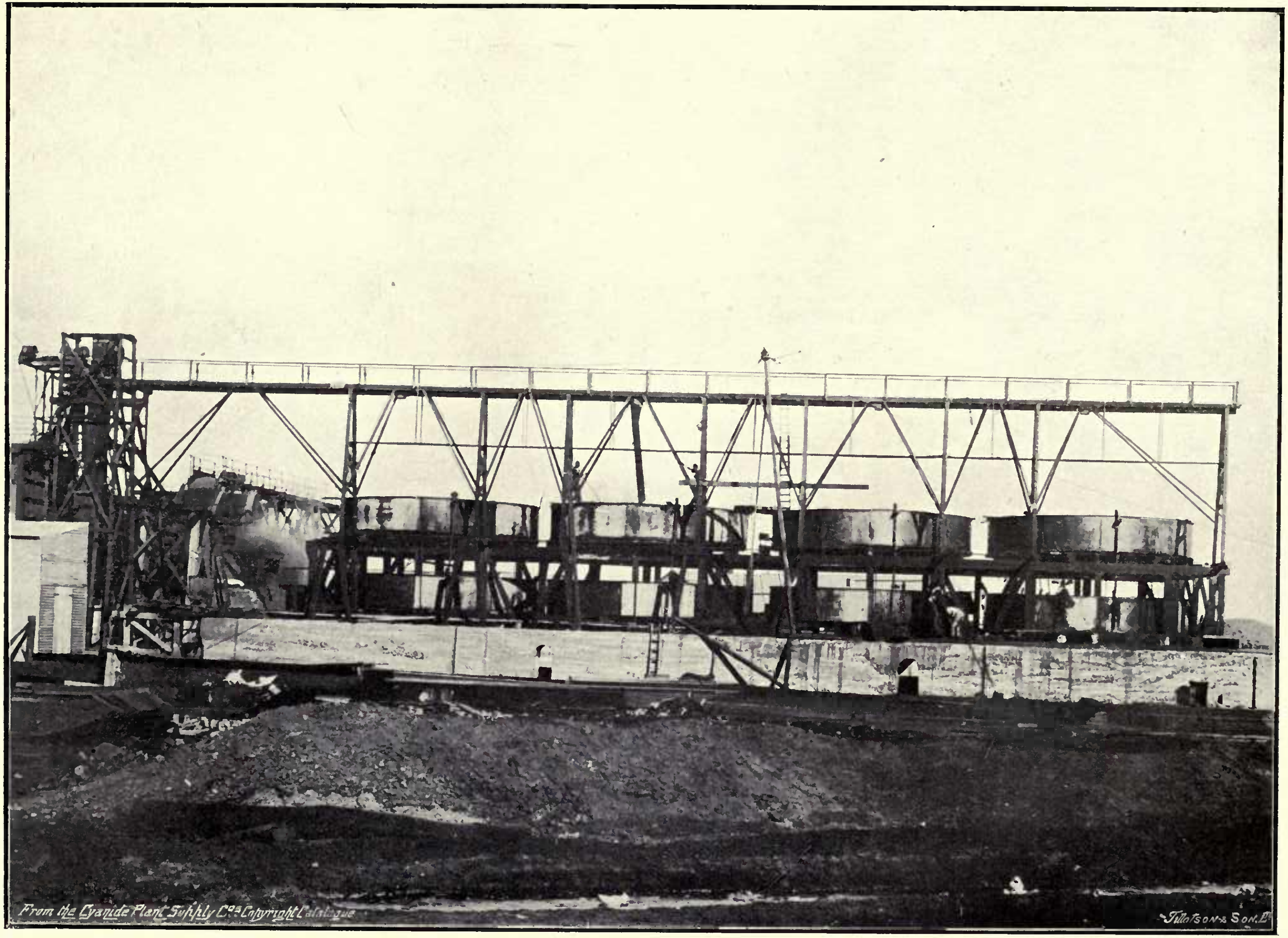
FEW who have not specially studied this subject are aware that a very large proportion of the total output of gold at the present day—the proportion being all the greater on account of the temporary shutting down of the African mills—is obtained by *dry crushing*.

One has only, however, to refer to the installations at such well-known mines as "Mount Morgan," "Waihi," "Mercur," and "Hannan's Brownhill" as evidence of the mammoth plants—some of them capable of handling more than 20,000 tons per month—which are utilising the dry-crushing process for the treatment of their ores.

One of the results of such huge installations has been the demand for machinery capable of effectively dealing with this problem. The recent "Waihi" report shows that, with wet and dry stamp mills *working on the same ores*, dry crushing is cheaper by 9·58*d.* per ton, there being an increase in the wear of the stamps and screens in the wet process, as well as of wages.

But even more favourable is the comparison of the results achieved by stamps (the admitted best wet-crushing appliances) as against ball mills, which have recently been doing such good work dry crushing in Australia and elsewhere. Thus the best African practice (wet crushing) shows 1½ tons crushed per H.P. (5½ tons crushed for 3½ H.P.), whereas in Australia the average output of the ball mills is over 2 tons per H.P. (65 tons crushed for 30 H.P.) on hard ores; and the ball mill output is purposely crushed through much finer screens than that of the African mills; so that, for equal size of screens, the advantage of the ball mill would be still more marked.

The object of these remarks is not, however, to refer to the large



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Tillotson & Son. Ld

HANNAN'S STAR DRY CRUSHING CYANIDE PLANT.

installations of comparatively rich mines, but to the *remarkable results obtained from the dry treatment of ores too poor for wet crushing.*

It has been the usual practice to consider that mines generally—with, of course, notable exceptions where huge quantities are to be handled—should have ores of an average grade of at least 12 dwts. (depending on the position, depth, labour, &c.) to afford profitable exploitation; but the new system of coarse dry crushing and cyanidation—applicable, of course, to oxidised or porous ores only—is *realising handsome profits on ores containing only* 6 *dwts. and under.*

A knowledge of these results will render valuable vast deposits of material hitherto regarded as too poor for profitable operation, and recent experience shows that it is not even necessary that the ores should be oxidised, as deposits containing large percentages of sulphide of antimony are now being worked most successfully by this method, although these have such large base metal contents, as to make the cyanide consumption absolutely prohibitive when fine crushing was attempted as originally proposed.

It is, of course, necessary that reliable expert opinion should be obtained as to the method of crushing and treatment to be employed for each particular ore, and it is confidently expected that this process will prove particularly suitable for the large deposits of oxidised reef on the West Coast of Africa.

It is not intended that the coarse crushing referred to in these notes should be accomplished by stamps (though the recent "Waihi" report shows that fine crushing with stamps is actually cheaper than wet crushing), but the disintegration would be effected by suitable means such as rock-breakers of either the Gates or Blake type, followed if necessary by finer crushers, such as rolls or the Dodge crusher.

Of course, where quite fine grinding is used, as already stated, the ball mills are pre-eminent, and in cases where the ore is to be ground to a flour, cement mills, such as the tube or grit mill and the Griffin mill, are employed with success, but all the latter equipment would be quite out of place on a low-grade proposition.

As evidence, however, of the wonderful results obtained, it may be mentioned that in South Africa, at "Barrett's Berlin," the treatment on 6-dwt. ore yielded continuous profits up to the commencement of the war in South Africa, as much as £1,800 profit in a month resulting from the treatment of 3,000 tons of 6-dwt. ore, and a number of similar low-grade propositions will probably be operated on a very huge scale so soon as the war is over.

In South America, at Santiago, 5-dwt. stuff has similarly been treated at a profit, and in New Zealand a considerable amount of work has been carried out on the same lines.

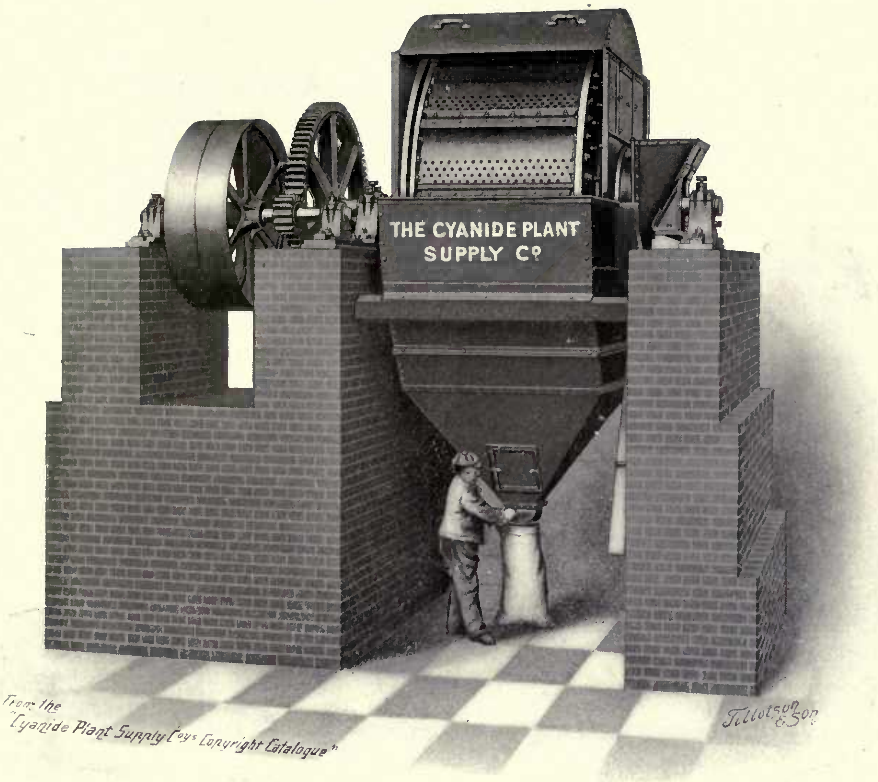
It is in Eastern Asia, however, that at the moment some of the largest and most profitable developments of this method have taken place, and large profits are being made from the treatment of material of only 3 to 4 dwts. value, the total cost of mining (quarrying) and treatment, including superintendence and other expenses, being under 6*s*. per ton.

With such figures before us it may be readily understood that even very low-grade deposits (given favourable conditions) may be made to produce most handsome profits.

Illustrations are given herewith of the dry fine crushing mills which have recently come into prominent use, and notes are added of their output and power requirements.

Coarse crushers are not illustrated, as Gates crushers, Blake breakers, and rolls are already so generally known as to require no illustrations.

First. The Ball Mill:—This apparently is the best dry crusher ever introduced and gives results unapproached by other dry crushers, taking into consideration the product obtained for the power used.



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BALL MILL.
From a Photograph of a No. 8 Size.

These mills are made with specially hard balls and liners. They have shown exceedingly satisfactory results with West Australian sulpho-telluride ores, on which they yield the outputs shown below. These outputs would be correspondingly greater on softer ores.

These mills are also employed on a huge scale at "Mount Morgan," and for general economy of results obtained, contrasted with H.P. necessary, labour, and repairs, are apparently unapproachable by any other form of dry crusher.

Their best speed for the large sizes seems to be about 24 revolutions per minute, and their output under good working conditions, *i.e.*, regulated rate of feed and number of revolutions, may be taken at 2 tons per H.P.

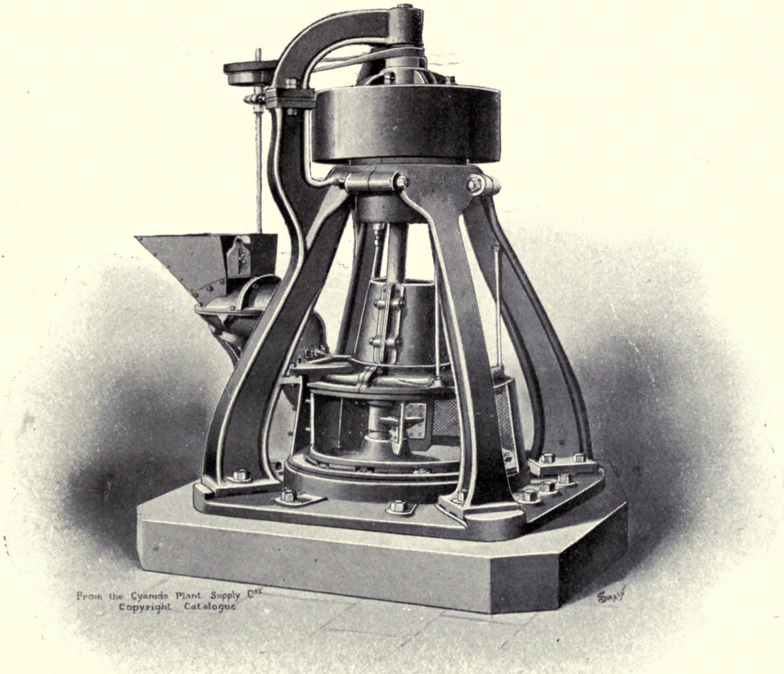
The ores may contain up to 2½ % of moisture, but this figure should not be exceeded.

Mr. Moss gives the cost of crushing by these mills at Kalgoorlie at 9*d.* per ton for repairs and renewals, and 5*d.* per ton for labour. The charge for H.P., of course, is additional to this.

These mills will take material of almost any size up to 6-in. cubes, but best work is done with 2-in. to 2½-in. stuff.

The output varies from 12 to 15 tons per diem for a No. 3, to 50 to 60 tons per diem for a No. 8.

Second. The Griffin Mill.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GRIFFIN MILL.

This mill is in use in the United States and at Kalgoorlie. At Delamar the mill is stated to crush 1 ton per hour from 1-in. cubes down to 50-mesh at 16 cents per ton, power required 25 H.P., but recent practice at Kalgoorlie

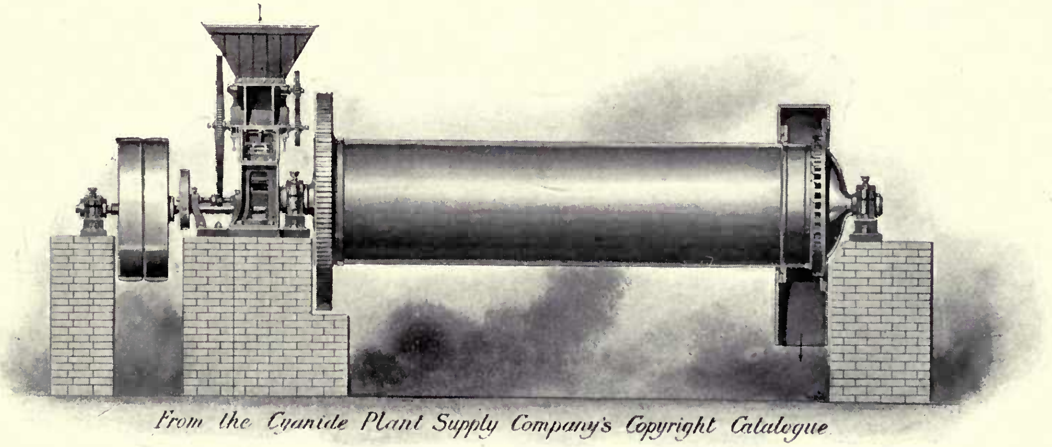
gives an output of 21½ tons per mill per diem, crushed from 1-in. gauge through a 15-mesh screen at a cost for wages and repairs (power not included) of 1*s.* 10*d.* per ton, as against 1*s.* 2*d.* for ball mills.

