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OF THE

## Chemical, Metallurgical and Mining Society

OF SOUTH AFRICA.

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### Proceedings

AT

### Ordinary General Meeting, August 17, 1907.

The Ordinary General Meeting of the Society was held in the Chamber of Mines, on Saturday, August 17th, Prof. J. Yates (President), in the chair. There were also present:—

38 Members: Messrs. T. L. Carter, F. Alexander, R. G. Bevington, W. R. Dowling, K. L. Graham, A. Heymann, A. McA. Johnston, A. Richardson, Prof. G. H. Stanley, H. A. White, Prof. J. A. Wilkinson, W. Beaver, G. Bernfeld, E. H. Croghan, N. M. Galbraith, G. Goodwin, J. Gray, F. G. Guthrie, W. H. Jollyman, J. A. Jones, J. Kennedy, C. B. Kingston, A. Kressmann, J. Lea, C. W. Lee, D. McKerrell, G. Melvill, J. T. Milligan, P. T. Morrisby, D. J. Pepler, O. D. Ross, W. Sharp, H. Taylor, J. A. Taylor, W. Taylor, J. P. Ward and F. W. Watson.

8 Associates and Student: Messrs. J. H. Harris, R. W. Leng, R. Lindsay, C. A. Robinson, C. Toombs, W. Waters, L. J. Wilmoth and J. Innes.

12 Visitors and Fred. Rowland, Secretary.

The minutes of the previous monthly meeting, as published in the *Journal*, were confirmed.

#### NEW MEMBERS.

Messrs. Gray and Croghan were elected scrutineers, and after their scrutiny of the ballot papers, the President announced that all the candidates for membership had been duly elected, as follows:—

BALLANTINE, ROLLO BOWMAN, 11 and 12, Steytler's Buildings, P. O. Box 1136, Johannesburg. Civil Engineer.

EVANS, A. W., French Rand G. M. Co., Ltd., P. O. Box 25, Luipaardsvlei. Assistant Surveyor.

METCALF, JOSEPH ERNEST, P. O. Box 209, Germiston. Chemist.

WIGHT, A., Randfontein Estates and G. M. Co., Ltd., P. O. Box 2, Randfontein.

The Secretary announced that His Excellency the High Commissioner had been pleased again to accept the Hon. Presidency of the Society, and that the Council had also elected the following new Hon. Vice-Presidents: Mr. de Villiers (Minister of Mines), Mr. Reyersbach (President of the Chamber of Mines) and Mr. George Albu.

The following gentlemen had been admitted Associates by the Council since the last general meeting.

BROWN, WILLIAM WAUGH, Mysore G. M. Co., Ltd., Marikuppam, Mysore State, South India. Chief Reduction Officer.

GERARDY, BARON GEORGE, Geldenhuis Deep, Ltd., Cleveland.

MAXWELL, ROBERT KERR, c/o Mr. J. Hiddleston, P. O. Box 140, Fordsburg. Chemist.

NEWLAND, NOEL NELSON, Robinson Deep G. M. Co., Ltd., P. O. Box 1488, Johannesburg. Cyanider.

RHODES, CLARENCE EDGAR, El Oro Mining and R. R. Co., Ltd., El Oro, Mexico. Cyanide Manager.

THOMAS, WILLIAM, Champion Reef G. M. Co., Ltd., Mysore State, South India. Assistant Cyanide Chemist.

**Mr. McArthur Johnston:** I beg to propose a very hearty vote of thanks to Mr. Rowland for the excellent index which accompanied the last issue of the *Journal*.

**The President:** I am sure we are all in sympathy with the motion. It is only when one comes to arrange an index oneself that we realise the amount of work it entails.

#### PATENTS PROCLAMATION AMENDMENT BILL.

**The President:** The proposed Patents Proclamation Amendment Bill has already been brought to your notice. You remember that the Government some time back appointed a Commission to investigate the system of administration of the Patents Office and other matters pertaining to patents and designs. The Commission took evidence and its Report has just been issued. May I place a few of their conclusions before you:

(a) In the interest alike of inventors and of the public it is undesirable to restrict the grounds on which oppositions may now be entered to an application for a patent.

(b) As stated in the Public Service Commission, the principle of having a trained officer to deal with such oppositions in the first instance is a sound one.

(c) There is not sufficient work arising in the Transvaal Patent Office as at present constituted to keep an officer fully occupied on his present work only.

(d) It is highly desirable that there should be uniformity of Patent Law throughout South Africa, and if that were achieved circumstances would warrant the appointment of a Commissioner of high scientific and legal attainments to carry out for South Africa the functions at present performed by the Commissioner in the Transvaal.

(e) The abolition of the office of the Commissioner of Patents with his present duties and qualifications would be a retrograde step from the point of view of the public and inventors alike; if, therefore, the above proposal (d) is absolutely impracticable, it is desirable that the Commissioner of Patents in the Transvaal should also be available as legal adviser to the Attorney General.

(f) If neither of the two previous proposals (d) and (e) are possible, patent oppositions in the Transvaal should be heard in the first instance by a Judge of the Supreme Court, subject to an appeal to the full Court.

(g) In that event the officer in charge of the Patent Office should be confined to administrative duties. He should, however, still be a gentleman with legal training, and the office should remain, as at present, a separate department.

You will gather from the above that the Commission, with the exception of Mr. Jutta, advocates, and I am justified in saying, strongly advocates, the retention of a Commissioner of high scientific and legal attainments, but in the face of this finding, and in opposition to the unanimous opinion of Scientific Societies in Johannesburg, the Government has introduced, and yesterday finally passed, a Bill providing for the withdrawal of the judicial functions of the Commissioner of Patents, thus lowering this official's status, and transferring the judicial functions to a Judge of the "expensive" Supreme Court.

The Attorney-General assured the Legislative Council on Thursday that the Government had acted largely on the advice of Mr. Bucknill, the late Commissioner of Patents, but why they should ignore the finding of the Commission specially appointed to investigate the subject is difficult to understand; there would appear to be a direct saving, by the new arrangement, of about

£1,000 per annum in the department, but as a debit against this we have, in the opinion of many competent to judge, such additional trouble and expense thrown upon inventors as will deter many from coming forward, and this will lead to a loss of revenue, both directly and indirectly, altogether out of proportion to the direct saving mentioned. And this is quite apart from the question of the advisability of having a really highly trained technical man to deal with opposition in the first instance, a point emphasised by the Commission.

There is, however, some satisfaction to be derived from the Attorney-General's statement to the effect that the Government propose to deal more fully with the whole subject of designs, trade marks and patents next year, and that it hopes to come to some arrangement with the other South African Colonies regarding these matters, and realises that, in the event of such an arrangement being arrived at, it might be advisable to have one man in charge; we are also given to understand that the present legislation is merely temporary.

We lodged an objection to the proposed Bill some weeks back, but without avail, and this week, in conjunction with the South African Association of Engineers and the Institute of Mechanical Engineers, we wired a request that the Government should receive a deputation from the Societies concerned, before finally passing the Bill. No heed was paid to our request, but we are indebted to several of the Honourable Members of the Legislative Council for taking the matter up and for assisting us in our endeavours to prevent, what we believe to be, retrogressive and injudicious legislation.

We take very strong exception to the opinion of one of the Hon. Members of the Council that the views of the Scientific Societies of this Colony should not be considered in such matters. Our member and the members of the sister Societies are pre-eminently the men who have to deal with inventions and patents, and their views are therefore entitled to respect, a respect which has hitherto not been denied them. If the Government will not accept the advice of its own Commission, and that of men whose business it is to deal with patent matters, whose advice will it accept, and by whom will it be guided?

The Bill being finally passed we can only hope that next year will bring forth legislation of a more satisfactory nature.

**Mr. Fred. Rowland:** I am requested to express the regret for the absence to-night of representatives of the S. A. Association of Engineers, who, owing to various unforeseen circumstances are unable to be present. I am, however, authorised to state that they are in entire agree-

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ment with the action we have taken in this matter in protesting so emphatically at the action of the Government in ignoring the recommendations of their own Commission and in hurrying through Parliament a Bill which will tend to increase the cost, despite the statements of the Attorney-General and Mr. Juta to the contrary, of patenting inventions, should there be any opposition, in this Colony.

In order to show you how hurriedly this matter has been dealt with, I beg to submit the following facts. You will remember that at our June meeting, after certain information came to our knowledge, a resolution expressing the views of the Society was passed. The resolution was forwarded to the Attorney-General and also to a number of members of the Legislature on the 19th of June, and on the 27th June a reply was received from the Attorney-General acknowledging its receipt and referring us to the reply given to Dr. Macaulay on the 25th June, which stated that the matter was receiving the careful consideration of the Government. We heard nothing further from the Government on the subject, and as we had no idea of their intentions, it was not possible for us to make any further representations. On the 2nd August the Attorney-General stated in the Assembly that on the 6th inst. he would ask for leave to introduce the amending Bill. The Government Notice, No. 866, accompanying the Bill emanated from the Prime Minister's office, and is dated the 2nd inst.

The Bill was published in an *Extraordinary Supplement to the Government Gazette* on the 3rd inst. (Saturday). Monday was a Public Holiday and even had we known of its existence it was practically impossible for us to have obtained a copy before the 6th inst. I might here mention, that it was only after the application by Mr. Chaplin, on the 11th July, that the Report of the Patents Office Administration Committee was laid on the table of the House on the 16th July. Even at this date the authorised Government issue of the Report will not be available for the public, so I am informed, for about another fortnight, although extracts have been published in the local press. The second reading of the Bill was carried after a protest had been made by Mr. Chaplin, who, I understand, acted in consultation with Dr. Macaulay in the matter, on the 12th inst., and the third reading on the 13th inst. The Bill went to the Upper House on the 14th inst. and was finally passed, as you have seen, on the 15th inst., altogether fairly rapid progress. The Attorney-General complains of our action in telegraphing to members of the Council and states that we should have written or consulted members on the subject. I might mention here that our wire was not the cause (as the Hon.

Mr. van der Merwe stated) of the business of the Upper House being wasted for a whole day. As the result of our resolution I may inform you that several Hon. Members had gone very thoroughly into the matter, and although it was not reported, the subject was very thoroughly debated in the Council, and the points we had raised brought well before the notice of the Government. Your thanks are due to the Hons. W. Dalrymple, W. A. Martin, T. A. R. Purchas and R. Feetham, for the very thorough manner in which the matter was dealt with, as also to the Hon. Mr. Robertson for his reply to Mr. van der Merwe. I need hardly say that it is a very difficult matter to obtain a representative meeting of the Councils of the several Societies at short notice, members of which are busy men and situated all along the Reef, but there was ample time between the end of June and the beginning of August for the Government to have asked and received the opinions of all those interested in this Bill, as used to be done in the past. There has really been no time in this instance for our having an opportunity of going into the Bill and making further representations to the Government.