This mill, apparently, is also inferior to the ball mill in requiring a very even feed of small crushed material (1-in. cube) and in producing a less evenly fine product. It also appears to be more expensive in running cost and repairs. If the feed is too great, or too little, the roller is apt to pound, and to thus cause frequent breakages of the vertical shaft. Moreover, as the feed enters the mill from one side only, there is heavy wear on the roller path liner at this point, making the shifting or replacement of liners frequently necessary, although the rest of the circumference is practically unworn.

Ball mills are therefore recommended in preference to Griffin mills.

The weight of each mill is about 5½ tons.

Third. The Tube or "Grit" Mill.



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TUBE MILL.

These mills are largely used for grinding granular material to a powder of a high degree of fineness, and are specially used to grind ball mill or Griffin mill products to the impalpable slimes found necessary in Western Australia for the highest extractions from the unroasted sulpho-telluride ores.

They use flint balls, and can be used for wet or dry grinding, and are the cheapest known method for the very fine grinding of sands. They require little labour, attendance, or repairs. Power required, 27 H.P.

Output (naturally varying with the hardness of the sand), 6 tons per hour at 30-mesh, or 3 tons per hour at 60-mesh, size of feed 12-mesh; or 3 tons per hour at 200-mesh, size of feed 30-mesh.

CHAPTER VIII.

Costs.

I. *Erection.*—These vary so entirely with the nature of the plant, of the country, and of the individual ideas of those responsible for the erection, that it is scarcely possible to do more than specify a general idea, with some details from the actual costs of erection of various small plants.

Thus, generally, a modern double-treatment steel cyanide plant may be assumed to cost by the time it is erected in Africa or Australia four times its original F.O.B. price; but this figure may be materially diminished or increased according to circumstances.

Wooden and steel single-treatment plants are much less costly and the total cost erected, including original cost of plant, may be as low as double the original F.O.B. price. But where inland carriage is costly the item for this alone may easily be more than the total erection costs of a steel plant.

Plants are seldom landed at the mines for less than £10 per ton for freightage, railage and landing charges, &c. The riveting of steel vats may be taken at £10 per ton in Africa or Australia for the weight of the vats only. Steel vats are of course much more expensive to erect than those made of wood, but they have a longer life, are not subject to the stress of climatic influences, do not require to be covered, and are generally much more satisfactory.

The following are actual costs incurred in erecting small plants in Australasia:—

(1) A 2,500-ton-a-month wooden plant, consisting of 8 22-ft. 6-in. vats, 3 sumps and 1 reservoir; with water power, and sluicing of residues; the whole plant being covered by a building; erected on a level site.

	£
Labour on the erection of vats, foundations, and flooring	367
Timber for the above, including vats	329
Building and excavations for ditto, including materials	368
Tramway	138
Pipework	60
Total	£1,262

This is less than the F.O.B. cost, in spite of the expense of the building; but on the other hand freightage charges are not included.

(2) A 1,000-ton-a-month wooden plant, consisting of 6 20-ft. vats and 3 14-ft. sumps; not under cover except the extractor house; erected on a level site.

	£
Freight, carriage, &c.	246
Labour	175
Timber	230
Tramway	44
Corrugated iron for extractor house	78
Painting vats	25
Sundries	12
Total	£810

or rather more than the original cost of the plant.

(3) A steel plant of 8 22-ft. 6-in. vats, 3 sumps and 1 reservoir, erected on sloping site.

	£
Freight	400
Riveting	400
Extractor house	310
Tramway	80
Painting	35
Connections and pipework	116
Total	1,341

or rather less than the F.O.B. cost; but in this case the foundations of masonry and timber are not included, as these were peculiarly expensive owing to the nature of the ground. On the basis of the preceding plants this item should amount to about £600, which makes the original cost of the plant, freight, and erection complete about two and a quarter times the F.O.B. price.

The above costs, it will be seen, are for single-treatment plants, and they bear out the remarks at the commencement of this chapter as to the outlay necessary.

The cost of double-treatment plants, which vary greatly in design and method of erection, are not given, but the estimate that the cost of an erected plant is about four times the F.O.B. price may be regarded as fairly reliable for ordinary conditions.

2. *Running costs.*—In some cases—chiefly those in which the tailings are quartz sand with free gold—running costs have been reduced to so low a figure that the recovery of ½ dwt. per ton pays all expenses.

The following are costs in various countries :—

In Africa tailings treatment costs 2*s.* 3*d.* to 2*s.* 9*d.* per ton complete.

Labour - - - - -	1*s.* 3*d.* to 1*s.* 5*d.*	per ton.
Cyanide, ¼ lb. to ½ lb., say - -	3*d.* to 6*d.*	„
Fuel and power - - - -	3*d.*	„
Zinc, 2½ oz. to 3 oz. per ton, say -	¾*d.*	„
Lime - - - - -	from nil to 2*d.*	„
Stores and charges - - - - -	2*d.* to 3*d.*	„

Slimes treatment by decantation costs from 2*s.* 6*d.* to 3*s.* 6*d.* per ton.

Concentrates are stated to cost about 7*s.* 6*d.* per ton for the 20 to 30 days' treatment, but these figures are an assumption difficult to verify, owing to the solutions not being kept separate, and the consumption of cyanide is therefore slumped with that of the tailings.

Mr. J. R. Williams gives the cost of handling tailings at 6*d.* per ton, handling slimes at 2*d.* per ton.

At "Barrett's Berlin" the cost of cyaniding dry-crushed ore is 3*s.* 6*d.* pe ton, of transport 1*s.* 6*d.*, of mining 5*s.* Total cost of 10*s.* per ton.

In Borneo the cost of cyaniding dry-crushed ore comes to about 2*s.*, of quarrying 2*s.*, of crushing and transport 1*s.* 6*d.* Total 5*s.* 6*d.* per ton, which is perhaps a record for this class of ore.

At Waihi, New Zealand, cyaniding costs about 4*s.* per ton, but there is a large amount of bullion in the ore treated—consisting chiefly of silver—and the cyanide consumption is necessarily heavy. One cannot expect to treat ores containing ounces of bullion for the same cyanide consumption as that obtained on 4-dwt. tailings.

In Australia generally cyaniding costs from 2*s.* 8*d.* to 5*s.* per ton, and even 8*s.* in Western Australia, but this figure has now been considerably reduced.

At "Mercur" in Utah a cost of 2*s.* 1*d.* per ton has been noted for cyaniding ores.

In India the treatment costs vary from the low figure of 1*s.* 10*d.* to 2*s.* 8*d.* per ton, the cost for cyanide itself being considerably over half the total expense.

CHAPTER IX.

Notes on Extractor Box Work and Sump Solutions.

SOME time since, in the Johannesburg goldfields of South Africa, the attention of the author was directed to certain statements to the effect that the gold-contents of foul or extremely dilute cyanide solutions could not be effectively precipitated in practice by the usual zinc method.

Such statements caused all the greater surprise, because it is well known that this difficulty is not encountered in other well-known districts, as, for example, in New Zealand, where solutions containing less than 0·05 °/₀ of KCy are regularly reduced in practice to below 1 grain of gold per ton, while in Mexico and the United States almost equally satisfactory precipitations are not uncommon.

As a result of these representations, however, the author visited several plants at work in the Transvaal, and noted that while at some few mines the precipitation was really excellent, at others the effluent, after treatment, was much too high in bullion contents.

An investigation into the causes of these unsatisfactory results disclosed the fact that more attention to the close packing of the zinc-shavings in the extractor compartments tended to improve the results, even with the most dilute solutions. The function of the zinc being one of surface, it stands to reason that a compartment well filled with zinc would be more effective than a half-filled one. Moreover, the quality of the zinc was inferior. This should be in long, tough, thin, light threads; light enough to ignite readily, and tough enough not to disintegrate easily into a slime. The coarse shavings too frequently employed appear to give the reverse of the best practice; they are brittle, expose less surface, and take up more room. It is almost impossible to pack a compartment properly with such zinc, which permits the existence of channels, through which the solution may ascend without coming into effective contact with the zinc at all.

The boxes which do the best work have compartments the depth of which is greater than any other dimension.

But even under the best conditions, solutions are encountered which give trouble when run through in the ordinary way. Such solutions are those charged with alumina salts, and those resulting from the treatment of weathered concentrates containing acid iron salts.

Very thorough investigation has been carried out on these solutions; and it is found that by simply giving them longer contact with the zinc they may be treated with complete success. With solutions, the alkalinity of which has been neutralised by acid salts, it is advisable to add a slight amount of soda; but even in the most difficult case an alkalinity of 0·025 %, or $\frac{1}{2}$ lb. per ton, has been found perfectly effective.

Moreover, with extremely dilute cyanide solutions, that is, solutions containing under 0·05 % of free cyanide, perfectly satisfactory precipitation may be carried out by passing the solution through the extractors at the rate of 1 ton of solution per 24 hours for each cubic foot of zinc thread employed. But even when the solutions are very foul, with iron and alumina salts in solution, and containing *no free cyanide*, one is able in practice to reduce the bullion contents to 1 *or* 2 *grains per ton* by passing the solution through the extractors at the rate of about $\frac{1}{2}$ ton of solution per cubic foot of packed zinc thread (weighing about 6$\frac{1}{4}$ lbs.) per diem, the solution being in contact with the zinc for one and one-half to two hours.

This small amount of extractor space, even with the most difficult solutions, as compared with that required for electric precipitation, deserves to be noted.

Liquors derived from the treatment of cupriferous ores and concentrates, and containing 0·2 $^{0}/_{0}$ of copper in solution, were found to be quite amenable to a treatment similar to that which sufficed for the foul liquors from weathered concentrates.

The following are notes of experiments carried out in connection with the investigation above referred to.

The solution examined was one that was returned in a cyanide works as containing 44 grs. of gold per ton, after it had been *twice* passed through the extractors in the usual manner. An investigation showed that it contained a considerable quantity of arsenic and nearly 1 $^{0}/_{0}$ of soluble sulphates in solution, resulting from the treatment of some exceedingly acid weathered concentrates.

The solution was run through the extractors with the following modifications :—

(*a*) The solution was run through a third time at the rate of 2 tons per cub. ft. of zinc per diem.

Assay before treatment, 2 dwts. 12 grs. per ton.

Assay after treatment, 1 dwt. 18 grs. per ton.

(*b*) There were added 4 lbs. of ammonium chloride per ton of solution, and it was run through at 2 tons per cub. ft. per diem.

Assay before treatment, 2 dwts. 12 grs. per ton.

Assay after treatment, 1 dwt. 17 grs. per ton.

The addition of the above amount of ammonium chloride had thus no beneficial effect at the above rate of extraction.

(*c*) There were added per ton of solution 4 lbs. of burnt lime, slaked and made into a cream, and the precipitate was allowed to settle. The rate of flow was not accurately determined, owing to the precipitate having reduced the rate of flow in pipe.

Assay before treatment, 2 dwts. 12 grs.

Assay after treatment, 14 grs.

(*d*) Solution (*à*) was run through again, for a *fourth* time, at 2 tons per cub. ft. per diem.

Assay before treatment, 1 dwt. 18 grs.

Assay after treatment, 5 grs.

(*e*) The solution was run through again at especially slow rate of ½ ton per cub. ft. per diem.

Assay before treatment, 1 dwt. 11 grs.

Assay after treatment, 2 grs.

This shows a great improvement, and indicates that slow running is, of itself, more practically efficacious and cheaper than the addition of chemicals to assist precipitation.

It was found, however, that after the run had been continued for some hours, the assay of the effluent rose to 12 and then to 21 grs. per ton. An examination of the zinc showed that it had been thickly coated with a white incrustation which consisted mainly of ferrocyanide of zinc.

In all the above experiments there was only a trace of free cyanide or of alkali present in solution; and the formation of the incrustation pointed to the necessity of the presence of alkali or free cyanide in solution to prevent any such reaction.

(*f*) Some of the same solution as that used for the previous experiment was now run through the extractor at the rate of about ⅓ ton per cub. ft. of zinc per diem, after sufficient cyanide had been added to the solution to make it show 0·08 °/₀ of free KCy.

Assay before treatment, 0 dwt. 19 grs. gold per ton.

Assay after treatment, 0 dwt. 2 grs. gold per ton.

KCy before treatment, 0·08 per cent

KCy after treatment, 0·023 per cent.

This experiment shows that the addition of free cyanide causes a satisfactory precipitation. There was no incrustation formed on the zinc

The consumption of cyanide, however, amounted to 1 lb. per ton; and to avoid this, if possible, the investigation was continued, and

(*g*) A similar experiment was carried out at the same rate of flow, but adding 4 lbs. of slaked lime per ton of solution instead of cyanide, the precipitated sludge being allowed to settle.

Assay before treatment, 0 dwt. 19 grs.

Assay after treatment, 5 grs.

KCy before treatment, *nil.*

KCy after treatment, *nil.*

The zinc was washed free from incrustation in dilute solution of caustic soda prior to the commencement of this experiment, and sufficient new zinc was added to replace the consumption of previous runs. At the conclusion of the experiment there was a slight incrustation on the zinc in the first compartment only.

(*h*) Experiment as above, but with 4 lbs. of caustic soda per ton added in place of lime. No fresh zinc was added, nor was the zinc washed.

Assay of solution before treatment, 0 dwt. 19 grs.

Assay of solution after treatment, 0 dwt. 1 gr.

After the run the zinc was very black, and there was no incrustation.

(*i*) Experiment as above, but with less soda, and the rate of flow increased to ½ ton per cub. ft. per diem.

Assay of solution before treatment, 0 dwt. 19 grs.

Assay of solution after treatment, 0 dwt. 3 grs.

(*k*) In this experiment the run was continued for some days at ½ ton per cub. ft. per diem, and samples were taken every 12 hours. There was no caustic soda added, there being present only ⅕ lb. per ton of solution; gold was added to make the solution up to 2 dwts. per ton.

The solution assayed before treatment, 2 dwts. 0 gr.

The solution assayed after treatment, 0 dwt. 2 grs. to 0 dwt. 5 grs.

The above experiments show the perfect practicability of precipitation from ordinarily troublesome solutions by keeping the rate of flow at ⅓ to ½ ton per cub. ft. per diem, with the presence of from 0·01 to 0·15 °/o of caustic soda.

A further test was made on the precipitation of auriferous solution carrying quantities of copper, with the following results :—

(*l*) Used solution from the sumps, containing 0·12 °/o of copper in solution ; was run through extractors at the rate of ⅓ ton per cub. ft. zinc space per day. Half a pound of caustic soda was added per ton of solution.

Gold in solution before treatment, 2 dwts. 0 gr.

Gold in solution after treatment, 0 dwt. 0·8 gr.—less than 1 *grain per ton.*

Copper in solution before treatment, 0·12 °/o.

Copper in solution after treatment, 0·097 °/o.

The zinc was coated with metallic copper, but the precipitation at above rate of flow was practically perfect.

Two more prolonged trials were then undertaken, in which a large quantity of solution was run over the zinc used in (*l*), without any addition of fresh zinc, for over a week, to note the effect of the metallic copper coating on the zinc. It was found that the coating became disintegrated in the upper compartment, and that the precipitation continued unimpaired, though, as the quantity of zinc in the extractors was gradually lessened, the effluent solutions rose in value proportionately. On replacing, however, with fresh zinc the zinc consumed, the solutions again became practically gold-free, as in (*l*).

In connection with the above precipitation experiments, two other points call for especial remark :—

1. The amount of alkali added is kept as low as possible, to avoid any great excess beyond the amount required to neutralise the acidity of the ore and for zinc-box reactions.

2. It is well known that the presence of alkali or alkaline carbonates,

always present in used solutions, interferes with the silver nitrate test making the solution appear to be stronger in available cyanide than is really the case. Chemists in charge of works should, therefore, be careful not to be misled by this test, or lowered extractions may result. The effect of caustic potash or soda on double cyanide of zinc and potassium is *not* to regenerate all the cyanide of potassium as shown by silver test, the test, as suggested by C. J. Ellis, being rendered unreliable by the silver throwing out the zinc from the double cyanide, and the zinc so displaced being dissolved by the KHO present, thus prolonging the titration.

$$K_2ZnCy_4 + 2AgNO_3 + 4KHO = 2KAgCy_2 + K_2ZnO_2 + 2KNO_3 + 2H_2O.$$

Zinc-lead Couple.—Reference is made in the chapter on recent cyanide practice to the use in Africa of the zinc-lead couple for the precipitation of the exceedingly dilute "slimes" solutions.

In Mexico the couple is also used to precipitate the copper out of the solutions, and thus to prevent such an accumulation of the metal in the working solutions as would interfere with their solvent powers on the precious metals. The solutions are first run through ordinary extractor boxes for the recovery of their gold and silver, and then through zinc-lead boxes for the precipitation of the copper and of any gold and silver not recovered in the zinc boxes. The effluent solutions carry only 1¼ lb. of copper per ton. The consumption of lead acetate is 10 % of the weight of the zinc: one ton of the former to ten tons of shavings.

The clean-up should not be less frequent than fortnightly, and in Africa it is preferred to perform this operation every ten days.

The suggested use of zinc fume for precipitation has been revived in various quarters. It is obvious that when the precipitant is added to a charge of liquor, one has not only to contend with the intermittent nature of such a process, as compared with the continuous flow of zinc boxes, but that one has to add the precipitant *in excess* to ensure complete precipitation of the gold.

This means impure bullion—contaminated with the excess of zinc dust—and greater expense on account of the larger quantity of zinc required.

On the other hand, in the ordinarily employed zinc boxes there is not only a continuous flow of solution but also a continuous excess of zinc shavings; the latter, however, are separated by a sieve from the precipitate itself, which is thus not contaminated, and the precipitate may be made absolutely high grade by the simple expedient of adding fresh zinc to the bottom compartments only and moving up the used zinc to the upper compartments at each clean-up, leaving only the richest zinc in the first compartment, which may be almost emptied of zinc if desired, as there is always an excess of zinc accommodation in the extractor boxes, and also of zinc to precipitate the solution, apart from that contained in the first boxes.

The physical nature of zinc shavings aids greatly in the separation of the fine gold precipitate, the shavings cohering as a sponge.

Messrs. Sulman and Picard have endeavoured to emulate the continuous nature of the zinc-box process by a most ingenious arrangement of cones, two or three in series. An emulsion of zinc fume is added to these cones every two or three hours, but the difficulty of preventing loss of zinc fume and gold in the overflowing solutions is obvious.

Moreover, the clean-up method proposed—that of distillation—is open to the fatal objection of loss of gold owing to volatilisation or mechanical carrying-over with the volatilised zinc. An ingenious attempt to overcome this difficulty has been made by Mr. Picard by balling the zinc-and-gold sludge with molasses and borax.

A process such as the above may be successful, but it undoubtedly requires far more expert knowledge and careful attention than the ordinary zinc-box method, and it has not therefore been generally adopted even in plants using the bromocyanide process.

Charcoal as a Precipitant.—In Victoria a number of cyanide plants have been equipped with charcoal tubs similar to those in use in chlorination works. This method appears clumsy and troublesome. It involves the use of a very large number of tubs, and the charcoal has to be most carefully prepared and graded.

Moreover, all the charcoal is burnt in a special furnace, and this is a tedious process, as the amount of charcoal is of very considerable bulk. The furnacing operation too affords an opportunity for considerable loss, as does also the handling of so large a bulk of fine light ash.

It requires 144 tubs to precipitate 400 gallons of solution per hour, so that for a plant of 800 tons to 1,000 tons a day capacity, 2,000 regulation tubs and no less than 40 regulation furnaces for burning the charcoal would be required, and the ash remaining at each clean-up would weigh 10 tons, containing only, say, 1 °/₀ of gold. The fusion of so huge a quantity of light ash would not only require large amounts of flux, but would probably be attended with considerable loss of gold. This method appears to be more than twice as expensive to run and to clean-up as the zinc process, and the initial expense of the installation is at least four times as great. It is not, therefore, probable that this method will be adopted in other fields.

Sump Solutions.—One of the problems at present agitating the minds of cyanide operators is the effect on the extraction of gold caused by the continued re-use of the cyanide solutions. Such solutions contain, amongst other matters, salts of zinc, iron, occasionally copper, alkalies and alkaline carbonates, ammonia, and sulpho-cyanides; and it is obvious that, unless these constituents are prevented from accumulating in the solutions, the solvent power on the precious metals of any added cyanide must sooner or later be impaired.

Solutions which had been in use for some months were therefore examined, and the results noted: it was found that the extractions returned were in every case less than those obtained on the same material—ores or tailings—by making up cyanide solutions of the same strength with fresh water. Thus :—

	Extraction with fresh solutions.			Extraction with used solutions.			
	oz.	dwt.	gr.	oz.	dwt.	gr.	
(*a*)	15	13	14	15	0	12	per ton
(*b*)	20	18	3	18	13	16	”
(*b*′)	21	0	17	19	0	5	”
(*c*)	3	17	2	3	13	5	”
(*c*′)	3	18	9	3	13	5	”

It may be suggested that the presence of double cyanide of zinc and potassium (K_2ZnCy_4) may have misled the operator as to the strength in cyanide of the solutions, but it will be shown that this matter has received special attention, and that the results were similar, even when equal quantities of solid cyanide of potassium were added to fresh water and to a solution of double cyanide of zinc and potassium with caustic potash present.

Experiments were carried out with the object of overcoming this loss of power in the used or “sump” solutions, and it was found that the addition of lime improved the extractions from ores and tailings containing practically only quartz and gold, but that such treatment had a deteriorating effect on ores, &c., containing sulphides. Thus :—

	Extraction with sump solution only.			Extraction with sump solution treated with lime.			
	oz.	dwt.	gr.	oz.	dwt.	gr.	
(*a*)	12	15	2	27	0	23	per ton.
(*a*′)	13	1	8	29	9	6	”
(*b*)	18	13	16	25	14	4	”
(*b*′)	19	0	5	25	14	7	”
(*c*)	4	1	1	3	1	0	”
(*c*′)	3	18	9	3	0	2	”

With fresh solution the extractions in this set on *c* and *c*′ were 4 oz. 1 dwt. 1 gr. and 3 oz. 19 dwt. 7 gr. respectively. *a* and *b* were simple quartz ores containing free gold, and *c* was an ore carrying a considerable percentage of sulphides.

A number of experiments were carried out on the above lines, and the above results were confirmed. Lime being thus shown to be ineffective, except with simple quartz ores, treatment of the solutions with sodium sulphide, followed by excess of a soluble lead salt, such as acetate or chloride, was tried with the following results :—

	Extraction with fresh solution.			Extraction with sump solution.			Extraction with "treated" sump solution.		
	oz.	dwt.	gr.	oz.	dwt.	gr.	oz.	dwt.	gr.
(*c*)	3	17	2	3	13	5	3	17	2
(*c'*)	3	18	9	3	13	5	3	19	16

It is thus evident that the continued use of sump solutions is a cause of serious loss in actual work, and that, though a universal law cannot be laid down for the best treatment of such sump solutions, it is strongly desirable that these solutions should, from time to time, be tested, and the results obtained compared with those from freshly made up solutions ; also that the addition of lime, with time for the subsidence of any precipitate formed, is of advantage in the case of very free and coarse gold ores ; and that the treatment with sodium sulphide, care being taken to avoid an excess, followed by the addition of a small amount of lead salt in excess, is efficacious where lime fails. In this latter treatment time must also be given for any precipitated sulphides to separate out and subside.

By such treatment the necessity of running foul sump solutions to waste may be avoided, when water is scarce or the cyanide or gold contents are high.

The Effect of Zinc in Solution.—The usual practice of precipitating the gold by running the aurocyanide solutions through boxes containing zinc, causes the accumulation in the solutions of a considerable quantity of zinc salts. It has been more than once suggested that the double cyanide of zinc and potassium in the presence of caustic potash splits up into a simple cyanide plus oxide of zinc and potassium—

$$K_2ZnCy_4 + 4KHO = 4KCy + K_2ZnO_2 + 2H_2O.$$

The author, however, discovered by means of experiments involving the crystallising out of the products, that the above equation did not hold good, but that, on the contrary, any oxide of zinc and potassium in solution combined with the added cyanide of potassium to form the double cyanide of zinc and potassium. Thus :—

$$K_2ZnO_2 + 4KCy + 2H_2O = K_2ZnCy_4 + 4KHO.$$

This shows that the zinc oxide in solution actually takes up the cyanide added by the chemist in charge to make the solutions of normal strength, though the silver test leaves the operator in ignorance of what has happened, and thus *renders it less effective for the solution of the gold than the cyanide by itself would have been.*

The following experiments illustrate this point :—

	Extraction with a fresh 0·5 % KCy solution.			Extraction with portion of same 0·5 % solution, to which 0·25 % K_2ZnO_2 was added.			
	oz.	dwt.	gr.	oz.	dwt.	gr.	
(*a*)	2	9	0	1	11	9	per ton
(*b*)	16	2	3	13	6	14	„
(*c*)	1	17	21	1	2	20	„
(*d*)	12	18	17	7	4	0	„

This is one of the drawbacks of the zinc process as compared with the electrical deposition process, which leaves the solutions in a condition of much greater energy, though the iron anodes used in the latter process are necessarily detrimental. Insoluble anodes are much needed. The accumulation of zinc in solution is, however, prevented by the action of the sulphides contained in the ore and in the cyanide, and the treatment with sodium sulphide and lead salt as shown above is a good remedy for this drawback.

CHAPTER X.

Losses in Cleaning-up.

PERHAPS in no other part of the process are there so many variations in the methods pursued as in that of the clean-up. At the end of the bimonthly or monthly period the boxes are charged with slimes containing usually gold, silver, lead, zinc, iron, lime, and, in certain instances, copper; and the object in view is the conversion of these slimes into bullion with the least possible loss. To accomplish this, the treatment varies from the direct melting of the coarse zinc and slimes to separation by sieves, roasting, acid treatment with or without filter-pressing, and subsequent fusion.

To determine, if possible, the amount of loss arising from these different methods, and the best general course to be adopted, a system of very carefully conducted experiments has been carried out and the results tabulated and compared.

At first the experiments were on known weights of metallic gold and zinc in the proportions used in practice: the gold was dissolved, precipitated from its aurocyanide solution by the zinc, shaken off the zinc in the form of gold slimes, the zinc remaining was dissolved separately and the residue added to the gold slimes, which were then treated by the various methods, and the amount of bullion recovered noted together with its fineness: the deficit from the amount originally taken represented the loss in treatment. In some cases the coarse zinc was disintegrated and dissolved by the addition of alkali and cyanide, instead of acid, with the results given below.

In the final experiments, however, to gain more accurate comparative results, a large quantity of gold slimes was made, shaken off the zinc, dried, well mixed, and equal quantities taken from the bulk for the various experiments; these were carried out in duplicate, no two duplicate examples however, were weighed consecutively. In the first set the losses were found to vary from 0·5 °/₀ to 6 °/₀, and, as a general rule, the less handling the slimes received the less was the loss. The heaviest losses were due to roasting, and varied with the amount of stirring or handling during the operation; *the least loss from this cause was* 0·23 °/₀ *additional* to that resulting from the rest of the treatment.

In a method of roasting with nitre described in the *Journal of the Society of Chemical Industry* (January, 1897), the *total loss amounted to* 2·57 °/₀ in spite of the greatest care in heating the mixture, and in adding as little nitre as would act in the manner described in the paper; probably, with practice, this loss would be considerably reduced, but it is still too serious a matter to be lightly considered.

Acid treatment yielded results varying with the kind of acid used, nitric acid showing a greater loss than hydrochloric and sulphuric. On the other hand, with 98–99 °/₀ zinc,—that in ordinary use,—nitric acid gave the purest bullion, but with lead-free zinc, dilute sulphuric acid gave both the purest bullion and the lowest acid loss.

A comparison between melting direct and a preliminary treatment with sulphuric acid gave results depending on the purity of the slimes; where these had been passed through a 40-mesh sieve to free them from coarse zinc, melting direct gave a fairly pure bullion and at least as good results as with acid treatment, but with scrap and coarse zinc included the advantage lay with the latter method.

The least total loss in the first set of 19 experiments was 0·75 °/₀, and this was obtained by treatment with sulphuric acid before melting. An experiment with lead-free zinc gave a total loss of only 0·52 °/₀ with direct fusion of the sieved slimes, the coarse zinc being treated with sulphuric acid and the residue added to the slimes. Pure zinc invariably showed a smaller loss than with ordinary zinc, and very careful work on identical lines showed such gain, or lessened loss, to be 0·2 °/₀.

Treatment of the slimes with strong solutions of alkali and cyanide, with or without subsequent acid treatment, gave heavier losses.

There was, however, considerable variation in the results obtained, though the general direction of the losses was well established, and to limit such variation as well as to still further study the effect of the use of lead-free zinc in the cleaning-up operations, the final sets of experiments were carried out in which all the samples were taken from the same bulk of previously prepared and well mixed gold slimes, as mentioned above, and these identical samples were then submitted to variations in treatment. Similar experiments were also made with the slimes from lead-free zinc prepared under identical conditions. The results were as follows, the loss for simple melting with borax being taken as x:—

Melting with borax - - - -	x
Roasting, then melting - - - -	$x + 0{\cdot}23$ %
Sulphuric acid, then melting - - -	$x + 0{\cdot}17$ "
Sulphuric acid, roasting, then melting -	$x + 0{\cdot}40$ "
Nitric acid, then melting - - -	$x + 0{\cdot}29$ "

and x in the case of pure lead-free zinc amounted to 0·43 %; in the case of ordinary zinc it was greater than this, the amount of excess exceeding the 0·2 % given above.

With lead-free zinc, sulphuric acid gave the purest bullion and roasting the most base, the roasted bullion being more base than that melted direct with borax. With ordinary zinc, however, nitric acid gave the purest bullion; sulphuric acid came next; direct melting with borax yielded bullion considerably baser than the others. The loss by acid treatment was greater than with lead-free zinc and the loss by roasting less; it is suggested that the presence in the slimes of lead, which is converted during the roasting to fluid lead oxide, may account for the lessened loss in roasting with the ordinary zinc.

These last results confirm those from former experiments, and show that with ordinary zinc, as well as with lead-free zinc, direct melting with borax gives the least loss, and that the safest method of purifying the bullion is by sulphuric acid treatment; that the use of lead-free zinc with a sieving arrangement renders any special method of purification unnecessary, and diminishes the treatment loss. All the experiments agree in showing that roasting is the cause of more or less heavy loss, and though effective with ordinary zinc in raising the grade of the bullion, is less efficient than acid treatment for this purpose.