The Attorney-General has been advised that the compilation of an Index of Transvaal Patents would cost £10,000. He also states that the number of patents applied for annually is very small. If the latter statement is correct, the cost of this index seems enormous. The Hon. Mr. Wolmarans rightly mentioned that we objected to the Bill on the grounds of the increased cost, but he is further reported to have said, we should pay the increased cost, a statement which will not appear encouraging to the inventor, who, as a rule, is only able at present to pay the existing charges with an effort. After our existence of 13 long years, during which period the press of this city have given much publicity to our work, we find an Hon. Member unaware of this Society's existence. I have taken the opportunity of enlightening him on this subject, and I am sure you will endorse the cordial invitation, which I have sent to him to attend our meetings.

Mr. A. C. Whittome (Past President of the Institute of Mechanical Engineers): It gives me great pleasure on behalf of the Transvaal Institute of Mechanical Engineers to support the vigorous protest you are making against the action of the Government. I think the Hon. Member of the Legislative Council who was unaware of the existence of this Society ought to be placed in our museum as one of the curiosities of the Transvaal. He must be the only public man in the country who is unaware of it. Together with your Association, the Transvaal

Institute of Mechanical Engineers, to which I have the honour to belong, has done its best to prevent this Bill being hurriedly passed through all its stages in both Houses of Parliament. As you know, we made a great effort at the time the Commission on Patent Office Administration was sitting. We laid our views before the Commission and pointed out that so far from taking any retrograde step, the Government should progress, that they should give us more than we had had in the past, and all that we are entitled to from our representative character. The Hon. the Attorney-General stated in his remarks the other day that there was not a single individual, who could say that he had been treated with scant courtesy or consideration. I almost think that in saying this the Hon. gentleman was playing with words. Surely the Councils of the three big Associations here, representing over 1,700 technical men of the Rand, surely, they have not only been treated with scant consideration but practically with no consideration at all. It may be perfectly correct that individual men have received every consideration, but certainly the interested parties on the inventor's side have received practically nothing at the hands of the Hon. the Attorney-General or the Government. It appears that the inventor is always to be the man who receives last and least consideration. Those of us who have had anything to do with opposed applications for patents know that even since the war the cost of opposition is tremendous. Under the new regulation the opposition is to be heard before the Supreme Court, and the cost will be very considerably increased. Some gentlemen have gone so far as to say that the cost will be ten times the amount it used to be; this is probably too high an estimate, but it will certainly be 100 per cent. higher. Yet these inventors are the men who are helping to build up the industries of the Colonies. It was pointed out in the evidence before the Commission that a saving of 10d. per ton by improved methods in the gold mining industry alone meant about £62,000 to the Government per annum in profits tax, yet for a paltry £1,000 a year they have handicapped the men who have saved, many times over, that gain. I hope this Association will, and I am sure my own Association will, continue their efforts to get this ridiculous Bill withdrawn from the statutes of the country. We are told apparently as a last thought of the Hon. the Attorney-General, that it is only a temporary measure. Possibly they have something in reserve for next year which will be even worse than this, but I hope that the whole of the Scientific Associations of the Transvaal will continue their efforts and see that this time next year, if possible, we get not only what we had before but something better.

**The President:** There the matter stands, and the only satisfactory feature about it is the possibility of things being put right next year.

During the week I have received a letter from one of our Past-Presidents, Mr. Crosse, who is at present at Frankfurt, Pilgrim's Rest, and who suggests that part of it may interest you, so I will read it.

#### ASSISTING THE SOLUTION OF GOLD IN THE CYANIDE PROCESS BY COMPRESSED AIR.

By ANDREW F. CROSSE (Member).

"For many years I have been impressed with the idea of using compressed air, produced by a "trompe," for assisting in the solution of gold in the cyanide process. Time forbids my entering into a description of the "trompe," but I believe that a "trompe" was first used in the Pyrenees, during the middle ages, for iron smelting. In this district (Pilgrim's Rest) we have abundance of water-power, and I erected a small "trompe" on my arrival here last October.

More than half the gold here is contained in the slime, so that a high extraction from the latter is absolutely necessary. You are probably acquainted with the Pohlé pump, which is the exact opposite of the "trompe," the latter being an arrangement by which compressed air is formed, by being sucked down by water and delivered under pressure, whilst the Pohlé pump is an arrangement by which water, or any other liquid, can be lifted, under given conditions, owing to the reduction of its specific gravity, by the admixture of air. I make use of a conical vat, with a vertical pipe in the centre of the same, through which the pulp in the vat is lifted by compressed air, supplied by the "trompe," 2 ft. above the surface of the liquid; it falls over into an outer tube and is discharged through four pipes, having their orifices tangentially arranged so as to force the pulp into a circular motion. By this means I obtain full aeration and very rapid circulation. The solution of the gold is extremely rapid, and there is no machinery of any kind, it being absolutely automatic.

As this method was so very successful, I applied the same principle, slightly modified, to the sand treatment. The sand must be perfectly clean and free from slime, such as is obtained from the Wilfley concentrator. I have arranged a vat from which the solution drains off into an air-lift pump; this solution is circulated through the sand. After a few hours the sand is full of air bubbles, and

a rapid circulation of the liquid is maintained. These air bubbles are formed owing to the cyanide solution being surcharged with air, and they are deposited on the sand, in the same way that a piece of bread, thrown into a glass of champagne, is coated with carbonic acid bubbles. I have obtained 92 per cent. extraction of the gold by this means from 6.2 dwt. tailings in 24 hours, including washing. This result is, I believe, a record."

The President said they were all pleased to hear from Mr. Crosse, and expressed the hope that he would soon contribute a paper embodying his experience in the Pilgrim's Rest district.

### NOTES ON THE ESTIMATION OF CAUSTIC LIME.

By EDW. H. CROGHAN (Member).

I must, to a certain extent, ask for your forbearance for introducing this subject. You well know that the changes which occur on burning ordinary limestone are such, that besides water and carbon dioxide being driven off, ferrous carbonate and carbonate of manganese become ferric oxide and manganese dioxide, and organic matter is burnt or at least decomposed, leaving a small residue of carbon; also, most important of all, the intermixed silicates are attacked, becoming so basic by combination with lime that their bases become soluble in hydrochloric acid, and their silica, on treatment with hydrochloric acid, is partly dissolved, and partly separates as metasilicic acid. Intermingled quartz undergoes scarcely any change in lime burning. On the other hand, if this quicklime be exposed to the action of the air, it reverts, gradually absorbing first water and then carbonic acid, and this reversion will be rapid or slow, according to the state of the atmosphere. Partly in consequence of this, it has always been and may possibly still be a vexed question, as to what is the best method extant for estimating the uncombined lime, if I may so call it, when it occurs admixed with its compounds and other constituents. In the title of this paper, it will be noted that the phrase caustic lime is employed. I adopt this expression in preference to such terms as "available lime" or "soluble lime," for in my opinion it seems a closer definition of this uncombined or free lime.

By "available lime" I should understand not only caustic lime, but also that existing as carbonate and other compounds, which would become available, as we understand the term

here, if brought under suitable conditions. On the other hand, "soluble lime" is hardly a technical designation, being really used in a popular sense, and being therefore lax. Other expressions might possibly be suggested by some of our members and discussed.

Regarding the existing methods for determining caustic lime, I am not aware of any special literature which has tended to prove or disprove their accuracy. At present there seem to be two methods in vogue locally. One is the gravimetric and the second is the volumetric sugar process. Previous to the appearance of Mr. G. W. Williams' paper on "Notes on Lime, Clean-up, etc." before this Society, in which he recommended the sugar method, the gravimetric process was principally employed. At that period many chemists, myself included, were very sceptical as to the quantitative value of the sugar procedure. From my experience I found that the gravimetric process, as usually calculated, always yielded results much higher than those given by the sugar method. Therefore, with a view of trying to account for these discrepancies, I carried out a series of analyses and tests on certain burnt limes as supplied to the mines. Time did not permit me to go into all the detail I would have desired to, so I take this opportunity of submitting to you the various data I have obtained so far.

On receipt of the samples, they were quickly crushed and passed through a 60 mesh sieve. An average portion of each was taken and transferred to a small glass jar, the cover of which could be screwed down tight on to a caoutchouc ring, thereby ensuring complete protection from the air. Samples of burnt limes may be kept in this way for months without showing any appreciable deterioration. In all 9 burnt limes were completely analysed, namely, 8 white limes, Nos. 1 to 8 inclusive, and one so called "blue lime" No. 9.

The table referring to the analyses will be found on the next page.

For comparison these samples were also analysed by the sugar method, with the following results as to true caustic lime:—

1	2	3	4	5	6	7	8	9
%	%	%	%	%	%	%	%	%
63.00	75.50	71.05	63.35	74.72	73.85	64.40	70.35	30.80

The sugar process was carried out as follows: Two grammes of the sample were transferred to a litre flask, and then filled up to the mark with a sugar solution containing 20 gm. per litre.

TABLE I.

	1	2	3	4	5	6	7	8	9
	White Lime.	Blue Lime.							
Quartz and Insoluble Matter ...	3.48	2.23	2.22	5.60	2.63	2.61	2.82	3.32	6.11
Soluble Silica ...	0.51	0.66	0.40	1.05	0.75	0.73	0.51	1.18	1.54
Fe <sub>2</sub> O <sub>3</sub> ...	0.32	0.08	0.12	0.16	0.40	0.44	0.48	0.72	2.24
Al <sub>2</sub> O <sub>3</sub> ...	0.27	0.09	0.13	0.24	0.62	0.36	0.19	0.12	2.56
CaO (total) ...	77.32	85.84	83.88	75.32	84.00	83.76	74.36	82.20	44.10
MgO ...	0.91	1.02	0.93	0.68	0.78	0.95	0.59	0.85	24.71
MnO <sub>2</sub> ...	nil.	2.79							
SO <sub>3</sub> ...	0.61	0.44	0.39	0.94	0.47	0.49	0.35	0.59	0.79
P <sub>2</sub> O <sub>5</sub> ...	trace.	trace.							
CO <sub>2</sub> ...	8.05	5.00	6.78	5.30	5.00	5.05	5.25	5.75	3.60
Moisture, Water of Hydration, Carbon, etc.	8.50	4.92	5.27	10.62	5.50	5.72	14.69	4.75	10.42
TOTAL ...	99.97	100.28	100.12	100.11	100.15	100.11	99.24	99.48	98.86

The mixture was thoroughly shaken off-and-on for about an hour, allowed to stand over-

night, shaken again for a minute or two, and an aliquot portion (250 c.c.) then taken and titrated by N.HCl. employing phenolphthalein as indicator. The tests were carried out at least in duplicate and checked with themselves exceedingly well, the variation never exceeding 0.5 per cent.

Discussion of former methods of calculating caustic lime.—In the older gravimetric method of ascertaining the quantity of caustic lime, such detail as is given in the preceding analyses was not gone into. All that was essential was to estimate the total lime (calculated as CaO) and the carbon dioxide, and to calculate the latter into carbonate of lime, and subtract this from the total lime, the balance being assumed to be caustic lime.

On carrying out the calculation as above stated, the following would be the results for caustic lime :—

TABLE II.

	1	2	3	4	5	6	7	8	9
	per cent.								
CaCO <sub>3</sub> ...	18.30	11.36	15.41	12.04	11.36	11.48	11.93	13.07	8.18
CaO ...	67.01	79.48	75.25	68.58	77.64	77.33	67.70	74.88	39.52

Percentage error of old gravimetric method :—

TABLE III.

1	2	3	4	5	6	7	8	9
per cent.								
6.46	5.27	5.91	8.25	3.90	4.71	5.12	6.44	28.31

Now, if these results are compared with those obtained by the sugar method, you will note that the caustic lime figures of the former are much higher than those of the latter. If the difference between the two methods on the figures here obtained be expressed in percentages of the caustic lime present as in above table (III.) it will be clearly seen that the error is a very serious one, especially in blue lime.

It, therefore, seemed to me that either (1) the sugar method is quite unreliable although checking with itself, or (2) in the case of the gravimetric process there are other factors that should be considered. Of course, it was also possible that both methods are incorrect. I was then somewhat nonplussed for some third method to decide between these suppositions, and eventually adopted the principle of one from Sutton's "Volumetric Analysis." As distinct from the "sugar" I shall term this the "acid" method. The process as devised and eventually carried out was as follows:—2 gm. of a sample is transferred to a vessel containing some 150 c.c. to 200 c.c. boiling distilled water, previously kept boiling some five minutes, and now boiled for a further two or three minutes, constantly stirring.

It is then kept gently simmering and titrated by *N*.HCl with phenolphthalein as indicator until disappearance of colour remains constant through half-a-minute.

Results of acid method:—

TABLE IV.

1 per cent.	2 per cent.	3 per cent.	4 per cent.	5 per cent.	6 per cent.	7 per cent.	8 per cent.	9 per cent.
63.42	75.74	71.47	63.63	75.04	74.76	64.68	70.70	Not Applicable.

On comparison of the figures with those of the sugar method, one sees them to be only slightly higher. This may possibly be due to the small amounts of oxide of magnesium present partly going into solution, and therefore being expressed as caustic lime. Sutton states the method is applicable in the presence of carbonates of lime and oxide of magnesium. My experience confirms the former, but certainly not the latter statement. In the case of the blue lime I could not apply the method, as not only was an excessive quantity indicated, (being far more than the total lime found), but the end point, owing to the suspension of the dark sample, was very difficult to see. As an experiment I added magnesia to one of the white limes, and found a much higher quantity of lime than before! Fortunately these eight white limes contained very small percentages of magnesia and, therefore, this method was quite applicable, the percentage differences from the sugar method being still plus, but quite small, thus:—

Percentage error of acid method.

1 Per cent.	2 Per cent.	3 Per cent.	4 Per cent.	5 Per cent.	6 Per cent.	7 Per cent.	8 Per cent.
0.66	0.32	0.59	0.44	0.43	1.23	0.43	0.50

It is evident therefore that in the absence of excess of magnesia, the two methods agree very closely. I came thus to the conclusion that the gravimetric method must be incomplete and therefore made the complete analyses given in Table I.

From the results there obtained it seemed evident to me that besides the CO<sub>2</sub> present, the consideration of the other acid radicals, such as SO<sub>3</sub>, Al<sub>2</sub>O<sub>3</sub>, and soluble to (combined) silica might probably afford some clue to the higher figures calculated for caustic lime from the ordinary gravimetric process. I therefore carried out a series of calculations taking into consideration certain possible compounds of lime only. These compounds were as follows:—

- (1) CaCO<sub>3</sub>. (2) CaSO<sub>4</sub>. (3) Ca<sub>3</sub>Al<sub>2</sub>O<sub>6</sub>. (4) Ca<sub>2</sub>Si<sub>3</sub>O<sub>8</sub> (trisilicate) (partly fusible at temperature of assay furnace). (5) CaSiO<sub>3</sub> (bisilicate) (fusible at temperature of assay furnace). (6) Ca<sub>4</sub>Si<sub>3</sub>O<sub>10</sub> (sesquisilicate) (fusible at temperature of assay furnace). (7) Ca<sub>2</sub>SiO<sub>4</sub> (monosilicate) (infusible). (8) Ca<sub>4</sub>SiO<sub>6</sub> (basic silicate) (infusible).

Other aluminates or silicates may be formed, but the above are well known ones, and I therefore take them for the purposes of illustration and subsequent calculation of the caustic lime. I assume all through that magnesia, being a weaker base than lime, remains uncombined as MgO.

The various tables, giving the result of subtracting each lime salt in turn are to be seen on the next page; the figures are in per cent.

If the various data which have been obtained for caustic lime be now summarised, we shall be better able to compare them. They follow in the order of (1) gravimetric process and the subsequent deductions from the calculations, (2) the acid and (3) the sugar method.

With the last supposition, therefore, viz. (4 *e*), I have arrived at a theory which approximates really closely to the figures of the sugar and acid methods, being in fact sometimes higher and sometimes lower. If the differences between the two are stated in percentages, it will accordingly be seen how low or high the sugar method is:—

	1	2	3	4	5	6	7	8	9
	%	%	%	%	%	%	%	%	%
High	2.23	1.41	3.13	0.41	—	—	1.29	—	—
Low.	—	—	—	—	1.64	0.20	—	0.68	5.81

The only high difference is that of the blue lime, No. 9, which is most probably due to the large amount of magnesia it contains, and there-

TABLE V.

	1	2	3	4	5	6	7	8	9
2. CO <sub>2</sub> and SO <sub>3</sub> into CaCO <sub>3</sub> and CaSO <sub>4</sub> .									
CaSO <sub>4</sub> subtracted ...	1.03	0.74	0.66	1.60	0.80	0.83	0.59	1.00	1.34
CaO left ...	66.65	79.18	74.98	67.92	77.31	76.99	67.44	74.47	38.97
3. CO <sub>2</sub> , SO <sub>3</sub> and Al <sub>2</sub> O <sub>3</sub> (calculated as Ca <sub>3</sub> Al <sub>2</sub> O <sub>n</sub> ).									
Ca <sub>3</sub> Al <sub>2</sub> O <sub>6</sub> subtracted ...	0.71	0.24	0.34	0.63	1.64	0.95	0.50	0.32	6.77
CaO left ...	66.21	79.03	74.77	67.53	76.29	76.40	67.13	74.27	34.76
4A. CO <sub>2</sub> , SO <sub>2</sub> , Al <sub>2</sub> O <sub>3</sub> , as in (3), and soluble silica as Ca <sub>2</sub> Si <sub>2</sub> O <sub>8</sub> .									
Ca <sub>2</sub> Si <sub>2</sub> O <sub>8</sub> subtracted ...	0.82	1.07	0.64	1.70	1.21	1.18	0.82	1.91	2.49
CaO left ...	65.90	78.62	74.53	66.88	75.83	75.95	66.82	73.54	33.81
4B. CO <sub>2</sub> , SO <sub>3</sub> , Al <sub>2</sub> O <sub>3</sub> , as in (3), and soluble silica CaSiO <sub>3</sub> .									
CaSiO <sub>3</sub> subtracted ...	0.98	1.27	0.77	2.03	1.45	1.41	0.98	2.28	2.97
CaO left ...	65.74	78.42	74.40	66.55	75.59	75.72	66.66	73.17	33.33
4c. CO <sub>2</sub> , SO <sub>3</sub> , Al <sub>2</sub> O <sub>3</sub> , as in (3), and soluble silica as Ca <sub>4</sub> Si <sub>3</sub> O <sub>10</sub> .									
Ca <sub>4</sub> Si <sub>3</sub> O <sub>10</sub> subtracted ...	1.14	1.48	0.89	2.35	1.68	1.63	1.14	2.64	3.45
CaO left ...	65.58	78.21	74.28	66.23	75.36	75.50	66.50	72.81	32.85
4D. CO <sub>2</sub> , SO <sub>3</sub> , Al <sub>2</sub> O <sub>3</sub> , as in (3), and soluble silica as Ca <sub>2</sub> SiO <sub>4</sub> .									
Ca <sub>2</sub> SiO <sub>4</sub> subtracted ...	1.46	1.89	1.14	3.01	2.15	2.09	1.46	3.38	4.41
CaO left ...	65.26	77.80	74.03	65.57	74.89	75.04	66.18	72.07	31.89
4E. CO <sub>2</sub> , SO <sub>3</sub> , Al <sub>2</sub> O <sub>3</sub> , as in (3), and soluble silica as Ca <sub>4</sub> SiO <sub>6</sub> .									
Ca <sub>4</sub> SiO <sub>6</sub> subtracted ...	2.41	3.12	1.89	4.97	3.55	3.45	2.41	5.58	7.29
CaO left ...	64.31	76.57	73.28	63.61	73.49	73.68	65.23	69.87	29.01

TABLE VI.