It is therefore suggested that in cleaning-up means should be employed to avoid handling, and to limit the number of vessels used, and that roasting should be absolutely avoided. (*See* also p. 58.)

Purity of Gold Bullion.—The fine, amounting to from 3*d.* to 1*s.* 8*d.* per ounce of fine gold, imposed by the London refiners upon base bullion containing lead, has caused much attention to be paid to the question of the advisability of returning bullion so pure as to avoid these heavy charges. An investigation has also been made into this matter, and a suggested remedy is to replace the zinc ordinarily used for precipitation by a special brand of lead-free zinc.

In the early days of the cyanide process the inventors feared that lead-free zinc would not be sufficiently energetic to precipitate effectively the gold present in very dilute solutions, but in recent experiments with some lead-free zinc, which contained a minute amount of iron, it was found that in every instance, whether with strong or with exceedingly dilute solutions, a better extraction was obtained with this zinc than with that ordinarily in use, with the single exception of a fortnight's run, in which the results were equally good.

The importance of this result will be understood when one recognises that the adoption of the lead-free zinc will give a purer bullion, and thus avoid the refiners' charges of, on an average, 9*d.* per ounce; and will lessen the amount of slimes to be treated and the loss during clean-up of 0·2 °/₀, as shown above. Against this is the greater price of the lead-free zinc, but as it takes with low-grade tailings an average amount of 1 lb. of zinc to produce 1 oz. of gold, it is easy to calculate that even if lead-free zinc were 11*d.* per pound dearer than the ordinary zinc, it would still be preferable and cheaper to use the former brand; but, as a matter of fact, the difference in price is very much less than this sum.

In conclusion, it is suggested that increased extractions may almost universally be obtained by the proper treatment of sump solutions, or where water is plentiful, by the making up of fresh solutions from time to time, the old solutions being got rid of as water-washes, after an intermediate period as weak solutions; and that much of the usual "mysterious" discrepancy between theoretical or assay results and the amount of bullion actually returned is due to losses in cleaning-up, and that these losses may be lessened by avoiding roasting and unnecessary handling, and by adopting the precautions detailed above.

The gain to the Witwatersrand Gold Fields from the treatment of sump solutions should be an additional extraction of 5 %, or, say £200,000 per annum, whilst the avoidable loss in cleaning-up is estimated to vary from 0·5 to 2 % of the cyanide output from each particular plant.

CHAPTER XI.

Bromo-Cyanide.

THIS chapter is headed bromo-cyanide as, although the other halogen salts such as chloro-cyanide are also effective, the bromide salt has been most prominently and successfully brought to public notice—mainly through the skill and industry of those responsible for its introduction.

It is not intended to notice here the actual work of the bromo-cyanide process, but in the chapter on the treatment of telluride ores the reader will find a description of perhaps the most successful application of this re-agent. At present this process will be dealt with more from the chemical standpoint.

It may be stated generally, that—

(1) The addition of bromo-cyanide to a solution of cyanide of potassium in the proportion of preferably not more than one of the former to four of the latter, effects a considerable acceleration in the rate of solution of the gold over that of the simple cyanide solution, although it may not affect the final percentage of extraction.

(2) The addition of bromo-cyanide causes a greater consumption or loss of cyanogen than the simple cyanide process, except in those cases where the treatment by the simpler process is ineffective or requires to be considerably prolonged, or where the presence of such minerals as marcasite may render a rapid treatment advisable, and thus prevent the formation of alkaline sulphide, which tends to retard or lessen the extraction by the simple cyanide process.

Bromo-cyanide may be added in crystals—this form has proved inconvenient and expensive to transport owing to the liability to decomposition—or it may be made on the spot by the addition of dilute bromine water to a solution of cyanide, the latter being in excess. Of course by cyanide is meant the usual salt of cyanide of potassium and—or—cyanide of sodium.

It is not advisable to add all the bromo-cyanide required at once, but in portions, say at every two hours, as in the presence of cyanide solution, decomposition is rapid, being completed in from three to six hours.

It is stated that in practice the crystals give lower cyanogen consumptions than the home-made solution, but this statement does not appear to have been verified.

The following is the result of some investigations carried out on this subject. The results of the tests on gold ores will be found in the chapter previously referred to.

Dilute solutions of bromo-cyanide were used for these experiments containing 1 % of bromo-cyanide or under; the solutions were made from bromo-cyanide crystals obtained by distillation, as well as from the addition of bromine water to the cyanide of potassium solution.

To test for bromo-cyanide one adds potassium iodide and hydrochloric acid in excess; the liberated iodine is then titrated with $\frac{N}{10}$ sodium hyposulphite in the ordinary manner, thus—

$$BrCy + 2KI + 2HCl = BrCy + 2HI + 2KCl.$$

$$BrCy + 2HI = HBr + HCy + I_2.$$

In forming bromo-cyanide by the addition of bromine water to potassium cyanide solutions, it must be noted that the reaction $KCy + 2Br = BrCy + KBr$ takes place slowly with dilute solutions and is not completed for some time. With stronger solutions however—those over 1 %—the reaction is immediate and even violent; heat is produced and decomposition products are formed.

DECOMPOSITIONS,

(I) With Cyanide (by "Cyanide" here is meant a dilute solution of cyanide of potassium and—or—sodium).

A solution of bromo-cyanide is much more stable than one would imagine, and does not appear to be decomposed on standing or even dissociated by dialysis, but after addition to a solution of cyanide, decomposition is set up with the formation of Cy_2, thus—

$$KCy + BrCy = KBr + Cy_2.$$

Experiments showed that with a solution containing 0·2 % KCy and 0·1 % BrCy one-third of the bromo-cyanide had been decomposed in 15 minutes, and over one-half in 30 minutes. In an hour and a quarter little over one-third remained of the amount originally present, and this had been reduced to a quarter half-an-hour later. Only one-fifth remained after 2 hours, one-tenth after 3 hours, and one-twentieth (or 0·005 %) after 4 hours. In 5 hours all the bromo-cyanide had been decomposed save 0·001 %, and in 5½ hours none remained.

A stronger solution containing 0·4 % cyanide and 0·2 % bromo-cyanide showed a decomposition of 0·5 % in 15 minutes, 1 % in an hour, and 1·5 % in 2½ hours. After 3½ hours only 0·02 % remained, and this was reduced to 0·01 % in 4½ hours. But the solution was not entirely free of bromo-cyanide until over 6 hours had elapsed.

A solution containing 0·8 % cyanide and 0·2 % bromo-cyanide—that is the same strength in bromo-cyanide as before, but with four times the amount of potassium-cyanide present instead of twice only—showed that in 5 minutes 0·5 % of the bromo-cyanide had been destroyed; 1 % (half the total quantity) in less than half an hour. In an hour less than 0·05 % remained; in 2 hours only 0·01 %; and by 3½ hours only the barest traces remained.

It thus appears that with a greater proportion of cyanide present the destruction of the bromo-cyanide is more rapid. In the first two cases half the bromo-cyanide had been destroyed within an hour, and in the third case the destruction was 50 % more rapid.

It was also noticed that bromo-cyanide decomposed much more rapidly if alkali hydrate were present *in addition to* the KCy. Thus, repeating the last test, that is, using a solution containing 0·8 % cyanide, and 0·2 % bromo-cyanide and 0·24 % potassium hydrate (caustic potash), one-half of the bromo-cyanide was destroyed *in less than one minute*, and after 5 minutes only 0·03 °/$_0$ remained. In an hour the whole of the 0·2 °/$_0$ had been dissipated.

This shows how necessary it is to avoid the presence of alkali in practical work, even when it is desired to coagulate the slimes. In this case bromo-cyanide treatment takes place first and the addition of lime afterwards.

Carbonates were not so rapid in their action as hydrates, but half of the bromo-cyanide was destroyed in 5 minutes and all save 0·05 °/$_0$ in 20 minutes, when 0·27 °/$_0$ of carbonate of potash was added to a similar solution as that used in the last experiment. In less than three hours not a trace remained.

This demonstrates that the cyanide used in the bromo-cyanide process should be absolutely as pure as possible, as should also the water used for making up the solutions. Thus, with the same strength of solutions, taking four different samples of commercial cyanide, in the best case the bromo-cyanide was not wholly consumed until 6¾ hours had elapsed, but in the worst total destruction had taken place in less than half this time. This also points to the advisability of paying special attention to the condition of sump solutions where these are re-used in bromo-cyanide treatment.

TABLE No. 1.—DECOMPOSITION OF BrCy WITH KCy.

Solutions tested after	Sol. No. 1. 0·2 % KCy. 0·09 % BrCy.	Sol. No. 2. 0·4 % KCy. 0·2 % BrCy.	Sol. No. 3. 0·8 % KCy. 0·2 % BrCy.	Sol. No. 3a. 0·8 % KCy. 0·2 % BrCy. 0·24 % KHO.	Sol. No. 3b. 0·8 % KCy. 0·2 % BrCy. 0·27 % K_2Co_3.
	BrCy °/$_0$.	BrCy °/$_0$.	BrCy °/$_0$.	BrCy °/$_0$.	BrCy °/$_0$.
1 min. ...	0·0920	0·1998	0·1725	0·0795	0·156
5 mins. ...	0·0848	0·1748	0·1484	0·0344	0·103
15 „ ...	0·0662	0·1457	0·1325	0·0212	0·054
30 „ ...	0·0477	0·1298	0·0822	0·0053	0·040
1 hour ...	0·0397	0·0987	0·0447	*Traces*	0·015
2 hours ...	0·0212	0·0662	0·0101	*Nil*	0·004
3 „ ...	0·0101	0·0371	0·0026	...	*Nil*
4 „ ...	0·0053	0·0137	*Nil*		
5 „ ...	0·0013	0·0074			
6 „ ...	*Nil*	0·0012			

The decomposition rates of cyanide were also noted; bromo-cyanide being added made from crystals in the proportions shown below.

1,000 grains of solution containing 0·5 °/o cyanide and 0·25 °/o bromo-cyanide after standing for 16½ hours in a flask loosely covered with paper, tested cyanide 0·31 °/o, bromo-cyanide *none*. The same solution at the end of a week tested 0·19 °/o KCy.

A solution containing 0·4 °/o cyanide and 0·2 °/o bromo-cyanide similarly treated, tested cyanide 0·25 °/o, bromo-cyanide *none*. At the end of a week the same solution tested cyanide 0·13 °/o.

Similarly 0·3 °/o cyanide and 0·15 °/o bromo-cyanide tested cyanide 0·18 °/o, bromo-cyanide *none*. At the end of a week, cyanide 0·16 °/o.

And finally, a solution containing 0·1 °/o cyanide and 0·05 °/o bromo-cyanide tested in 16½ hours, cyanide 0·05 °/o, bromo-cyanide *none*. After a week, cyanide 0·04 °/o.

The stronger solutions turned brownish-yellow on standing.

From the above it will be noticed how comparatively little is the consumption of the dilute solutions of cyanide after the decomposition of bromo-cyanide has taken place. This was confirmed by other experiments extending over some days. Comparative tests carried out at the same time in open and closed vessels showed that the loss was due to actual decomposition, and not to volatilisation from solutions.

TABLE NO. 2.—DECOMPOSITION OF KCy.

Tested after standing.	Sol. No. 1. 0·5 % KCy. 0·25 % BrCy.	Sol. No. 2. 0·4 % KCy. 0·2 % BrCy.	Sol. No. 3. 0·3 % KCy. 0·15 % BrCy.	Sol. No. 4. 0·1 % KCy. 0·05 % BrCy.
	KCy °/o.	KCy °/o.	KCy °/o.	KCy °/o.
16½ hours	0·31	0·25	0·18	0·05
1 week	0·19	0·13	0·16	0·04

To confirm the equation given above that

$$KCy + BrCy = KBr + Cy_2$$

1,000 grains of 0·4 °/o bromo-cyanide solution were mixed with 1,000 grains of 0·24 °/o KCy solution, and left standing over-night. The results, when tested, showed that complete reaction had taken place, as no bromo-cyanide or potassium-cyanide could be discovered in the solution. On adding excess of KHO, however, the presence of potassium-cyanide was discovered to the extent of 0·12 °/$_0$, thus showing that Cy_2 was present in the solution.

(NOTE.)—The silver nitrate test is not reliable in the presence of cyanogen, as it tends to give too low results.

To prove the reaction, the amount of bromide of potash formed was determined quantitatively as follows. The cyanogen was driven off by distillation with bicarbonate of soda; the residue was then carefully neutralised with nitric acid and titrated with silver nitrate, using chromate as an indicator; a blank test was also made at the same time of the chloride existing as impurity in the amount of bicarbonate used.

The bromide of potash found was 0·88 grain as against 0·9 grain calculated. Repeat experiments were made, varying the quantities used, and 2·27 grains KBr were determined as against 2·25 grains calculated. The determination of the amount of bromide of potash formed thus proves the above reaction to be correct.

(II) With Alkalies.

The addition of alkali hydrates or carbonates to dilute solutions of bromo-cyanide do not appear to result in the formation of cyanide, although Watt's Dictionary gives equations showing that this reaction should take place with the formation of bromate and cyanate of potassium and water, or carbonic dioxide, respectively. Even after standing, no trace of cyanide was discovered, and this was so although the solutions were heated. A large number of experiments were carried out on this point.

(III) With Metals.

To test the solution on metals of bromo-cyanide *alone*, a series of squares of specially prepared metallic foil were used, each having an area of two inches. The following were the metals investigated :—gold, silver, iron, lead, copper, zinc, and aluminium.

The results were as follows:—

Zinc showed the greatest amount dissolved—0·13 grain out of a total of 27·454 grains, with a solution rate of 0·00059 grain per square inch per hour.

Lead came next, with a loss of 0·08 grain and a solution rate of 0·00037 grain.

Gold followed, with a total loss of 0·07 grain and a solution rate of 0·00031 grain.

Then *silver*, with a total loss of 0·05 grain and a solution rate of 0·00027 grain.

Aluminium showed a total loss of 0·04 grain and a solution rate of 0·00018 grain.

Iron a total loss of 0·03 grain and a solution rate of 0·00015 grain.

And finally *copper*, with a total loss of 0·027 grain and a solution rate of 0·00013 grain.

On a basis of actual weights taken, instead of surfaces exposed, aluminium was the most acted on, and silver and gold in both cases were more acted on than zinc.

After treatment it was found that the surfaces of the gold, silver, aluminium, and zinc discs were bright, but the lead, copper, and iron had become dull—the iron especially so.

Action on Gold.—Bromo-cyanide *per se*—that is, apart from its reactions in the presence of cyanide—can scarcely be claimed as a solvent of the precious metals. Indeed, as previously pointed out, it is remarkably stable until added to cyanide solution, but if in addition gold is also present, a reaction will take place as follows:—

$$3KCy + BrCy + Au_2 = 2KCy + Cy_2 + Au_2 + KBr = 2KAuCy_2 + KBr,$$

and it is undoubtedly to this reaction that the fame and extensive use of bromo-cyanide is due, whether as an accelerator or as an aid to actual solution.

From the above it will be seen that although there may undoubtedly be a large waste of bromo-cyanide in forming KBr, yet the power of attack of the Cy_2 radical *at the moment of formation* on the gold present, is the cause of a marked acceleration of the extraction rate over that of ordinary solutions.

It has been previously stated that bromo-cyanide, although stable by itself, decomposes when introduced into a solution of cyanide of potassium with the formation of bromide of potassium, and that if gold is present, auro-cyanide of potassium is formed. It has been claimed by Bettel and others that air is necessary for this reaction just as in the simple cyanide process. The following experiments, in which the air was carefully expelled and excluded by the methods stated, prove that this is not so.

The bromo-cyanide in crystals was at first sealed in a thin glass tube from which the air had been previously expelled by heating the BrCy and thus filling the tube with bromo-cyanide vapour at the moment of sealing, and the amount of BrCy remaining in the tube determined. This tube was introduced into the neck of a Dumas bulb containing cyanide solution and filled with cyanide solution vapour by boiling. The tube was broken and specially prepared gold foil cylinders were introduced into the neck and the bulb sealed.

Additional experiments were carried out in parting bulbs from which the air had been expelled by similar methods and in which the BrCy and gold were introduced with the same precautions.

To compare the results with those yielded by an oxidising agent, similar experiments were at the same time carried out with the addition of ferri-cyanide of potassium and potassium hydrate in place of bromo-cyanide.

In the table herewith KFC = ferri-cyanide of potassium, KHO = caustic potash, BrCy = bromo-cyanide.

TABLE No. 3.—EXPERIMENT A.

Solution taken 750 grains. Strength of cyanide 1·39 %. Time taken 23 hours. Each gold cylinder weighed about 3·9 grains (weight determined to third place in decimals of a grain), with a surface of 0·5416 inch.

No.	Air.	Reagent.	Gold Dissolved per Square Inch of Surface.
		Per cent.	Grains.
1	No	BrCy 0·9	0·526
2	No	KFC 5·58 KHO 0·95	0·986
3	Yes	...	0·283

EXPERIMENT B.

Strength of solution 0·54 % KCy. Time of each experiment 44 hours.

No.	Air.	Reagent.	Gold Dissolved per Square Inch of Surface.
		Per cent.	Grains.
1	No	BrCy 0·364	1·214
2	No	KFC 1·38 KHO 0·37	1·109
3	Yes	...	0·314
4	Yes	BrCy 0·364	1·117
5	Yes	KFC 1·38 KHO 0·37	4·187

These results are most interesting. They not only show that air is unnecessary for the solution of gold in the presence of bromo-cyanide, but that the addition of air causes no improvement in extraction where bromo-cyanide is concerned; although with ferri-cyanide the improvement by such addition is most marked.

The experiments given in the table are only a few of the total; all of which confirm the results stated.

It was also shown that in spite of the decomposition of the bromo-cyanide, which might have been expected from the treatment with heated solution, this reagent appeared to be destroyed less rapidly in the absence of air, as much as 0·011 % remaining in the solution after 44 hours' treatment. All the bromo-cyanide solutions showed signs of decomposition products (paracyanogen), and in the stronger solutions a large amount of black insoluble matter separated out.

(IV) With Minerals.

Powdered haematite and marcasite had practically no effect in decomposing a solution of bromo-cyanide. Zinc blende had but little and calamine had rather less. Copper pyrites, however, decomposed it energetically, although malachite was almost unacted on.

Sulman claims that bromo-cyanide will decompose a crystal of calaverite, and on the other hand it has been laid down in the same way that sylvanite is not decomposed by this reagent.

(V) Decompositions with Alkaline Sulphide.

Alkaline sulphides are well known retarders of the solubility of gold in cyanide solutions, and the use of lead acetate has been propounded to get over this difficulty. Alkaline sulphides do not react with bromo-cyanide to form alkaline cyanide, as one would infer from Watt's Dictionary, but the reaction yields sulpho-cyanide of the alkali and a bromide of the alkali. It was suggested at first that bromo-cyanide was effective as an accelerator owing to its power to react with alkaline sulphide in solution, and this suggestion was to some extent confirmed by the results of a great number of experiments which showed that the addition of lead carbonate very frequently gave much higher results than the simple cyanide treatment, where bromo-cyanide also had this effect.

But the energetic action of bromo-cyanide was so marked in these cases, as well as in those of metallic gold where there could be no question of the presence of alkaline sulphides, that other explanations were necessary, and this explanation is undoubtedly the remarkable power of the molecule Cy_2 when in contact with gold or certain gold-bearing minerals at the period of its dissociation or nascency.

Tests made by adding solution of cyanogen to cyanide solutions showed a certain acceleration over the simple cyanide process, but with not nearly so marked an effect as when BrCy is present. And this tends to confirm the suggestion that the remarkable efficiency of this reagent is due to the production of the molecule of cyanogen in contact with the material to be treated.

CHAPTER XII.

Chemical.

Oxygen Necessary.—It is now universally recognised that air is necessary for the solution of gold by cyanide—except where bromo-cyanide or other Cy_2 forming reagent is present, in accordance with Elsner's equation:—

$$4KCy + 2Au + H_2O + O = 2KAuCy_2 + 2KHO.$$

Dr. McLaurin has made most able researches in this connection, and has also discovered a connection between the effective strength of cyanide solutions and the amount of oxygen contained in them. Dr. McLaurin's investigations into this subject are classic, and should be perused by all interested in the subject. They are to be found in the journals of the Chemical Society, 1893 and 1895. He puts the maximum effective solution as that containing 0·25 % KCy, and states that a solution of 0·005 % has but slight power for dissolving gold, but in a solution of 0·01 % the power is very much greater. It will thus be seen that the presence of reducing agents, or oxygen absorbers, in solutions also tends to interfere with the extraction, and that these reagents should be either oxidised or eliminated from the solutions.

Testing Solutions.—It does not appear to have been generally recognised in proposals to employ hyposulphite of soda for extracting the silver, with cyanide treatment for the extraction of gold, that the presence of hyposulphite lessens the solution effect of cyanide and also interferes with the sivler test, the interference in the test being only slight with dilute solutions, but of quite considerable amount when the amount of hyposulphite present is large. Roughly, the increase is rather more than the percentage amount of hypo present; thus, a 1 % solution of cyanide will test over 1·01 % if 1 % of hypo is present; about 1·06 % if 5 % of hypo is present; and 1·12 % if 10 % of hypo is present.

In testing solutions it will also be noticed that ammonia interferes with the reaction by dissolving the milky precipitate of silver cyanide formed. The addition of potassium iodide obviates this error. A similar error is also caused by the presence of zinc and alkali in solutions (*see* Chapter IX).

Regeneration of Waste Solutions.—This has led many to imagine, and even to patent schemes for the regeneration of waste solutions, when further investigation would have shown them that the so-called regenerated cyanide solution was effective in theory more than in practice. Thus, one of the first methods proposed was to regenerate the solutions by precipitating the zinc by sulphide of soda. This appeared to yield quite good results until the regenerated solutions were tested against freshly made up solutions of the same strength. Further investigations showed that roughly only one-third of the theoretical amount of zinc was precipitated by the sodium sulphide present; and even if large additions of sodium suphide were made, and the excess precipitated by the addition of lead salt, the results were not such as to cause this method to be adopted in practice.

As a matter of fact, this is probably the very process which so fortunately takes place naturally, and much more slowly than by the sudden addition of the precipitant. The alkaline solutions acting on sulphides in the ore cause the formation of soluble sulphide, which precipitates the zinc sulphide out on to the ore, and thus prevents the accumulation of this metal in the works' solutions.

A very great amount of research work has been carried out on regenerative schemes by distillation and otherwise; but, in view of the small cyanide consumptions now prevalent in treatment, and the fact that *cyanogen does not appear to accumulate in the sump solutions*, even after years of re-use, all such schemes have to be regarded as unpayable, even apart from their inconvenience in practice. Thus, Argall shows that solutions which had been in use for 2½ years with the cyanide strength made up to 0·5 % each time before treatment, yet only contained 0·53 % of cyanide, 0·023 % hydrocyanic acid, 0·039 % sulpho-cyanide of potassium, and 0·058 % ferro-cyanide of potassium; and this experience of his of the decomposition of the cyanogen in solutions has been the experience of all other workers with the cyanide process.

Assaying Solutions.—In assaying solutions the author has previously referred in Chapter I, to the widely divergent results which are obtained from the neglect to add litharge at the commencement of or early in the evaporation.

The method suggested (by Crosse?) of boiling the solution with a few drops of nitric acid, adding silver nitrate, and boiling until all the precipitate is flocculent, filtering and cupelling the precipitate, also gives correct results.

Good results may be obtained by absorbing the amount of test solution in paper pulp or in a crucible full of flux. Absorption in kieselguhr or ground pumice-stone gave less good results. Acidification by hydrochloric acid; evaporation direct on lead foil and cupelling have both a tendency to give far too low results.

Estimation of Cyanogen in Waste Solutions.—The following simple method of estimating cyanogen in "waste" (or used) solutions, and in Prussian blue, has been found by the author to give reliable results, provided the estimation is carefully done, with attention to the points stated, otherwise the results may be widely divergent.

A measured quantity of the solution, or a weighed amount of the Prussian blue, is boiled with mercuric oxide in excess, and, of course, water if necessary, until all the double cyanides have been decomposed. A 10 % solution of sulphide of soda is then run in from a burette in excess to the quite hot solution, with constant shaking. When no more precipitate is formed add freshly precipitated carbonate of lead in excess, filter, *cool*, and estimate cyanide present by standard nitrate of silver solution. This method determines the cyanide present, whether existing as single or double cyanides.

The best method of adding the sodium sulphide was found to be slowly, but continuously, drop by drop, with constant shaking. This resulted in the sodium cyanide and mercury sulphide being formed at once with practically no sulpho-cyanide. The metals present in the "waste" liquors may also be determined in the mercuric oxide precipitate.

Effect on Cyanide of Various Minerals.—It is well known that certain minerals decompose cyanide most rapidly, particularly the copper carbonates, sulphates, and sulphides. A complete determination of the effect on cyanide of various minerals met in association with gold and silver does not appear to have been carried out, but the following shows the effect of some of the minerals most commonly encountered.

Antimony.—Stibnite gives only a slight reaction.

Cobalt.—The earthy forms, such as asbolane, react strongly and cause heavy consumption of cyanide.

Copper.—The sulphides, such as copper glance, erubescite, and copper pyrites, give heavy consumption of cyanide, but the sulphantimonides, such as bournonite and tetrahedrite, have comparatively slight effect. Of course, the carbonates and sulphates react strongly.

Iron.—Iron minerals generally, including even pyrites, mispickel, and marcasite, give only a slight consumption, but the effect to a considerable extent depends on the alkalinity of the solution.

Lead.—Galena reacts very slowly.

Manganese.—The carbonates and oxides have but slight effect, but the impure variety "wad" causes very serious loss.

Mercury.—Cinnabar reacts very slowly.

Silver.—Silver chloride is readily dissolved, but the sulphides and sulphantimonides are attacked with difficulty, and are only slightly affected.

Zinc.—Blende and the silicates of zinc have only a slight reaction, but spartalite and calamine react strongly.

Poisoning.—The author has fortunately had little experience of the poisonous effects of cyanide solutions, an ounce or two of 20 % solution, which unfortunately ran up into his mouth from a pipette, being ejected so rapidly as to leave the sensation only of a strong alkali burning the tongue and palate.

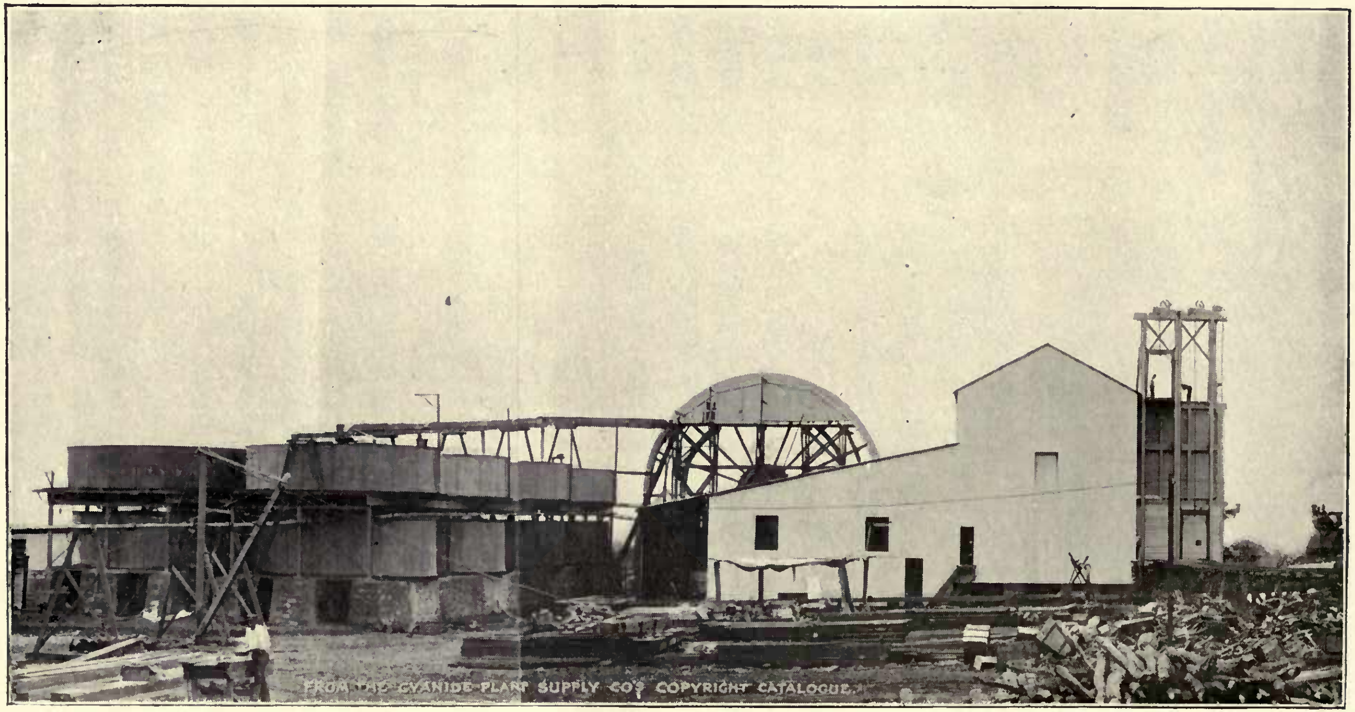
In the early days of the process, when agitating solutions in the height of an African summer, one or two Kaffirs were overcome by fumes, but speedily recovered after a bottle of ·880 ammonia had been held to their nostrils. The indignant look of the Kaffirs argued a preference for hydrocyanic acid rather than for ammonia treatment, but there were no more collapses.

Unfortunately, two valuable lives were lost in the early days by hitherto unsuspected dangers. Dr. Hy. Liepmann died in America from the effect of the arseniuretted hydrogen generated by the acid treatment of zinc precipitate;—this operation is best effected in the open, or with a connection from the vat to a flue;—and Mr. Taggart was overcome in New Zealand by the fumes from an almost empty sump. Sumps should thus either be in the open, or, if in a building, the latter should be well ventilated.

Various remedies for poisoning have from time to time been suggested, and the African Chemical and Metallurgical Society devoted some time to the discussion of this problem.