	1	2	3	4	5	6	7	8	9
	Per cent.								
(1) CaCO <sub>3</sub> ...	67.07	79.48	75.25	68.58	77.64	77.33	67.70	74.88	39.52
(2) CaCO <sub>3</sub> , CaSO <sub>4</sub> ...	66.65	79.18	74.98	67.92	77.31	76.99	67.44	74.47	38.97
(3) CaCO <sub>3</sub> , CaSO <sub>4</sub> , 3CaO.Al <sub>2</sub> O <sub>3</sub> ...	66.21	79.03	74.77	67.53	76.29	76.40	67.13	74.27	34.76
(4) do. as (3) with 2CaO.3SiO <sub>2</sub> ...	65.90	78.62	74.53	66.88	75.83	75.95	66.82	73.54	33.81
(5) do. as (3) with CaO.SiO <sub>2</sub> ...	65.74	78.42	74.40	66.55	75.59	75.72	66.66	73.17	33.33
(6) do. as (3) with 4CaO.3SiO <sub>2</sub> ...	65.58	78.21	74.28	66.23	75.36	75.50	66.50	72.81	32.85
(7) do. as (3) with 2CaO.SiO <sub>2</sub> ...	65.26	77.80	74.03	65.57	74.89	75.04	66.18	72.07	31.89
(8) do. as (3) with 4CaO.SiO <sub>2</sub> ...	64.31	76.57	73.28	63.61	73.49	73.68	65.23	69.87	29.01
(9) Acid ...	63.42	75.74	71.47	63.63	75.04	74.76	64.68	70.70	not applicable
(10) Sugar ...	63.00	75.50	71.05	63.35	74.72	73.85	64.40	70.35	30.80

fore the slight solubility of magnesia in lime sucrate solutions would naturally tend to slightly increase the apparent percentage of caustic lime. It is now evident from the agreement between the sugar method and the acid method, that the gravimetric process even with all these calculations cannot be considered to be particularly correct, the results from the above being erratic through the accumulation of experimental errors; whereas in comparing the acid and the sugar methods, the latter is too low in a very slight degree only, which can possibly be accounted for by the acid method dissolving small amounts of magnesium oxide.

I am therefore of the opinion that the sugar process is not only an excellent and quick technical method, but is also one yielding results possibly as accurate as can be got in our present state of knowledge even in the presence of much magnesium oxide, which is in reality not an error, because slightly higher results should be more than borne out in practice, owing to the slight protective action of magnesium oxide on cyanide.

In addition to the above I should also like to state the results of a few experiments carried out on one of the samples (No. 4) to see the effect of time of contact with the solvent. One gramme was transferred to a flask and made up to 1 litre with distilled water, shaken according to periods stated below, an aliquot part taken and titrated by  $N.HCl$  with phenolphthalein as indicator.

After shaking 15 minutes	60.2 %	CaO was found
" " 30 "	61.6 %	" "
" " 60 "	63.0 %	" "
" " 10 "	allowed to stand overnight,	and shaken again for a minute or so,
	62.3 %	CaO was found.

I do not lay any special stress on these experiments, but I simply wish to state the data obtained. Others may possibly be desirous of further experimenting. There seems no definite reason why, if practically  $CO_2$  - free water be used, the best results should not be obtained by an aqueous extraction.

Before concluding, I wish to draw your attention to the so called "blue limes" and their general composition. My analytical experience has shown me that, as a usual thing, they are highly dolomitic limes, and sometimes even approach nearly to burnt dolomites. In addition they contain in practically all cases very appreciable quantities of manganese oxides.

A certain theory has, therefore, suggested itself to me about their utility. Those of you who have analysed the white limes as supplied to the mines, will have noted that their magnesia contents are quite small, not exceeding 2 per cent., and that they rarely contain manganese;

if any, it will not be above traces or 0.1 per cent. Now it is most probable that these white limes deteriorate or revert at a very rapid rate to  $CaCO_3$ , whereas a so called "blue lime" would possibly revert very much slower, say, at a very small ratio to the amount of caustic lime present. Now under the present conditions under which these white limes are received, stored, and subsequently used, this may be of some practical importance. I am inclined to think that their large magnesia contents act as a partial protection and delay this reversion, the result being that, although a white lime may be up to the standard when it was tested, yet when ready to be used, it will not be of a much better quality in CaO content than that of a blue lime stored for a longer period. It, therefore, seems to me that under present conditions where white limes are exposed to the air, having merely a galvanized iron shed to protect them (which in my opinion is of little avail, since the whole affair has really a very free access to the atmosphere), that a blue lime which is much cheaper will do the same amount of work required as a white lime. The magnesia must not be forgotten, for although it is so insoluble—1 in 55,000 parts of water at  $20^\circ C.$  as compared with lime 1 in 800 parts of water at  $20^\circ C.$ , *i.e.*, about 7 times as insoluble—yet it must play some part in consequence of the large volumes of liquid used. In addition we have also the manganese oxide contents, and these are possibly present in sufficient amount to cause an increase of extraction, without affecting the working conditions. It will therefore be of very great interest to hear the practical experience of those who may have used blue limes at any specific time. Also, perhaps some of our members might carry out some experiments, comparing the deterioration of white limes and blue limes under similar conditions, simply doing the assays by the sugar method.

There is just one other point, and that is, for those who try to purify their boiler-feed waters, never to employ these blue limes, but only the best white limes with as little magnesia present as possible. I mention this as I have been told of such blue limes being used, and thus making the purified waters in a worse condition than they were originally.

This paper is only a summary of various results that I have obtained, but I hope it will help in trying to solve this problem of determining caustic lime, and that not only through vigorous discussion but through others taking up the trend and experimenting, we shall be able to go further ahead.

**The President:** I have very much pleasure in moving a vote of thanks to Mr. Croghan for

his paper. It has raised many interesting points and I hope we shall have a free discussion upon it. Lime has certainly become cheaper during the last ten years, but, nevertheless, during the twelve months ending June last year, the Transvaal mines spent about £64,000 on lime alone, so that it is not without an incidence on our working costs, and its use and testing are of interest.

**Mr. McArthur Johnston:** In rising to second the vote of thanks proposed by our President, I feel that we ought to congratulate Mr. Croghan on again raising this question, as the subject is a very important one to our industry. In buying lime we ought to rate it on the basis of caustic lime. About two years ago Mr. G. W. Williams brought this matter before us and practically proved that the sugar method was not only useful and quick but reliable. I was surprised to learn that Mr. Croghan only adopted the process some short time ago, as consideration easily shows that the gravimetric method is bound to give higher results than are available by solution of the caustic lime, which is, of course, the factor on which we buy.

That he tackled the question, however, in a very thorough manner his paper to-night proves. If I criticise Mr. Croghan's paper at all, I may say that he has failed to prove the presence in the lime of silicates and aluminates of lime. It is also to be welcomed in as much as it will ventilate a matter concerning which the lime burners are naturally very sore. To state the position as clearly as possible, we, that is, the consumers, desire to buy the lime soluble in water, and the higher that is, the better we are pleased, for two reasons, firstly reliability of quality saves the cyanide manager some little trouble, owing to the alkalinity being a constant factor; and secondly, the other ingredients are so much waste material to be handled.

Mr. Croghan has failed to elucidate, however, what I consider a very important point, that is, the sampling of the lime. His contention about its deterioration from being exposed to the air in sheds is in my opinion very little based on facts. It may lead to a certain loss, but this I contend would be only superficial in as much as the dusting which takes place in storing the sacks covers the sack itself and prevents almost entirely infiltration of air. Rather would I put before you the best method of grinding, say, ten bags of lime, the quantity mentioned by Mr. Dowling in a previous note, from each truck. This ought to be crushed in a small room with little access of air, so that the floating particles absorb the carbonic acid and moisture from the atmosphere, and so preserve as much as possible the sample to be assayed. This latter should be enclosed in an

air-tight jar on the spot, and forwarded to the analyst or assayer. The sample is then thoroughly mixed in the jar, a small quantity taken out and crushed in porcelain with a pestle and mortar till no grits are felt, and then 2 gm. are put into a litre of sugared distilled water. Our custom is to shake this for the best part of an hour that day, allow it to stand over night, and next morning shake for another 10 to 15 minutes. After settling an aliquot part is measured off, filtered and titrated, the first portion filtered through being used to wash out the funnel, receptacle and burette, but not, of course, the titrating vessel. We usually add about .5 to 1 per cent. to allow for any little absorption of moisture and carbonic acid. In again thanking Mr. Croghan I would express the hope that further discussion will result from the facts and theories he has placed before us.

#### NOTES ON FEEDERS, WITH A DESCRIPTION OF A NEW DRIVING DEVICE.

By D. J. PEPLER (Member).

A high stamp duty is a great factor in the reduction of milling costs, and is moreover responsible for the reduction of costs in all the departments connected with a mine, since costs are usually tabulated on the basis of tonnage crushed. It is essential therefore to obtain as high a stamp duty in the mill as possible, in order to effect a reduction in cost per ton treated. Take for instance a battery of 100 stamps, crushing, say, 600 tons per day with a 400 screen. If the total duty could be brought up to, say, 650 tons per day, the cost of crushing the extra 50 tons would be practically nothing to speak of, as this extra tonnage requires no more labour, power, fuel, etc., than are required to crush 600 tons; the only thing that exception can be taken to being the extra wear and tear on the shoes and dies. In working up the stamp duty from 6 to 6½ tons, it is necessary to see that the stamps are kept plumb and centred, that the dies are placed absolutely dead centre under the stamps, and the mortar boxes re-lined when required, so as to prevent unnecessary space round the dies. It is necessary to regulate every stamp, so that it has its proper height of drop, to be sure that the belts do not slip and that the engine runs at a regular speed. It is highly necessary also that the rock be crushed to a reasonable size in the crusher-station. But the greatest factor in high efficiency in the mill is, keeping the feed low and regular so as to enable the stamps to crush every

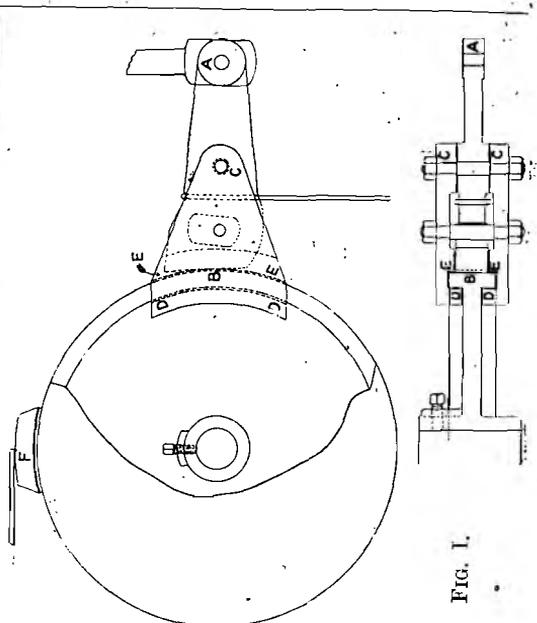
thing on the face of the die, for if any accumulation of rock takes place in the mortar boxes, the stamps will be making slimes instead of performing their crushing duty. There is no doubt that judicious feeding plays an important part in the attainment of a high stamp duty, and it is essential that a reliable feeder should be used. To the best of my knowledge, the Challenge feeder is the most suitable known, but the driving device is defective, being cumbersome and too complicated in connection with the suspension gear.