The best suggestion appears to be that of Dr. Loevy—the hypodermic injection of a 2 °/₀ solution of hydrogen peroxide through a Pravartz syringe; and cyanide operators are recommended to acquire the use of this instrument.

The pioneer works always kept solutions of ferrous sulphate and of an alkaline carbonate ready to hand, so that the mixing might produce freshly precipitated ferrous hydrate which could be swallowed at the moment of emergency. A dose of nitrate of cobalt solution has also been recommended in various quarters, and F. S. Tuttle recommends the swallowing of two drops of ammonia on a lump of loaf sugar. He also recommends for external poisoning a warm bath containing washing soda and common salt.



FROM THE CYANIDE PLANT SUPPLY CO'S COPYRIGHT CATALOGUE.

A RECENT DOUBLE-TREATMENT PLANT ("SONS OF GWALIA," W.A.).

CHAPTER XIII.

Recent Cyanide Practice.

WITH the great and ever increasing success of the cyanide process in Africa, local prejudices in other places have disappeared, and now it is difficult to find any large gold mining centre in which the process does not play an important part.

Naturally, in so many different territories the practice has become modified to suit local conditions, and it is with these modifications that this paper will principally deal.

American practice has been so thoroughly described by Scheidel, Janin, Leggett, and Bosqui, and by articles in the mining papers, that it is not the intention of the writer to enter into minute details, as these are already at hand and available. It may be noted with interest, however, that America, though apparently late in adopting the cyanide process, has already taken it in hand in characteristic fashion, and is now operating the biggest plants in the world running on dry-crushed ores.

Until the Transvaal War closed down the Johannesburg mines, the major portion of the cyanide output of the world was produced from tailings, but at the present moment dry crushing and direct treatment, as carried on in America, West Australia, New Zealand, and Eastern Asia, appears to take first place. This is a most interesting point, and of course the position is only possible through the sudden cessation of the huge African cyanide output of 900,000 ounces of fine gold in conjunction with the increased returns from the large installations in North America and Western Australia.

AMERICAN PRACTICE.

IF one considers the large installations, illustrated descriptions of which appear almost weekly in the American technical journals, the following features stand out prominently as specially marking American practice :—

(*a*) The ore is dry crushed.

(*b*) The ore is roasted or calcined before crushing.

America was practically the first gold mining country to boldly adopt the roasting of pyritic or telluride ores.

Elsewhere such ores were either not treated or were given long contact, as, for example, the Johannesburg Spitz concentrates, containing about 7 °/₀ of pyrites, which received a treatment of as long as from 24 to 30 days. The American system of roasting lessens the time, increases the extraction, and at the same time diminishes the cyanide consumption, which on raw telluride ores is apt to be very heavy owing to the action of the alkaline solutions in combining with the sulphur of the pyrites. The soluble sulphides thus formed in solution exercise a baneful effect on the extraction of the gold, and it has been found advisable to remove them by the addition of a little lead salt.

But the American system would be only economically possible by the aid of efficient mechanical roasting furnaces, and it is a very remarkable tribute to the skill of the American inventor that until quite recently practically every good mechanical furnace at work was based on American designs and practice.

From a consideration of the treatment at the mammoth installations of the Metallic Extraction Company at Cyanide (Argall Roasters), the Economic Gold Extraction Company (Argall Roasters), the De la Mar Mines at Mercur (Jackling Furnaces) we find that at the former practically a dead roast has been obtained at a cost of 28·7 cents per ton of ore roasted. This is a most wonderful achievement, and foreign countries find some difficulty in crediting such cheap and effective work.

An interesting feature of local practice is the use in America of vats with wooden bottoms and steel sides, and it is also the custom in certain districts to use steel plates of much thinner gauge than that employed elsewhere for the purpose. To caulk such vats is impossible, and they are therefore kept tight by a liberal use of paraffin paint, a dressing of which takes up any " weeping " as soon as this starts. Generally American practice also takes first place in the arrangements for the conveying of ore residues, ashes, &c., and for the automatic and rapid cooling of roasted ore prior to cyanide treatment.

Fine crushing is performed almost invariably by rolls. The ball mills which play so important a part in Australia and elsewhere do not appear to have found favour amongst American millmen. On the other hand, with roasted ore such fine crushing is seldom necessary as is required in the raw treatment of pyritic or telluride material.

The following outline of the " Argall " plant at Cripple Creek is typical of American practice :—

Large Gate's crusher.
Small Gate's crusher to 1-inch cube.
Crushing rolls to 4-mesh.
Conveyor to storage bins.
Tubular dryers (the ore heated to 300° Fahr.).
Elevators.
Coarse screens.
36-in. rolls crushing to 8-mesh.
Screens.
15-in. × 26-in. rolls crushing to 15-mesh.
Two finishing rolls crushing to 30-mesh.
Crushed ore bin.
Argall roasters.
Conveyors.
Mechanical coolers.
Conveyor of cooled ore to plant.

AFRICAN PRACTICE.

African Practice is almost the precise reverse of the American. Roasting for cyanide treatment is practically unknown. Dry crushing exists at only out-of-the-way mines, whose ores are too poor to pay by the usual wet crushing process. The development of the process in Africa has been almost entirely mechanical, and mainly in the direction of larger equipment. But problems in their way as difficult of solution as that of the "Cripple Creek" tellurides—*e.g.* the direct treatment problem—have been encountered and successfully surmounted.

The main features of African practice are these:—

(*a*) Wet crushing with double treatment of previously amalgamated tailings.

(*b*) The treatment of rough concentrates by percolation.

(*c*) The treatment of slimes by decantation.

(*d*) The precipitation of extremely dilute solutions carrying low gold contents.

In the early days of the process, when huge stacks or dams of tailings were available, a plant was a simple matter consisting of percolators and reservoirs, extractor house and sumps, but as soon as it was found advisable to bring cyaniding into line with the rest of the ore treatment scheme, and thus to avoid the settlement of tailings in dams and the subsequent necessary handling, one was confronted at the outset with the tendency of the tailings to pack in the vats or to distribute themselves in layers with intermediate impermeable beds of slime.

To avoid this difficulty collecting vats were adopted, and distributors similar to those used in Cornish buddles were so arranged that the outlets being tangential the force generated by the delivery of the pulp was sufficient to revolve the apparatus. The water and slimes were allowed to flow off by a circular channel surrounding the rims of the vats. It was necessary to keep the vats full of water, and any stoppage in the delivery of pulp or in the flow of water—resulting from a clean-up, or temporary stoppage of the battery, or other cause—resulted in the settlement of an impermeable layer of slime on the tailings already deposited in the vat.

The chief group of mines—that of which Mr. Hennen Jennings was consulting engineer—adopted a different method, that of providing the vat with variable level discharge, as in buddles, and filling through hose pipes shifted over the vat as desired. This method appears to have given results in practice superior to those of the automatic distributors, and it has been generally adopted at the large mines in spite of the manual labour connected with the shifting of the hose. Automatic methods of adjustment of the feed do not appear to have been so successful as the "intelligent Kaffir."

Special rubber hose is used for this purpose, as unless internally armoured, it kinks, and on the other hand the internal spiral wire causes the sand to deposit and the hose to be choked; the hose is thus specially manufactured for the purpose with absolutely smooth internal bore, below which the spiral wire is embedded.

At the "JUMPERS" and "NEW KLEINFONTEIN" attempts were made to collect and leach in the same tanks, and at the former plant automatic revolving distributors were used in conjunction with variable level discharges, but it could not be expected that the lixiviation of packed vats would yield such high extractions as could be obtained from the more loosely filled charges, to say nothing of the opportunity for the aëration of solutions afforded by the transferring system, and this system of collecting and leaching in the same vat has not therefore gained favour.

To save expense in transferring the charge from the collecting to the leaching vats it is the practice to superimpose the former on the latter, the discharge being effected through doors in the bottom of the upper tank; these permit of the contents falling into the lower at a cost of say 1*d.* per ton for handling.

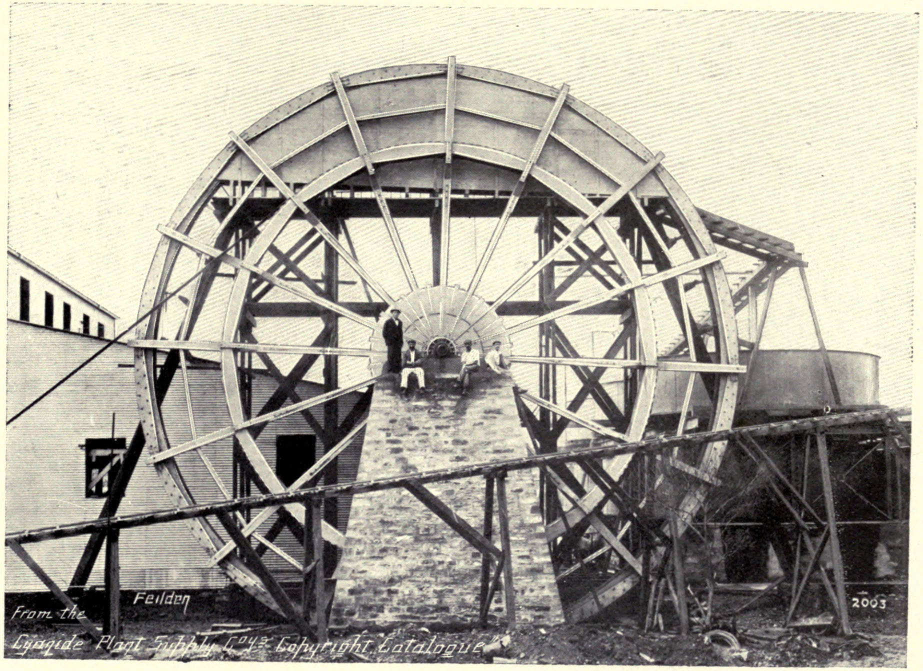
The original scheme of dumping tailings over the side of the collecting vat into the adjacent percolators or of discharging through doors into trucks, which in turn deliver their contents into the percolators, has been superseded by the above arrangement. This, together with the use of spitzlutten for eliminating the concentrates, has necessitated the delivery of the pulp at a greatly increased height, and this is effected most cheaply and conveniently by tailings wheels, which are now made of huge dimensions, one wheel delivering as much as 10,000 tons of pulp—ten of water to one of tailings—per diem. Good examples of this type of wheel may be seen at the deep-level mines at Johannesburg, some of these having a diameter of as much as 60 feet. These wheels are usually made with steel or iron centres, with wooden framework and buckets, the largest wheels being double—that is, with two sets of buckets, one set on each side of the driving pulley, which is really the centre of the wheel.

Plunger pumps have also been used to elevate the tailings, with a water washing arrangement for the stuffing boxes, but in practice these have not been found so desirable as the tailings wheels.

The slimes are elevated by centrifugal pumps made of special wear-resisting metal, with spindles and bearings of greatly increased size over those used for pumping water only; the difference in the specific gravity alone of the slimes pulp and of ordinary water accounting for a portion of the greatly increased power necessary to run these pumps.

Spitzlutten are generally used for eliminating rough heavy concentrates before the pulp passes into the tailings vats, and spitzkasten remove the slimes from the sands before the latter are delivered into the collecting vats.

One of the details of African practice is the practice of placing the steel tanks on 9-in. × 3-in. spaced wooden deals instead of directly on the metal joists. The deals allow the rivets to sink into the body of the wood, and altogether are much more effective as an intermediate support, and prevent the sagging of the bottom of the tank, which is so usual where steel joists only are employed.



“A TAILINGS WHEEL (SONS OF GWALIA”).

Owing to the particular development of the industry in South Africa into plants of huge size, much attention has been given to the supporting of the upper collecting vats. The usual method was to rest these on columns, some of which were placed inside the lower vats. This not only strained the bottom of the lower vat, but was found generally inconvenient in filtering and discharging the contents of the percolators. Spherical bottom tanks were accordingly designed which should be supported from the periphery only, all the stresses of the tank being in tension. This plan was found, however, to be more expensive than the usual type, and of doubtful advantage. Very heavy girders have also been designed to carry the strain of the upper tanks to columns placed outside the lower tanks, but these composite girders are naturally heavy and expensive. The latest plants are arranged for the columns to pass right through the bottom of the lower tanks, an enveloping pipe being fixed to each hole in the bottom of the tank and passing up to above water line. Thus leakage is prevented at the expense of space, as the columns themselves do not come into contact with the lower tank anywhere, the enveloping pipes being bolted to the bottom plates with rubber joints pretty much as a discharge door combing.

The tanks are discharged through doors of 16-in. to 20-in. diameter. Numerous types of doors have been placed on the market, and at one time doors on rollers running on tracks were extensively employed, but preference is now given to lighter doors of pressed steel, which are convenient in practice and easily manipulated by one man.

The tanks usually discharge direct into trucks, which are run by a system of mechanical haulage on to the tailings dumps, but arrangements are now being made for the introduction of travelling belts, on to which the vats will discharge their contents.

For some time the standard vats in Africa were 40-ft. × 8-ft. and 38-ft. × 10-ft., but vats are now being made of a diameter as great as 60-ft. The sides are usually made of $\frac{3}{16}$-in. or $\frac{1}{4}$-in. steel plate, the bottoms being $\frac{1}{16}$-in. thicker. Sides of $\frac{3}{16}$-in. plate higher than 8-ft. are usually stayed by an outer ring of channel iron half way up the side. Conical vats have thicker bottoms.

Originally vats in Africa were built of wood. Later on, circular and square concrete vats were built at some of the largest plants, such as the "LANGLAAGTE ESTATE," "CROWN REEF," and "CITY AND SUBURBAN" plants, but owing to the difficulty of locating leakage, and the great expense of building, this type of vat is no longer constructed; steel plates being used exclusively.

Reference has already been made to the treatment of slimes, and some of the largest plants are designed on the principle of continuous discharge, from an outlet at the bottom of a conical receiving vat, of pulp containing a certain fixed percentage of moisture, only the clear effluent flowing over the periphery of the tank into a circular launder, by which it is conveyed back to the water dam.

The decantation slimes treatment process has already been described in Chapter IV, and, owing to the war, but few modifications have been recently made except in the design of the vats themselves.

The gold in the solutions is usually precipitated by zinc with or without previously deposited lead. Electric precipitation has survived at a few works only.

A neat method of cleaning up zinc boxes has been introduced into Africa, and consists of running the suction pipe of a filter-press into the boxes; the gold precipitate is thus collected as cakes in the filter-press, and may in this form be readily handled without loss, especially if thin filter papers have been placed over the cloths to prevent the latter being impregnated by the gold slime; the papers are of course burnt and the ashes smelted with the rest of the precipitate.

Elaborate furnaces have been designed in Africa for roasting the slimes, although the majority of operators prefer acid treatment. The bullion smelting furnace which has found most favour is a reverberatory, built to hold 6 or 10, or even more, No. 60 plumbago crucibles at one charge. The whole of the clean-up is thus smelted in one operation.

Dry crushing has found but little favour at Johannesburg, and doubtless the very good work done by the wet crushing mills has much to do with this.

A feature of African dry crushing practice is the coarse grade to which the material is reduced, in some cases the particles treated being as coarse as marbles, and in no case does dry crushing appear to have been carried to so fine a mesh as is prevalent in America, in New Zealand, and Western Australia.