To explain what I mean, the device consists of one disc, one brake wheel, three springs, three pawls, and three shoes with their necessary studs. If one of the three springs happens to break, which occurs frequently, the pawl and shoes fall out, in consequence of which the feeder stops feeding. Again, the pawl and socket in the brake wheel wear pretty quickly, and when the wear reaches  $\frac{3}{16}$  in., the pawl and shoe will pass the centre line and fall out at the opposite side. This also causes the feeder to stop. I have had cases where, through the pawl wearing a little short, it has slipped half through on to the central position where it actually jams the disc, causing the feeder to stop feeding.

The above-mentioned defects to the suspension driving gear retard crushing, and are also the cause of 40 per cent. more stems breaking than would be the case if a more reliable driving device were attached to the Challenge feeder. It would not only reduce the breaking of stems by 40 per cent., but would also obviate the breaking of cams and cam shafts to a great extent, and the loosening of heads and shoes. It follows that a considerable saving of time and expense will be effected by a reduction in the above-mentioned breakage and causes of stoppage. When a reliable driving device is attached to the Challenge feeder, by which the feed can be regulated to the exact requirement of the stamps, and which can be depended upon as absolutely reliable, the duty of the mill is increased by fully 8 per cent. Such a reliable device exists, and is called "Pepler's Operative Device to Challenge Feeder."

A few points as to how the grip is produced will, no doubt, be useful and can be followed from the accompanying sketch (Fig. I).

The feeding stamp striking the rock shaft lifts the lever at A, the lever pressing against wheel at point B on centre line lifts side cheeks at point C, thus causing a twist grip on the wheel flanges at D and E, bringing the wheel forward. After relief of blow on the rock shaft the combination drops back to its original position ready for the next tap. F represents brake block checking wheel from turning backwards.



The combination will move the wheel, which is connected to the pinion shaft driving the feed tray, from  $\frac{1}{8}$  in. to any distance that may be necessary, in proportion to the blow delivered by the feeding stamp on the rock shaft. This movement is adjustable, and therefore no more rock is fed into the mortar than the stamps require. Provided that the feed is properly adjusted, the mortar boxes will not run empty and cause unnecessary breakages, the feeder being reliable, feeding the stamps as determined by the adjuster.

I put the first one in on trial, at the Ferreira Deep battery, twenty months ago, and during that time no repairs have been made to the device, it having operated without failure up to the present day. It is in as good working order to-day as when first installed. The device is so sensitive and can be adjusted so closely that the feeder works with each blow of the stamps. I have this device on three-quarters of the feeders in the battery, and the stamp duty has increased considerably in consequence. There has also been a marked reduction in the breakage of stems. A log was kept for comparative purposes when the first device was placed on trial, and was kept for a period of six months to show the difference in stem breakages, if any. I found that 50 per cent. less stems broke in the battery fitted with the new device than in the battery next to it not so fitted, and driven by the ordinary suspension gear. I compute, that over the whole mill the breakages were 40 per cent. less through the introduction of the new device.

It is, I think, proof positive therefore, that a reliable feeder is of great advantage, not only on

account of the lesser breakage, but through the higher stamp duty obtained by regular feeding. If any member wishes to see the device at work in order to satisfy himself, he is quite welcome to visit the Ferreira Deep battery, and also the Langlaagte Deep or the Village Deep where there are a few operating.

**The President:** This is one of those short but instructive papers of which we ought to have more. It rightly attaches weight to simplicity and reliability. They are both very desirable in connection with feeders. The complex feeder is a first class thing for the manufacturer, but it is undoubtedly a worry to the man in the mill, and, therefore, the simple design that Mr. Pepler has put before us commends itself to our attention. I beg to move a vote of thanks to Mr. Pepler.

**Mr. C. W. Lee:** I beg to second the vote of thanks. At the same time I hope that Mr. Pepler's paper will arouse some of the battery managers, who have been buried in the grave of lethargy for some considerable time. Mr. Pepler's paper has struck a note that will continue beyond this meeting. There are similar devices to that described, which have been in use on the Rand for many years, but owing to the reticence of battery managers they have never been brought to the front. The paper will, I am sure, provoke quite a healthy discussion.

#### THE SCREEN ASSAY ON THE MEYER AND CHARLTON G. M. UNDER "THE NEW METALLURGY."

(Read at March Meeting, 1907.)

By CHRIS. TOOMBS (Associate).

#### REPLY TO DISCUSSION.

**Mr. C. Toombs:** Since my paper was read, crushing with cyanide solution in the mill has been dropped at the sister plant on the New Goch G. M. Co., and work under ordinary Rand conditions, except for the filter pressing of slimes, resumed, so perhaps a few words about the working of the two plants may be opportune.

On the Meyer and Charlton mine from a few months after the inauguration of the system, more gold was invariably produced than called for by the screen assay, and the surplus was not a matter of a few ounces. This is what led me to experiment with the screen samples, of which this new method for determining the screen assay was the outcome.

On the other hand, from the inauguration of the "New Metallurgy" on the Goch, a serious monthly deficit was almost invariably the rule, and so serious did this become, that after exhaustive experimental work and checking the tonnage, the conclusion was come to that the gold was being stolen, and the whole mine staff fell under suspicion, and two complete staff changes were made inside a very short period. This can hardly be wondered at; here are two mines supposed to be working under exactly similar conditions, one has a monthly surplus of some hundred ounces, and the other a monthly deficit of several hundred. I agree with Mr. Wisdom, that the failure to account for the deficit may have been caused by the "personal factor." With this system very great care throughout the whole plant is required, or very serious losses may occur. On a small compact plant like that of the Charlton, greater care and personal supervision is more possible than on a plant like the Goch, and a striking instance of the "personal factor" is seen in the query put by Mr. Chapman—"What did we do, when the 'hose solution' was higher than the 'tub solution' in gold value?" I never experienced this, but that it was due to careless sampling I have no doubt. What I was struck with on the Meyer and Charlton was the interest taken by the men both in the mill and cyanide plant in their work, and the precautions taken, to prevent any loss of the gold bearing solution, and probably one of the causes of the deficit on, the Goch was caused by constant loss of this solution.

Coming to the discussion on the paper:—Mr. Acheson Jones' contribution in the shape of a gold return for the two months, February and March, interested me most, but it would have been a much more finished and generally interesting contribution if he had also given the gold return for two months under the original method of assaying, say, for November and December, 1906, the last two months that that method was in use. I know that at that time Mr. Jones could not so cheerfully say that "the method of determining the screen assay at the Meyer and Charlton gives a result as close to the true value as the system in vogue with the ordinary water process." Certainly to account for the gold to within 0.8 per cent. is extremely good work, and I must congratulate the manager and various heads of departments on the results for these two months.

Messrs. Chapman and Wisdom are both inclined to cavil at my calling the improved method a new one; Mr. Chapman says it was the first method used by him on the Goch, and Mr. Wisdom, who should speak with authority, says it was tried there, because the other method

was a failure. I wonder which of these two statements is correct? That assaying the unwashed pulp was tried I quite believe. The method was as follows: Assay pulp unwashed, take another weighed portion and wash with several weighed quantities of water, assaying each to find out the amount of gold washed out, and so by comparing with the "tub solution" assay find out the amount of gold dissolved in drying. This method introduced at least two more assays, and I do not wonder that it was dropped, as Mr. Wisdom says, because it was too troublesome. I distinctly claimed that my method was less troublesome than the old, and that it was quite possible to have the return the same day, and I also claim that my method of calculation is original.

Mr. Chapman remarks that there is possibly a loss of gold caused by drying the sample in an iron dish. I do not think there is, as finding one of these dishes scaling, and fearing that the scale might foul the sample, I scraped it off and assayed it, but found no gold. He also says that he had no difficulty in getting beads to agree when assaying the washed pulp. My experience was that only on rare occasions did the beads agree, and when some checks were done by two independent consulting assayers, there was a variation of about 1.5 dwt. between their highest and lowest results on each of six washed pulp samples. As a point of interest I might mention that the average of all six samples returned by one of these assayers agreed very closely with mine.

I agree with Mr. Whitby in what he says about screen samples not being taken often enough, but I do not see how the difficulty he complains of, viz., "unwieldy samples," is to be overcome. I do not know how the sampling is done on the Robinson Deep, but I should say it must be extremely accurate. The 24 hours' sample there for 300 stamps, according to Mr. Whitby, is only 50 lb., sampling being done once an hour, or it is roughly about the same as for the 70 stamps of the Meyer and Charlton, where sampling is only done three times per shift. On working out these figures, I find that the weight of sample per mortar box of 5 stamps per time taken on the Charlton is 6 oz., which does not seem excessive, while the Robinson Deep sample is only 1/2 oz. I think that members will agree that this is the "smallest quantity compatible."

Mr. White asks, "if the sample is properly taken?" I can assure him that it is, and that the low ratio is not due to loss of solution; and I should say that the loss, if any, would require to be very appreciable, and would be very apparent to the most careless millman, to increase the ratio from 6.5 : 1, as it is approximately at pre-

sent, to 7.5 : 1, and that would still be a very low ratio of solution to pulp.

Mr. Laschinger's formula is certainly much more concise than the array of figures given by me for calculating the screen assay value in the new method. I tried to arrive at one, but it was complicated, and it did not strike me to simplify it in the neat manner done by Mr. Laschinger.

Mr. Sharwood's contribution is interesting, and he seemed to be on the way to obtain results, when he stopped experimenting. He seems to lay the blame on magnetic iron. I have recently in the course of my work done a good deal of concentrating for pyrites, and I have gone through them with a magnet and separated, and weighed any metallic iron. In no case was it higher than 0.0005 per cent. on ordinary sand samples, and it certainly was not magnetic. I expect the samples he speaks of had been ground on the bucking board, and, of course, this would add to the percentage given above, but I do not see why it should be magnetic, and in any case the percentage would be so small, that on the ordinary fluxing balance it would not be appreciable.

I will now give the results of some experiments which I think show where the trouble lies. A large sample of pulp dried down with gold bearing cyanide solution, was well mixed and divided into two equal parts, one part being well-water washed and re-dried. Assays of both were made, and a grading with assays of the various grades done. The figures are:—

	UNWASHED.				WASHED.			
	Beads		Av'r'ge		Beads		Av'r'ge	
Original	9.1	8.6	8.7	8.80	7.5	8.2	8.2	7.97
+60	6.8	5.1	6.0	5.97	5.1	5.6	5.5	5.40
-60+100	10.9	8.8	9.2	9.63	9.0	8.7	8.0	8.56
-100+150	12.9	12.4	12.8	12.70	12.0	11.4	10.6	11.33
-150+200	13.2	12.6		12.90	12.8	11.6		12.20
-200	9.6	9.8	9.6	9.70	7.2	7.2	7.2	7.20

It will be seen that there are serious differences in the bead weights both washed and unwashed, except in the -200 portion, in which there is a big drop in gold value on washing, and I should say from the above figures that the very fine gold, which in ordinary samples probably helps to average the coarser particles when assaying, is dissolved, leaving the coarser particles locked up in the coarser grades, and it is not possible to get these evenly distributed throughout the ample taken for assay.