The use of lime for precipitation of the slimes has led to a rather curious result, as it is found to increase the yield from the plates and also from the tailings, on account of the greater deposition of the fine gold on the plates and in the vats, to such an extent as in some cases to make the slimes no longer payable for treatment, and it thus looks as if the preliminary treatment with lime had come to stay, even although decantation slimes plants may not be erected.

Another feature of African practice is the huge crushing and sorting station usually at the shaft itself. By means of travelling belts or revolving tables the worthless ore is rejected by hand-picking, and the grade of the material, which is actually crushed in the mill, and on which the subsequent treatment charges are incurred, is thus very materially increased, and the expense of crushing and cyaniding worthless material avoided.

A typical African plant consists of—

Grizzleys.
Crushers (either Gate's or Blake's).
Sorting table or belt.
Mechanical haulage in trucks to the wet-crushing battery.
Ore bins.
Wet-crushing stamps weighing 1000/1250-lbs.
Copper plates 12-ft. long.

Tailings wheel.
Automatic sampler.
Spitzlutten (for concentrates and for sands).
Sand distributors (hose or revolving).
Upper collecting vats.
Percolating vats.
Mechanical haulage of tailings to residues dump.
Spitzlutten (for "return" sands).
Spitzkasten or settling tanks (to separate clear water from the slimes).
Conical slimes-settling vats.
Treatment vats in which the slimes pulp is agitated by centrifugal pumps.
Slimes pump and pipe-line to slimes dams or tailings heaps.
Extractor house equipment.
Also automatic samplers for concentrates from the spitzlutten and slimes from the spitzkasten.

In addition, some mines have a small separate percolating plant for treatment of the spitzlutten concentrates. Other mines have vanners, and send the richer concentrates thus produced to chlorination works.

In a huge industry such as mining in Africa, where operations are carried on under the high pressure of a healthy but keen emulation, each manager striving to make his costs less than those of his neighbours, there are one or two possible sources of loss in connection with local practice which deserve the most careful attention. Of course, these sources of loss are by no means peculiar to Africa, but the magnitude of the industry makes it appropriate to refer to them in this section.

In the first place, the desire to use the weakest possible solutions has had a tendency to lessen the extraction percentage, and it has been found desirable to revert to the 0·2 % to 0·25 % and even 0·3 % "strong" solutions formerly employed. It is bad economy to lose a shilling's worth of gold for the sake of a reduction of *2d.* per ton on one's cyanide consumption. Weaker solutions take longer time for extraction, and thus require a plant of perhaps double the size usually laid down—one permitting ten or twelve days' treatment, for example—or not infrequently the vats have to be discharged before their contents have been sufficiently treated, owing to the exigencies of plant accommodation. Moreover, it is still a moot point whether extremely weak solutions are not liable to precipitate a portion of their gold before they arrive at the extractor boxes.

Again, in the desire to economise water or to preserve the "balance" of the solutions, in some instances residues after lixiviation are washed with sump solutions instead of fresh water, and in other cases the water wash is insufficient in amount to displace the 15 % to 20 % of gold-containing solution contained in the residues as moisture. It will be seen that this practice may result in a very heavy loss of gold, unless the extractor-box work has been extremely good. But here again the tendency until quite recently was not towards really satisfactory work. Very coarse shavings, and a mere layer of them, were placed in the boxes, instead of the compartments being well filled and packed with zinc of the proper texture. So late as 1896 solutions carrying 2-dwt. per ton were not uncommon, and one manager of a well-known cyanide plant openly stated that his solutions after leaving the boxes ran 8-dwt.!

Increased supervision, however, has altered all this, and immediately before the war Johannesburg zinc-box work was probably as satisfactory as that of any territory in the world, with the exception of the silver-gold propositions of Mexico, New Zealand, and Eastern Asia, where the extractor work has always been extremely good, the solutions running from ½ to 2 grains of gold only per ton.

In Africa, too, they have now made a special feature in zinc-box work with extremely dilute and poor solutions. This is being done by zinc shavings which have been dipped in a 10 % solution of acetate of lead, so as to make use of a lead-zinc couple (*see* Chapter on Extractor Box Work).

It is hoped that when the war is over increased attention will be given to the loss of gold incurred in cleaning up the precipitate, treating it with acid, and melting into bars. This loss was formally estimated by the metallurgist of one of the large groups at from 4 °/o to 8 °/o. The author's investigations do not place it at nearly so high a figure, although injudicious attempts to refine gold by the use of nitre are undoubtedly responsible for extremely heavy losses, but estimating that the loss, whatever it is, say 1 °/o, may be diminished by ½ °/o by the means of a proved and carefully examined method of cleaning up, this ½ °/o on the total output of the Rand amounts to an annual saving which would far more than justify a considerable heavy initial expense—not that this would be required—on an investigation as to the best method of handling this precipitate.

AUSTRALIAN PRACTICE.

It is difficult to generalise owing to the diverse conditions prevailing in the different Australian Colonies. Thus, in Queensland, where there are perhaps the largest number of cyanide plants, these are mostly small and built of wood—steel vats not yet having been introduced—and used practically exclusively for the treatment of tailings. On the other hand, in New South Wales there are some very fine double-treatment plants built on most modern African lines.

In cyanide matters, however, West Australia naturally takes the lead from the large output produced by the process there, the total output of the majority of the mines being obtained from this source.

Details of the methods employed in the typical plants are given elsewhere in the special chapter on this subject (*see* Chapter VI—The Treatment of Sulpho-Tellurides).

So far, roasting appears to more than hold its own, providing the roasting has been properly and efficiently conducted, which was not always the case in the plants first erected in the Colonies. But whether or not roasting is employed, fine grinding appears to be absolutely essential to obtain high extractions in subsequent treatment, and accordingly the usual practice is to have grinding pans for this purpose.

In the special article above referred to the costs at the "GREAT BOULDER MAIN REEF" are stated at 26*s.* 7*d.* per ton, but these have now been reduced to 21*s.* mainly by lessening the consumption of cyanide.

Much of the difficulty at first experienced appears to have been due to the unsuitable method of firing the furnaces, but this has now been rectified, with resulting very high extractions.

Mechanical furnaces are well nigh universally employed, and the cost of the operation has been reduced to as low as 3*s.* per ton, including fuel.

"Griffin" mills do not appear to have had the success originally anticipated, and at the "GREAT BOULDER PERSEVERANCE," where they were introduced, it has been necessary to add pans for fine grinding.

Moreover, the "Griffin" mills require very careful feed of sized material, and are subject to very considerable wear and tear. The Ball mills thus appear to have more than held their own.

The type of filter-press preferred permits of washing the cakes in the frames, and each chamber has a separate valve, so that any one chamber may be cut out without interfering with the operations of the press. All feed channels are in lugs outside the plates, the surface of the latter being entirely free from orifices, so that the filter-cloths merely hang, and require no fitting, as was formerly the case. Different methods are in use for closing the very heavy headpieces, and some presses have recently been introduced in which the waste air from the montejus is utilised to actuate a piston for this purpose.

The cakes are usually 3-in. thick, but Mr. Simpson, of the "GREAT BOULDER MAIN REEF," has recently discovered that 6-in. and even 12-in. cakes are perfectly feasible in practice.

The filter-presses are filled by montejus, in preference to the three-throw pumps which were at first introduced for this purpose. It is claimed that the montejus form a more even cake, which does not separate in the press into layers of the various constituents of the pulp.

The precipitation of the gold from the solutions is effected by the agency of zinc shavings, even where bromo-cyanide is used.

An interesting feature of Australian practice is the smelting of the gold precipitate in Fabre du Faure zinc distilling tilting furnaces. The slimes are charged into the retort, and after fusion the molten gold is run out by merely tilting the furnace.

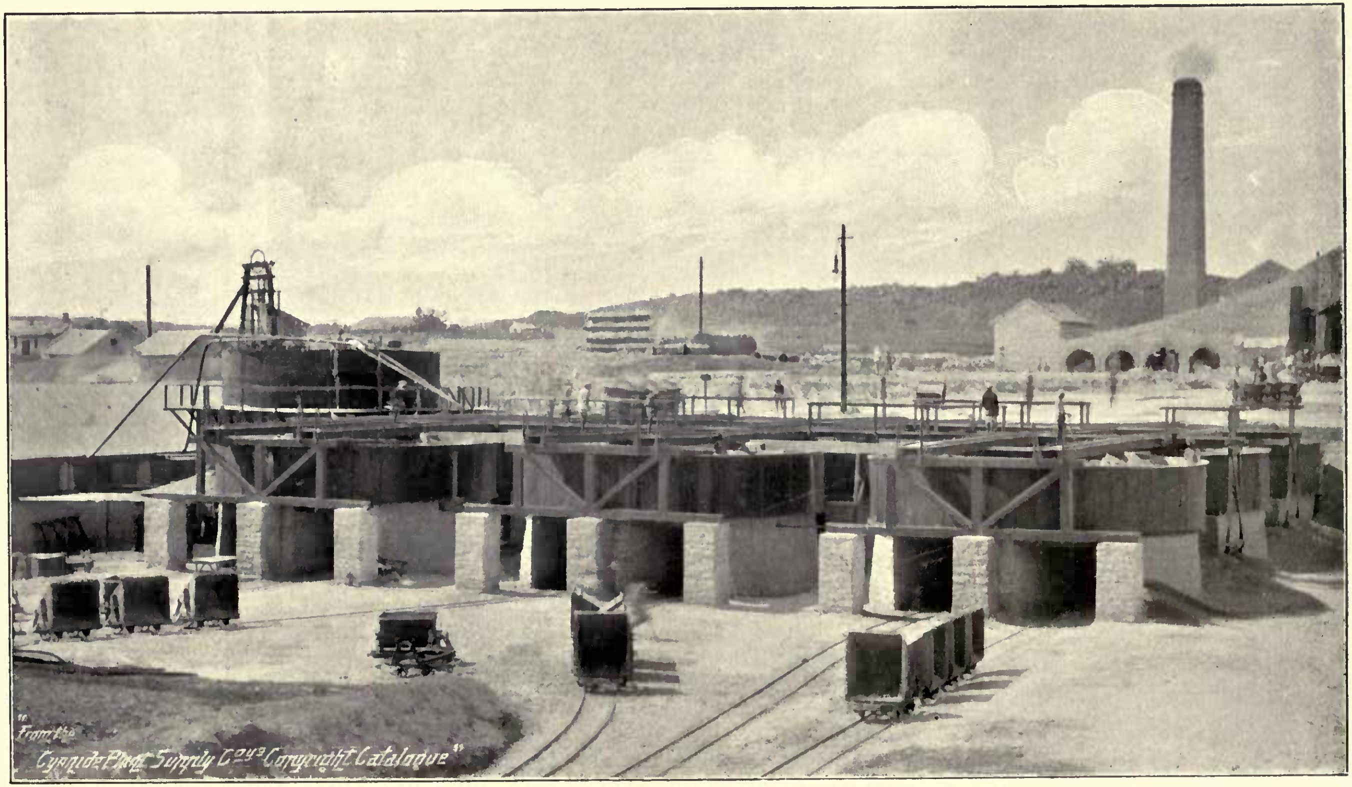
Owing to the prevalence in Western Australia of very rich ore, quite a considerable percentage of the output has been recovered by the use of smelting furnaces, the extraordinarily rich tellurides being bagged and shipped to smelting works for this purpose.

NEW ZEALAND PRACTICE.

A NUMBER of mines in New Zealand have been making experiments in the treatment of crushing the ore with dilute cyanide solutions flowing through the batteries, but owing to the difficulties in connection with the filtering of the crushed material and of the slimes, it can scarcely be claimed that the original results were remarkably successful, and accordingly for the last two years experiments have rather tended to proceed in the African direction of separating the slimes and treating these by settlement, agitation, and decantation, with or without the use of 0·1 % cyanide solution in the mortar boxes.

The chief plant in New Zealand is that of the "WAIHI COMPANY," in which dry crushing is employed on the oxidised ore, and the scheme of treatment is as follows:—

Kiln dryers.
Gate's crusher.
Trucks.



"From the Cyanide Plant Supply Co.'s Copyright Catalogue"

ONE OF THE LARGEST PLANTS IN INDIA.
The Mysore 10,000-Ton-a-Month Plant.

Dry stamp battery.
Worm conveyors.
Internal spiral conveyor.
Belt elevator.
Cross conveyor.
Two conveyors for vats.
Cyanide plant.

Vacuum suction is employed to assist the filtration of the weak solutions and washings. The cyanide consumption per ton appears rather heavy owing to the large amount of bullion recovered, frequently ten of silver to one of gold.

INDIAN PRACTICE.

In India are some of the largest cyanide plants in the world, both of steel and of cement, and probably the Indian Companies managed by Messrs. John Taylor and Sons hold the world's record, among the large and well-established gold mines of the world, for low first cost of plants and lowest cyanide consumptions and treatment costs; a cost of about 2*s.* per ton treated being not exceptional, whilst that of the "NUNDYDROOG COMPANY" is under 1*s.* 10*d.* per ton: of this 1*s.* is for cyanide, and maintenance, fuel, and other items are also included.

Practically all the Indian plants have been on single treatment lines—there having been large accumulations of tailings on hand for immediate treatment—but in the new plants, which take their charges direct from the mill, double treatment will be employed. Special provision will be made for the cheap handling of the residues.

The following is a list of some of the Indian plants, commencing with the original Mysore plant of 1894:—

Mysore Company - - - -	2,000	ton-a-month plant.	
" " - - - -	4,000	do.	
" " - - - -	10,000	do.	(Steel).

Champion Reef Company	8,000	ton-a-month plant (Steel).
” ” ”	4,000	do.
Ooregum	10,000	ton-a-month plant (Cement with steel collecting vats).
Nundydroog	4,000	ton-a-month plant (Steel).
Balaghat	2,500	ton-a-month plant.
Nine Reefs	2,500	do.
Goldfields of Mysore	2,000	do.
Road Block	1,500	do.

A remarkable feature of Indian practice is the large amount of mercury recovered from the extractor-box precipitate by retorting or roasting before fusion. This is due to the mercury floured during pan treatment, and similar results were encountered in Africa when the “Salisbury” pan tailings were treated.

OTHER COUNTRIES.

THE Nicaraguan “SANTA FRANCISCA COMPANY” have recovered quite a considerable amount of bullion from their slimes by treating it in pans with zinc and copper amalgam (*see* page 78), and in Eastern Asia this method has also been successfully put into practice. It is, however, more expensive than the filter-press method which has superseded it.

In countries such as Mexico, New Zealand, and the Dutch East Indies, a large amount of silver is produced in the ordinary cyanide treatment, frequently ten times as much silver as gold, and this renders an especial form of extractor box necessary to hold the great bulk of precipitate produced: from 5 to 15 ounces of bullion per ton treated.

In Borneo some very interesting cyanide practice is met with in the treatment of ore containing considerable percentages of antimony and arsenic. Fine crushing gave low extractions at a high cost for cyanide, but the schemes evolved at Bau and Bidi, of coarse crushing to the size of from road metal down to marbles, have given most satisfactory results, the bullion being extracted without appreciable interference by the base metal compounds. Most successful work is done in this district at a total cost of 5*s.* 6*d.* per ton, including mining and cyaniding.

CONCLUSION.

From a general consideration of the practice of the various countries of the world, we see that for the treatment of non-refractory ores wet crushing takes precedence owing to the large output usually obtained, and it may be added that for wet crushing the ordinary gravitation stamp still holds the field.

For dry crushing, stamps are employed on the extremely hard ores at Waihi, but ball mills continue to give an exceedingly good account of themselves, and are now extensively used not merely for soft ores but for hard ores also.

Rolls are favoured in America, but their limit of economical crushing seems to be about 800 meshes to the square inch.

The question of the successful treatment of slimes appears no longer to present unsurmountable difficulties. The African method of decantation is employed on very low grade material, as, although the percentage recovery is not high, the costs are low.

On the other hand, the West Australian filter-press method yields extractions as high as 94 °/₀, but it costs 6*s.* 6*d.* per ton (including cyanide treatment). It has, however, been so successful that the most recent methods of treatment designedly *reduce the whole of the ore to slimes* for subsequent filter-press treatment.

In roasting furnaces, the "Argall," the "Edwards," and the "Ropp" appear to be most successful, and the roasting of some hundreds of thousands of tons of sulphide ore at Cripple Creek at a cost of under 1*s.* 3*d.* per ton appears to create a world's record.

The high multiple hearth "MOUNT MORGAN" furnaces are found to have very serious limitations, both as regards the amount of rabbling necessary and as to the liability of accidents, such as the sudden shooting of a "green" charge down to the lowest hearth, where it thus mixes with and prejudices the treatment of the finished charges.

It is evident from the foregoing that the cyanide process is not at all the simple method so frequently claimed. An engineer has now the choice of coarse crushing with breakers only, and treating the ore in this condition, as in Africa, Borneo, and elsewhere; of fine crushing with stamp batteries, as at Waihi; with ball mills, as in Western Australia; with rolls, Griffin mills, Tustin mills, tube mills, and other revolving mills, as in America and Mexico; and of either roasting the fine product prior to cyanidation, or treating this direct without having recourse to furnaces. Or again, he may wet crush and amalgamate, or concentrate, or grind in pans, treating the tailings in vats, and the slimes either by decantation, by filter-press, or other method, as above described.

As to which method is the most suitable for its own particular ore, he may usually be guided to a considerable extent by local practice, but he will probably save a great deal of expensive experience by having recourse direct to a qualified expert on this subject.

By careful attention to the choice of proper methods it is now possible to *make considerable profits from tailings assaying only* 1 *dwt.*, and even ores may be mined (quarried), crushed, and cyanided, as in Borneo, *for* 5*s.* 6*d.* *per ton.* This opens out a vast field for the profitable treatment of low grade deposits.

One Thousand Grain Assay Table.

Weight in 1,000 grs.	Assay per ton.	Weight in 1,000 grs.	Assay per ton.	Weight in 1,000 grs.	Assay per ton	Weight in 1,000 grs.	Assay per ton.	Weight in 1,000 grs.	Assay per ton.	Weight in 1,000 grs.	Assay per ton.	Weight in 1,000 grs.	Assay per ton.
	oz. dwt. gr.		oz. dwt. gr.		oz. dwt. gr.		oz. dwt. gr.		oz. dwt. gr.		oz. dwt. gr.		oz. dwt. gr.
·001	0 0 16	·031	1 0 6	·061	1 19 20	·091	2 19 11	·121	3 19 1	·151	4 18 16	·181	5 18 6
·002	0 1 7	·032	1 0 22	·062	2 0 12	·092	3 0 2	·122	3 19 17	·152	4 19 7	·182	5 18 21
·003	0 1 23	·033	1 1 13	·063	2 1 4	·093	3 0 18	·123	4 0 9	·153	4 19 23	·183	5 19 13
·004	0 2 15	·034	1 2 5	·064	2 1 19	·094	3 1 10	·124	4 1 1	·154	5 0 15	·184	6 0 5
·005	0 3 6	·035	1 2 20	·065	2 2 11	·095	3 2 1	·125	4 1 16	·155	5 1 6	·185	6 0 20
·006	0 3 22	·036	1 3 12	·066	2 3 3	·096	3 2 17	·126	4 2 8	·156	5 1 22	·186	6 1 12
·007	0 4 14	·037	1 4 4	·067	2 3 18	·097	3 3 9	·127	4 2 23	·157	5 2 13	·187	6 2 4
·008	0 5 6	·038	1 4 20	·068	2 4 10	·098	3 4 1	·128	4 3 15	·158	5 3 5	·188	6 2 20
·009	0 5 21	·039	1 5 11	·069	2 5 2	·099	3 4 16	·129	4 4 7	·159	5 3 21	·189	6 3 11
·010	0 6 12	·040	1 6 3	·070	2 5 17	·100	3 5 8	·130	4 4 22	·160	5 4 12	·190	6 4 3
·011	0 7 4	·041	1 6 19	·071	2 6 9	·101	3 5 23	·131	4 5 14	·161	5 5 4	·191	6 4 18
·012	0 7 20	·042	1 7 10	·072	2 7 1	·102	3 6 15	·132	4 6 5	·162	5 5 19	·192	6 5 10
·013	0 8 2	·043	1 8 2	·073	2 7 17	·103	3 7 7	·133	4 6 21	·163	5 6 11	·193	6 6 2
·014	0 9 3	·044	1 8 8	·074	2 8 9	·104	3 7 21	·134	4 7 12	·164	5 7 3	·194	6 6 18
·015	0 9 19	·045	1 9 9	·075	2 9 0	·105	3 8 14	·135	4 8 4	·165	5 7 19	·195	6 7 9
·016	0 10 11	·046	1 10 1	·076	2 9 16	·106	3 9 6	·136	4 8 20	·166	5 8 11	·196	6 8 1
·017	0 11 2	·047	1 10 17	·077	2 10 7	·107	3 9 21	·137	4 9 11	·167	5 9 2	·197	6 8 16
·018	0 11 18	·048	1 11 9	·078	2 10 23	·108	3 10 13	·138	4 10 3	·168	5 9 18	·198	6 9 8
·019	0 12 10	·049	1 12 0	·079	2 11 15	·109	3 11 4	·139	4 10 19	·169	5 10 10	·199	6 10 0
·020	0 13 1	·050	1 12 16	·080	2 12 6	·110	3 11 20	·140	4 11 11	·170	5 11 1	·200	6 10 16
·021	0 13 17	·051	1 13 8	·081	2 12 22	·111	3 12 11	·141	4 12 3	·171	5 11 17	·300	9 16 0
·022	0 14 9	·052	1 13 23	·082	2 13 14	·112	3 13 4	·142	4 12 18	·172	5 12 9	·400	13 1 8
·023	0 15 0	·053	1 14 15	·083	2 14 5	·113	3 13 20	·143	4 13 10	·173	5 13 1	·500	16 6 16
·024	0 15 16	·054	1 15 7	·084	2 14 21	·114	3 14 11	·144	4 14 2	·174	5 13 16	·600	19 12 0
·025	0 16 8	·055	1 15 22	·085	2 15 12	·115	3 15 3	·145	4 14 17	·175	5 14 8	·700	22 17 8
·026	0 17 0	·056	1 16 14	·086	2 16 4	·116	3 15 19	·146	4 15 8	·176	5 15 0	·800	26 2 16
·027	0 17 15	·057	1 17 6	·087	2 16 20	·117	3 16 10	·147	4 16 1	·177	5 15 15	·900	29 8 0
·028	0 18 7	·058	1 17 21	·088	2 17 12	·118	3 17 2	·148	4 16 17	·178	5 16 7	1·000	32 13 8
·029	0 18 23	·059	1 18 13	·089	2 18 3	·119	3 17 17	·149	4 17 8	·179	5 16 23	2·000	65 6 16
·030	0 19 14	·060	1 19 4	·090	2 18 19	·120	3 18 9	·150	4 18 0	·180	5 17 14	3·000	98 0 0

Table of Internal Diameters, Circumferences, and Areas of Vats; tons of Water, and tons of Tailings (at 30 cubic feet per ton) per foot of Depth.

(Copyright.)

Diameter.	Circumference.	Area in square feet.	Tons of Water.	Tons of Tailings.	Diameter.	Circumference.	Area in square feet.	Tons of Water.	Tons of Tailings.
ft. ins.	feet		per foot	per foot	ft. ins.	feet		per foot	per foot
1 0	3·14	0·78	0·02	0·03	13 6	42·41	143·13	3·98	4·77
1 6	4·71	1·76	0·05	0·06	13 9	43·20	148·48	4·12	4·95
2 0	6·28	3·14	0·09	0·10	14 0	43·98	153·93	4·28	5·13
2 6	7·85	4·90	0·14	0·16	14 3	44·77	159·48	4·43	5·32
3 0	9·42	7·06	0·19	0·24	14 6	45·55	165·13	4·59	5·50
3 6	11·00	9·62	0·27	0·32	14 9	46·34	170·87	4·75	5·70
4 0	12·57	12·56	0·35	0·42	15 0	47·12	176·71	4·91	5·89
4 6	14·14	15·90	0·44	0·53	15 3	47·91	182·65	5·07	6·09
5 0	15·71	19·63	0·55	0·65	15 6	48·69	188·69	5·24	6·29
5 6	17·28	23·75	0·66	0·79	15 9	49·48	194·82	5·41	6·49
6 0	18·85	28·27	0·79	0·94	16 0	50·27	201·06	5·59	6·70
6 6	20·42	33·18	0·92	1·11	16 3	51·05	207·30	5·76	6·91
7 0	21·99	38·48	1·07	1·28	16 6	51·84	213·82	5·94	7·13
7 6	23·56	44·17	1·22	1·47	16 9	52·62	220·35	6·12	7·35
8 0	25·13	50·26	1·39	1·68	17 0	53·41	226·98	6·31	7·57
8 6	26·70	56·74	1·57	1·89	17 3	54·19	233·70	6·49	7·79
9 0	28·27	63·61	1·77	2·12	17 6	54·98	240·52	6·68	8·02
9 6	29·85	70·88	1·97	2·36	17 9	55·76	247·45	6·87	8·25
10 0	31·42	78·54	2·18	2·62	18 0	56·55	254·46	7·07	8·48
10 6	32·99	86·59	2·40	2·89	18 3	57·33	261·58	7·27	8·72
11 0	34·56	95·03	2·64	3·17	18 6	58·12	268·80	7·47	8·96
11 3	35·34	99·40	2·76	3·31	18 9	58·91	276·11	7·67	9·20
11 6	36·13	103·87	2·89	3·46	19 0	59·69	283·52	7·88	9·45
11 9	36·91	108·43	3·01	3·61	19 3	60·48	291·03	8·08	9·70
12 0	37·70	113·09	3·14	3·77	19 6	61·26	298·64	8·30	9·95
12 3	38·48	117·85	3·27	3·93	19 9	62·05	306·35	8·51	10·21
12 6	39·27	122·71	3·41	4·09	20 0	62·83	314·16	8·73	10·47
12 9	40·06	127·67	3·55	4·26	20 3	63·62	322·06	8·95	10·74
13 0	40·84	132·73	3·69	4·42	20 6	64·40	330·06	9·17	11·00
13 3	41·63	137·88	3·83	4·60	20 9	65·19	338·16	9·37	11·27

Allow 1 ft. extra for Filter Bottom and Water Space. Thus, a vat to hold 20 tons of Tailings, the Tailings to be a 3½-ft. layer, would be 20 ÷ 3½ = 5·7 tons per foot = 14 ft. 9 in. × 3½ ft. + 1 ft. for Filter and Water = 14 ft. 9 in. × 4 ft. 6 in.

To calculate quantity of Special Paint required to give two coats to interior of vats:—Divide internal area by 100; the result is the number of gallons required.

Table of Internal Diameters, Circumferences, and Areas of Vats; tons of Water, and tons of Tailings (at 30 cubic feet per ton) per foot of Depth—*Continued.*

(Copyright.)

Diameter.		Circumference.	Area in square feet.	Tons of Water.	Tons of Tailings.	Diameter.		Circumference.	Area in square feet.	Tons of Water.	Tons of Tailings.
ft.	ins.	feet		per foot	per foot	ft.	ins.	feet		per foot	per foot
21	0	65·97	346·30	9·62	11·55	28	6	89·56	637·94	17·72	21·26
21	3	66·76	354·66	9·85	11·82	28	9	90·32	649·18	18·03	21·64
21	6	67·54	363·05	10·08	12·10	29	0	91·11	660·52	18·35	22·02
21	9	68·33	371·54	10·32	12·38	29	3	91·89	671·96	18·67	22·40
22	0	69·12	380·13	10·56	12·67	29	6	92·68	683·49	18·99	22·78
22	3	69·90	388·82	10·80	12·96	29	9	93·46	695·13	19·31	23·17
22	6	70·67	397·61	11·04	13·25	30	0	94·25	706·86	19·64	23·56
22	9	71·47	406·49	11·29	13·55	30	3	95·03	718·69	19·96	23·96
23	0	72·26	415·48	11·54	13·85	30	6	95·82	730·62	20·29	24·35
23	3	73·04	424·56	11·79	14·15	30	9	96·60	742·64	20·63	24·75
23	6	73·83	433·73	12·05	14·46	31	0	97·39	754·77	20·97	25·16
23	9	74·61	443·01	12·31	14·77	31	6	98·96	779·31	21·65	25·98
24	0	75·40	452·39	12·56	15·08	32	0	100·53	804·25	22·34	26·81
24	3	76·18	461·86	12·83	15·40	32	6	102·10	829·58	23·04	27·65
24	6	76·97	471·44	13·10	15·71	33	0	103·67	855·30	23·76	28·51
24	9	77·75	481·11	13·36	16·04	33	6	105·24	881·41	24·48	29·38
25	0	78·54	490·88	13·64	16·36	34	0	106·81	907·92	25·22	30·26
25	3	79·33	500·74	13·91	16·69	34	6	108·39	934·82	25·97	31·16
25	6	80·11	510·71	14·19	17·02	35	0	109·96	962·11	26·72	32·07
25	9	80·90	520·77	14·47	17·36	35	6	111·53	989·80	27·49	32·99
26	0	81·68	530·93	14·75	17·70	36	0	113·10	1017·88	28·27	33·93
26	3	82·47	541·19	15·03	18·04	36	6	114·67	1046·35	29·07	34·88
26	6	83·25	551·56	15·32	18·39	37	0	116·24	1075·21	29·87	35·84
26	9	84·04	562·00	15·61	18·73	37	6	117·81	1104·46	30·68	36·82
27	0	84·82	572·56	15·90	19·09	38	0	119·38	1134·11	31·50	37·80
27	3	85·61	583·21	16·20	19·44	38	6	120·95	1164·15	32·34	38·81
27	6	86·39	593·96	16·50	19·80	39	0	122·52	1194·59	33·18	39·82
27	9	87·18	604·81	16·80	20·16	39	6	124·09	1225·42	34·04	40·85
28	0	87·96	615·75	17·10	20·53	40	0	125·66	1256·64	34·91	41·89
28	3	88·75	626·80	17·41	20·89						

Allow 1 ft. extra for Filter Bottom and Water Space. Thus, a vat to hold 20 tons of Tailings, the Tailings to be a 3½-feet layer, would be 20 ÷ 3½ = 5·7 tons per foot = 14 ft. 9 in. × 3½ ft. + 1 ft. for Filter and Water = 14 ft. 9 in. × 4 ft. 6 in.

To calculate quantity of Special Paint required to give two coats to interior of vats:—Divide internal area by 100; the result is the number of gallons required.

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Mining
dept