**The President:** We are indebted to Mr. Toombs for his paper and the very satisfactory discussion it provoked. One might summarise the position by saying that the continuous slimes process has had a set back on the Goch and that its running on the Van Ryn and Meyer and Charlton is being watched with attention.

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## MINE SUBSIDENCE.

(Read at March Meeting, 1907.)

By ALEX. RICHARDSON, A.I.M.M. (Member).

## REPLY TO DISCUSSION.

**Mr. A. Richardson:** The task of replying to the discussion on my paper would have been rendered easier had those who took part in it treated my views in a less complimentary and more stimulating manner. It is, however, gratifying to know that my conclusions have in the main proved acceptable and have been the means of inducing others to bring forward some very instructive matter.

It is interesting to learn that, as Mr. Innes said, our mines owe their coolness to the immunity of the Witwatersrand from any earthquake of recent age, and to this cause might be added the unoxidisable nature of our rock, which, no doubt, greatly assists.

Our freedom from earthquakes is a reassuring circumstance, because mining operations induce a position of unstable equilibrium which must give way to a stable one sooner or later, and earth tremors would hasten the change.

In this connection the following list of earth tremors, notified to our Observatory in recent dates will be of interest:—

Date.	Locality.	Duration in Seconds.	Direction.	Effect.
May 5, 1906	Warmbaths	10	N to S	Shook house
May 20	Warmbaths	20	N to S	Shook house
June 2	Warmbaths	—	—	—
Aug. 27	Alicedale	—	N to S	Shook doors
Dec. 10	Amsterdam	4	Various	—
	Alicedale			
	Holyrood			
	Amsterdam			
	Piet Retief			
	Swazieland			
	Zululand			
	Natal			
March 3, '07	Leydsdorp	6	S to N	Shook house
June 6.	Warmbaths	6	—	—
June 15	Warmbaths	—	—	—
June 22	Kasempa	—	ENE to WSW	Shook house
	Rhodesia			

A Milne bracket seismograph properly installed would cost at the outside £80, and considering the importance attached to the instrument in other countries and colonies, Kimberley alone possessing three, the Transvaal might well fall into line, even though seismographic records have no direct bearing on mining. The damp factor, to which Mr. Innes has alluded, appears to depend on the underground water level. This varies greatly in this country and is at present low. It rises with the rainfall, and the accompanying percolation, which latter must be considerable as our spruits and rivers only carry from 2 or 3 per cent., leaving evaporation and absorption to account for the remainder: it is probable that three good wet seasons would make the level as high as it could go. The maximum amount of water in our mines seems to be present at about the beginning of March, and it is about this time, therefore, that the disintegrating effect of moisture should be most noticeable.

Prof. Yates' contribution, with Prof. McNair's address, is a valuable addition to our knowledge of the subject, and they both agree that with deep mines the increased difficulties of roof support will call for drastic changes in our present methods. The subject promises to become increasingly important as the developments in large mines everywhere increase in extent. In the United States extensive caving and crushing of pillars, in some cases reaching over large areas of workings, has taken place during the past year in one or two of the Lake Superior copper mines. One mine, the Atlantic, was forced practically to shut down and for some months has been in process of re-opening. The very deep Quincy mine, also in the same district, has had disturbances of the hanging wall which have called for careful investigation.

Prof. Yates asks the question, "is it worth while keeping the workings of our outcrop mines open in view of the expense that such a procedure would entail?" I am of opinion that it is. A subsidence is bound to cause grave inconvenience and probably also great expense; it will bury numbers of valuable pillars which, owing to their scattered positions, it would never afterwards pay to recover; it will close up hundreds of feet of development which will have to be re-opened in order to make low grade and unworked blocks accessible. It often pays handsomely to re-open an old level for the sake of the pillars along it and the vamping under the rails. How much better would the return be if this were done while the ground was still standing? Another feature is that it sometimes predisposes portions of the workings in the vicinity to an attack of the same malady through the shaking up of the strata and the consequent increased

facilities for percolation. It might be possible, starting at the boundary, to work back along a level taking out everything in the stope above and allowing it to come in. It would be dangerous work because of the strong nature of the hanging which, if it held up over large areas, would cause the resulting crash to be tremendous. The gust of air alone from such a fall would be comparable to an explosive wave.

Estimates as to the depth to which mining can attain are as various as the countries from which they emanate. Prof. Park says:—Mining at great depths will be limited by the underground temperature and the economic conditions. By the adoption of secondary ventilation and methods of refrigeration mining could doubtless be carried on at depths ranging up to 6,500 ft., according to the local increment of temperature.

Mr. A. C. Lane, in an able article in the *Mineral Industry*, gives 10,000 ft. as the general limit, with a possibility of reaching 15,000 ft. under very exceptional circumstances.

If we turn to coal mining we find that the last two Royal Coal Commissions of England have fixed 4,000 ft. as the limit at which coal can be profitably worked.

Geologists have also a word to say in the matter. Spurr states that openings cannot exist beyond depths variously estimated at from 16,000 to 33,000 ft., and Kemp writes as follows:—The maximum depths at which cavities can exist varies from 1,625 ft. for soft shales to 32,500 ft. for firm granites. The maximum depths introduce the zone of flowage, wherein the load is so excessive that the yielding to deformation comes in the way of a viscous flow, or plastic yielding, evidences of which are visible in many gneisses. That such a zone exists will appear to anyone who reflects upon the necessary behaviour and yielding of rocks, which are confined on all sides and yet are compressed beyond their limits of resistance.

The hydraulic filling of workings with culm, sand, sweepings, ashes, and other similar material, to which Mr. Lane Carter refers, is largely practised in the coal mines of Silesia, Westphalia, Hungary, France and America, and a remarkable diminution in the number of accidents from falls of rock as well as other benefits has attended its use.

A great advantage of hydraulic filling is that it fills every inch of the space, whereas careful packing by hand will only fill up 60 per cent. The material packs solid in a very short time and does not run if it is fine; it has been piped underground horizontally for a distance of 3,000 ft. under a head of 656 ft. The water consumption is somewhere about 1½ lb. of water per 1 lb. of culm flushed. Tailings from the cyanide works

are being used in Australia; the practice had to be discontinued for a time owing to the noxious fumes overcoming the men, but the installation of a special washing plant overcame the difficulty. Any system of hydraulic filling has not, as far as I am aware, ever been tried in either our coal or gold mines, but it deserves attention in view of its success elsewhere, although the cost of installing and operating would prohibit its use in our deep levels.

Mr. Lane Carter remarks that it is an utter impossibility to keep up a large area of ground when on the move; this is true generally because collapse follows on so very rapidly after the unmistakable signs of subsidence that the danger of men being buried up or imprisoned becomes too great to permit of their entering the affected zone. But there are times when timber stacks filled with waste can be rushed in along the levels and much of the development saved from destruction. Timber alone will not delay the collapse by a single hour, and is only useful where the hanging is bad in small patches and where there is no weight coming on from the strata above. The practice of leaving stope pillars over the hanging wall of an incline shaft sunk in the foot was not a suggestion of mine and meets with my emphatic disapproval, as does also the suicidal practice of stopping under the footwall which has been frequently indulged in in the past.

Facts certainly point to the pillars, as laid down by the present regulation, being in a wrong position; it must be noticed, however, that there were two exceptions to the law of the angle of fracture, and that any change would have an economic, even a legal, bearing. It would be manifestly unfair to deprive an outcrop property of ground which has been taken into its ore reserves in order that some object on the surface of the adjoining deep level, for which ground had already been allocated, should receive support. We may be justified in saying that the present regulation is wrong, but we are scarcely in a position yet to say definitely which other would be right.

To make our mines secure so that posterity may derive benefit from what we leave, as Mr. Wager Bradford suggests, is, I venture to say, a policy that is carried out conscientiously in no mining camp in the world, however self-sacrificing its inhabitants may be in other ways, and however much they may applaud such an admirable sentiment. And when we consider the continual fight which miners wage against natural forces and the uncertainty of how long that struggle can be carried on, it is little to be wondered at that most people prefer to leave for posterity certain sums of money in the bank, if they can, rather than doubtful blocks of ground in the mine.

Mr. Reid Bell's clear description of the settlement of subsided ground, in conjunction with that of Prof. Yates, explains fully what may be expected to occur in any class of formation. He is wrong, however, in the deduction which he drew from my remarks on the Geldenhuis Deep subsidence, as this from its sudden nature was a very likely one to have affected the surface, and not having done so it is regarded by me as one of the strongest arguments in favour of the 1,000 ft. horizon of no surface effects. His remarks on longwall working, of which he has intimate knowledge, are so interesting that I hope he will see his way to extend them for our benefit.

The necessity of a careful examination of wet hanging walls is emphasised in Mr. Whitton's remarks. Now this examination is an excellent safeguard, but in many cases quite ineffective. We have no means yet of testing the solidity of thick slabs of hanging and of thus protecting ourselves against accidents, such as the one which occurred on the New Heriot, where a solid mass, 5 ft. thick and weighing about 400 tons, came away without any warning.

The usual way of testing hanging is by careful examination and by sounding it with a 4 lb. hammer. Now with a 4 lb. slab 3 ft. thick will not, when struck, emit the hollow sound which indicates looseness, even a 20 lb. hammer will fail to elicit any information if the slab is over 4 ft. thick; a 5 ft. slab cannot be tested at all. It occurred to me that an instrument might be devised which would assist the ear in distinguishing with greater nicety the quality of the sound emitted by different thicknesses of material. Accordingly I made a number of tests with an instrument, a special form of stethoscope, familiar to medical men under the name of phonendoscope. The results have been nugatory so far, and the matter is merely mentioned to draw the attention of those of an inventive turn to a direction where their energies might be well and humanely employed.

The serious problem of falls of rock is solved very considerably by resueing, and from this and other aspects I wish to amplify my original remarks.

It must be borne in mind, that resueing far from being a new thing, is an ancient mining method and that it has been tried in several of our mines and found unsuitable. It cannot be practised where the reef is broken up into stringers, and it has no utility where the reef is wide, its application is also rendered very difficult when bedding planes are not present. A narrow reef with well defined partings is the ideal thing. Its use necessitates a large number of working faces, as it is obvious, that in the alternate breaking of waste and reef the quantity mined of the latter will not be so great as in ordinary working. Furthermore,

broken ore cannot be allowed to accumulate in the stopes; this will render underground conditions inelastic. The improvements in ventilation, the lessened danger of accidents from falls of rock, and the increased possibility of making narrow poor reefs pay, are factors which entitle it to respectful consideration.

Mr. S. Richards, the acting manager of the Jumpers G. M. Co., has kindly given me some figures illustrating the two systems, and I have picked out at random a few returns showing the work done by tributors on that mine.

Reef.	Reef Width. Inches.	Milling Width. Inches.	Boys per Shift.	Tons Hoisted.
MRL	6	12	9	50
SR	8	16	20	137
SR	6	12	5	46
MRL	4	8	9	62
MRL	4	12	6	60
MRL	5	10	10	63
MRL	12	15	14	105
MRL	8	16	8	75

In the comparative statement it will be noticed that tributors are charged 10s. per ton on all rock sent out by them. This and the system of paying by the dwt. per ton is an effective check against sending up waste and against losing reef, for it is obviously to the tributors' advantage to send up clean reef and fines so as to get the greatest result, and the obvious rarely requires much pressing when there is money in it.

#### EXAMPLE OF TRIBUTORS' CONTRACT.

Stope width 30 in.	(7½ tons per fathom)	
Stope value 4.0 dwt. per ton		
Reef width 3 in.	(15 cwt. reef per fathom)	
Reef value 40 dwt. per ton		
Assuming tributor saves 6 in., this would equal 20 dwt. per ton, 1½ tons ore per fathom value 20 dwt. per ton. Value per fathom ...		£6 5 0
1½ tons ore @ 20 dwt. @ 2s. 9d. per dwt. = 55s. per ton		
Less 10s. per ton reduction charges		
1½ tons @ 45s. per ton per fathom ...	£3 7 6	
Underground Costs—		
1½ tons shovelling, tramming, etc., @ 3s. ...	4 6	
1½ tons reduction charges, etc., @ 10s. 1d. ...	15 1½	
1½ tons residues to dump @ 3s. ...	4 6	
	£4 11 7½	
Profit per fathom ...	1 13 4½	
	£6 5 0	£6 5 0
Profit per ton milled, 22s. 3d.		

STOPING BY CONTRACT @ 60s. PER FATHOM.

Taking the same as above.

Value per fathom	...	£6. 5 0	
To contractor per fathom	£3 0 0		
7½ tons ore per fathom			
Costs, shovelling, tramming			
per ton 3s.	... 1 2 6		
7½ tons less 20 per cent.			
sorted at mill			
6 tons reduction charge			
10s. 1d.	... 3 0 6		
6 tons residues to dump 3s.	18 0		
		£8 0 0	
Loss per fathom	... 1 15 0		
		£6 5 0	£6 5 0

Loss per ton milled, 5s. 10d.

Mr. Weldon questions the figures given for the crushing strength of pillars and quotes Eytelwein & Euler in support of his argument. Eytelwein's formula, which has been superseded by Euler's, when reduced to its simplest form is to the effect that the strength of rectangular columns varies directly as the longest side multiplied by the shortest side cubed and inversely as the height squared. Similarly Euler's formula is that the strength of a column varies directly as the area squared and inversely as the height squared. These two formulæ are identical when the columns are square. Now Euler's formula can only be used for long pillars, that it is for those that fail by bending. In the case of square pillars of rock Euler's formula cannot be used unless the height of the pillar is 27 times the length of the side. This would mean, taking the ordinary mine pillar of 10 ft. square, that this formula will give quite erroneous results unless the pillar is 270 ft. high.

For shorter pillars, that is those whose height is from about 10 times the side upwards, there are numerous formulæ in use, such as those of Gordon, Grashof, Hodgkinson, Navier, Tetmajer, etc., but these introduce considerations of bending as well as of crushing and cannot therefore be applied. Gordon's, the one in most general use, gives a result which never exceeds the crushing strength of the material. As our mine pillars are neither columns nor pillars, in the strict engineering sense, but short blocks, the only formula applicable is that furnished by the researches of Bauschinger, the eminent German authority on the testing of materials.

There is a general law governing the strength of blocks, such as are used for testing. For

test blocks of geometrically similar form the crushing strength varies as the area of the surface on which the crushing pressure acts, or, generally, the strength of geometrically similar test pieces varies as the square of homologous sides. Bauschinger's formula, which holds good to heights up to five times the side, is as follows:—

$$f = \left( \frac{4\sqrt{A}}{C} \right)^{\frac{1}{2}} \left( k + c\frac{\sqrt{A}}{h} \right)$$

A = area of section of prism.

C = circumference or perimeter.

h = height of piece.

f = strength per unit section.

k and c = constants for each material.

It gives at once the law that prisms and cylinders of geometrically similar form have the same strength per unit section, also that in prisms of the same height and sectional area the strength per unit section varies inversely as the square root of the circumference.

For square prisms of different heights the equation becomes  $f = k + c\frac{s}{h}$  where s is the side of the square, and h its height.

Bauschinger gives several values for these constants for different materials; the following are for sandstone:—

	k	c
Swiss fine grained sandstone	283.5	316
kg. per sq. cm.		
Fine grained Bunter sandstone	327	107.5
kg. per sq. cm.		

Series of tests with square and rectangular prisms of different heights and side ratios; fine grained sandstone 356.5 97

In a series of tests of square prisms of very fine blue-gray Swiss sandstone, the pressure applied normal to bed, and with ratios of side to height of values varying from .98 to 4.32, the results indicate that the strength increases in the following manner:—

Height = side.	Strength per unit section = 1
Height = ½ side.	" " " = 1.3
Height = ⅓ side.	" " " = 2.5
Height = ¼ side.	" " " = 3.4

The average stope pillar on the Rand has probably a height of from ½ to ⅓ of its side or a mean factor, as above, of about two, giving a calculated crushing resistance of 20,000 lbs. per sq. in. on this basis. Now all rock specimens selected for testing are carefully prepared, are of homogeneous material and without flaws. A mine pillar is rarely of homogeneous material,

it is certainly not free from flaws naturally, and the vibration of dynamite explosions all round it will have the effect of creating others artificially. It is also subjected to the continued influences of moisture, vibration and uneven pressure; when these injurious factors are given their due weight it will be found, that my estimate of a crushing strength of from 10,000 to 15,000 lb. per sq. in., is, in the present state of our knowledge, quite warranted.

Mr. Weldon remarks that roofs rarely fail by shearing in practice. Failure by bending is certainly the common form of collapse, although from theoretical reasoning the contrary would appear to be the case. Bending is enormously affected by flaws in the rock and shearing very little. A slab will fail by shearing where the greatest weight rests on the smallest area that is at the circumference of the pillar. The area of resistance to shearing is the thickness of the slab multiplied by the circumference of the pillar, and it is only when a flaw follows the contour of the latter that the shearing strength of the slab is greatly reduced. The splitting up of a slab by bedding planes, resulting in separate bodies in course of time owing to moisture, exercises a tremendously prejudicial effect on bending and none at all on shearing, and this one reason alone will explain many instances of roof failure.

The figures relating to the size of pillars, their distance apart, etc., are based on experience, and in this they should carry with them a recommendation to Mr. Coombe.

A few notes on the failure of mine pillars may be of interest. The stress on any oblique plane in a prism loaded with a weight  $P$  can be resolved into two components, one a normal component,  $P\cos^2 A$ , and the other a tangential component,  $P\sin A\cos A$  where  $A$  is the angle between the normal to the plane and the axis of the pillar. This tangential component or shearing stress becomes greatest when  $A$  is  $45^\circ$ , and is equal to  $\frac{1}{2}P$ . The normal component  $P\cos^2 A$  influences the angle and intensity of the stress at which the pillar fails. Acting in a direction normal to the plane of shearing it causes a frictional resistance, which restrains the action of the shearing stress. In very short mine pillars the action along any one oblique plane would be greatly restricted, and this may account for an increase in strength. A cube breaks along symmetrical planes similarly situated with regard to its axis, that is, into six similar and equal pyramids with a rectangular base, and a prism similarly; but in the case of the prism the upper and lower pyramids are terminated by a line instead of a point. A cylinder

resolves itself into two cones with the edges split off all round. This action can be often seen in a mine when the pillar is fairly long, pieces flaking off about half way up, the top and bottom remaining solid, until at last it degenerates into a wasp-waisted affair of shattered rock.

Mr. Boright's remark that falls are more prevalent before the month of May supports the supposition that percolation resulting from the rainfall exercises a marked effect. The danger consequent upon the breaking of a footwall in order to obtain waste for packing in steep stopes, to which Mr. Homersham alludes, does not, of course, apply to those of moderate dip. The main point to be considered then would be the thicknesses of hanging and foot, supposing the waste to be required in a stope occupying an intermediate position.

I regret that the scope of my paper will not permit me to discuss fully with Mr. Coombe the relative merits of practice and theory, so will conclude by observing that a wrong idea, even if embodied in a formula, is not such a bad thing, as it may be confuted. It is the want of ideas that is anathema.

**The President:** Mr. Richardson has certainly struck a field which is quite new to us, though I am sure that as years go on we shall hear more and more of it, because we must expect something like a general subsidence right along the reef. He mentions the matter of a seismograph. Mr. Innes, the Director of the Observatory, also touched upon that subject in his contribution to the discussion, and if I remember rightly you listened to his suggestion that the Observatory should have a seismograph very sympathetically. I had no idea that these instruments were so cheap. A matter of £80 is not much, considering the utility of such an instrument, and I would like your views as to whether the Society should move in the matter of seeing that the Observatory is provided with one.

It was agreed that this matter should be left in the hands of the Council.

**The President:** May I make a remark in connection with the Jumpers mine, which Mr. Richardson mentioned? I was round there to-day with some of my students, and I hope that before long the Council will be able to arrange a visit for the members of the Society. Undoubtedly it is the most interesting mine on the Rand. You not only have rescueing there but you have special forms of timbering and supports which you see nowhere else on the Rand.

## NOTES ON SOME RECENT IMPROVEMENTS IN TUBE MILL PRACTICE.

(Read at April Meeting, 1907.)

By KENNETH L. GRAHAM, M.I.M.M. (Member).

### DISCUSSION.

**Mr. J. Kennedy:** In spite of having used banket exclusively as a substitute for imported pebbles, for the past 15 months, I have always been under the impression that better results could be obtained by using pebbles. I fail to see how an irregularly shaped piece of ore can possibly be as effective as a smooth round pebble.

Without presuming too much, I venture to suggest that, in Mr. K. L. Graham's test, the No. 2 mill consuming pebbles was underloaded, as I do not consider 670 lb. sufficient feed for a mill of this size per 24 hours. It is not surprising that Mr. Fraser Alexander got better results with banket in his test. As stated, his trial was of shorter duration, and two tons of Danish pebbles were fed daily. If not an impertinent question, I should like to ask Mr. Alexander, for what period was the feed of two tons per mill per 24 hours continued?

With Mr. Graham's theory of the higher percentage of work being done by impact, rather than by grinding, I entirely disagree, and for this reason:—I had come to the conclusion some time back that, manganese liners had not been given a fair trial before being condemned, as, at the Glen Deep, the life of manganese liners was about 90 days, while at the Robinson Deep, the liners lasted only about 50 days; I, therefore, set to work to ascertain the cause of this marked difference in life, and in endeavouring to approach as nearly as possible the real thing, *i.e.*, manganese lining, I found that by loading a glass jar with small pebbles, which had been discharged from the tube mills, the pebbles reaching just under the neck of the jar, when lying on four rollers, and clean sand and water at 1 to 1 being added to reach just under the centre of the jar, the jar then being made to revolve—the result was just as I had anticipated. Instead of the load being raised sufficiently high to enable the pebbles to roll and tumble over one another, the load, with the exception of a few pebbles slipped, on the side of the jar whilst revolving.

More pebbles were then added to bring the load well up over the centre, and the jar was started again. This time, much to my satisfaction, the pebbles, with pulp clinging to them, began rolling and tumbling over one another down the inclined surface of the load, formed by the rotary motion of the jar.

It is interesting to note how the pebbles and pulp rise at the ascending side of the jar, although, to all appearances, the water is level to about the outflow on the descending side.

Some members might still be of the opinion that this experiment does not prove that the grinding is not done by impact, and for their benefit I should like to explain that, in this experiment, when the jar was under loaded, there was scarcely any sound noticeable while the jar was revolving, but, when the pebble load was increased to the point mentioned, there was the noise of the pebbles striking the liner, which coincides with Mr. S. H. Pearce's experience while the manganese liners were being tried at the Glen Deep, and which was mentioned while Mr. H. A. White's paper on "The Theory of the Tube Mill" was being discussed before this Society about two and a half years ago. Mr. Pearce stated that a good deal of difference was noticed in the sound of the tube mill when this liner was first run, inasmuch as there was an absence of rumbling which had been a feature previously. He also stated that there was a tendency for the pebbles to wear flat, and the crushing efficiency dropped considerably. Later, the rumbling occurred again, and the crushing efficiency increased. He also concluded by remarking, that he supposed that there was considerable slip on the smooth surface when starting, and that later, the liner had acquired a rough surface and raised the pebbles a little higher than the smooth surface had done, which fact causes crushing to be accomplished instead of reduction by attrition. The manganese liners when discarded at the Robinson Deep, were as smooth as glass, and this proved that Mr. Pearce's supposition was incorrect. The sole cause of the absence of rumbling, and the dropping of efficient grinding was, that the mill was under loaded. An important point which appears to have been overlooked, when advocating the theory of crushing by impact, is that, as soon as the rolling or falling pebble touches the water, the pulp clinging to it becomes disengaged, and the bare pebble strikes the liner, and this accounts for the rumbling referred to.

With reference to hydraulic classification, I run seventeen classifiers, five abreast in the first row, and six in the two following, and use water on all, finding it essential to do so when dealing with a big tonnage with a limited tube mill installation.

The longest life of siliceous liners, 4 in. thick, I have experienced is, 108 days, and a lining of local chert, 4½ in. thick, has been running for 102 days, and to all appearances will last a fortnight longer; but it seems that the days of siliceous and

cherts as liners, are numbered, judging by the success attending the lining accidentally invented at the Los Estrellas mine, El Oro, Mexico,\* where the inventor had lined his mill with riffle-like castings in order to raise the pebbles nearly to the top of the mill, thus increasing the height of drop. When the mill was opened after several hours' run, it was found that the pebbles had become wedged into the riffles, and thus formed a lining of themselves. After eight months' run, the wear of the casting was found to be so slight as to indicate a life of two to three years.

#### A FEW NOTES ON THE REFINING OF BASE BULLION.

(Read at May Meeting, 1907.)

By C. W. LEE and W. O. BRUNTON (Members).

##### DISCUSSION.

**The President:** I think Mr. Lee has an apparatus which he would like to show the members.

**Mr. C. W. Lee** exhibited an air pump for generating compressed air. He pointed out that the air from the compressors could not be made available owing to the moisture contained therein, apart from the heavy cost of conveying as most of the smelting works are situated some considerable distance from any air connections, and therefore some local means of generating compressed air cheaply must be adopted.

The apparatus shown was very simple but efficient, consisting merely of a petrol tin reservoir and a motor pump which could be worked by a native.

**Mr. T. Lane Carter:** Since reading the paper I have been struck with the trouble our friends have had with their cyanide bullion. It is apparent that there is a difference in the fineness of gold along the reef in the banket. I have worked out the average fineness from our cyanide works for the last three months, and find it to be about 892 fine. We have had one or two bars as high as 906, and several bars over 900, but the average is 892. That shows how fine the gold is on the West Rand. We have also been on the look out for copper and other metals in the ore, but my experience out west is that we have extremely small quantities of base metals in the banket. We have succeeded in finding traces only of copper, zinc and lead. I have been wondering whether the trouble Messrs. Lee and Brunton have had, is due to the base metals being in the banket in larger quantities in East Central section of the Rand.

**Prof. J. A. Wilkinson:** I should like in the first place to congratulate the authors on having successfully introduced a practical method of refining poor cyanide bullion by means of air, but I was sorry to find on reading their paper that data, such as are required for the study of the scientific aspect of the problem, are entirely wanting. It was from this point of view that my attention was drawn to the paper as I was studying the Bessemer steel process at the time. I would, therefore, ask the authors if they could supply the missing data, so as to make it possible to work out the problem.

**The President:** We will not close this discussion to-night, but keep it open until the next meeting.

#### THE UTILISATION OF WASTE HEAT IN SLIMES SETTLEMENT.

(Read at June Meeting, 1907.)

By A. SALKINSON (Member).

##### DISCUSSION.

**Mr. W. R. Dowling:** Throughout the pages of our *Journal*, and other technical papers and books, are to be found scattered references to heating cyanide solutions, but really no definite information or figures are given so far as I have read.

I do not think there is any doubt that heat does accelerate the settlement of ore slimes. Most practical cyanide workers are satisfied that the settlement is better during the summer months than the winter. Mr. Salkinson is the first on these fields to apply the artificial heating of the solution on the working scale, and to produce results which prove the value of the scheme beyond a doubt. His figures go to show that by increasing the slime solution temperature to about 80° F., the capacity of the plant is increased by about 20 per cent. If heat had to be produced specially for the slime works, there would be room for doubt as to whether it paid, but, as Mr. Salkinson points out, there is much waste heat on the mines which can be used cheaply. I think our mechanical engineers now have a good opportunity of developing the scheme of condensing steam with the cyanide solution, which the cyanide works would, no doubt, lend them, on condition that it was returned so many degrees hotter. What surprises me is, that our mechanical engineer members have not jumped at the idea. In using warm water for milling, as far as the cyanide works are concerned, some decompo-

\* See this *Journal*, p. 60.

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sition of the sulphur compounds of the ore might take place, but this would probably not be more than could be easily overcome. For amalgamation, I think, most mill men will agree, that no harm is done as long as the temperature is constant, and, of course, not too high say, about 80° F. I am anxious to have Mr. Salkinson's scheme tried on the plant in which I am interested, and hope later to report that it has been introduced.

## NOTES ON WAIHI ORE TREATMENT.

(Read at July Meeting, 1907.)

By RALPH STOKES (Member).

### DISCUSSION.

**Mr. W. R. Dowling:** I have read Mr. Stokes' paper with interest, as it is a description of the practice of other countries by one of ourselves. Tube milling in Australia is so different to ours that it is very difficult to make comparisons. The economy of the honeycomb liners (Barry's patent), would probably depend on the existence of a cheap and suitable, local material for filling. If it had to be imported or transported any distance, the only saving, if any, would be the dressing of blocks, against which would have to be set the cost of the iron castings, and the labour and cement for filling. The castings, I take it, wear out with the filling. I think the economy, or otherwise, of this ingenious idea would have to be proved by actual experiment, using some local material, of which there is plenty in the Transvaal. I believe some of our tube milling men are making blocks of worn out liners and small pebbles. The procedure, as I saw it on the Glen Deep, in the hands of Mr. Osborn, the cyanide manager, was to break up the silex and pebbles into small pieces, and fill them with cement into wooden moulds. When set, the blocks were removed from the moulds. The moulds are shaped to produce a block, curved at the back to fit the circle of the mill, and with sufficient batter. The honeycomb liners certainly have the advantage of being built in very quickly, reducing the time of stoppage to even less than the creditable records of some of our local men.

A separate house for the amalgamating plates is a point that recommends itself. The hanging up of the mill, because the gold recovery portion of the works require attention, seems to me wrong in principle. With a plate to each 5 stamps, the stoppage of our batteries for dressing and scraping plates, amounts to about half-an-hour per day, or about 2 per cent. of the possible time.

Mills as constructed on these fields require men capable of running machinery as well as recovering gold by amalgamation, qualifications which are quite opposite. The better arrangement is that described by Mr. Stokes where an engineer attends to the machinery and a metallurgical man to the gold recovery, in a separate building. Finally, as pointed out by Mr. Stokes, the danger of theft, if it does exist, is very much reduced.

With reference to the treatment of slimes, I think that Mr. Stokes' prophesy—that filters will take the place of presses—is true, in localities where the decantation process does not prove better than either.

**Mr. C. E. Deakin** (contributed): I read Mr. Stokes' paper with very great interest, as for some short time I was employed on the Waihi Company's property. The first part of the paper is confined to methods of treatment which have been introduced since I left there, and it is more particularly as regards the milling practice that I venture these few remarks.

In dealing with the milling arrangements of the three mills belonging to the Waihi Company it must be borne in mind that the two principal mills, the Waihi (90 stamps) and the Victoria mill at Waikino (200 stamps), the latter situated some 3 or 4 miles further down the Ohinemuri River, were originally installed and equipped for dry-crushing with stamps, the ore having first been dried in big kilns cut out of the hill side. It was only when the great advantages of wet-crushing could no longer be argued against, and the quantity of sulphide ore, which necessitated concentration, had increased to an extent beyond the capacity of the small Union mill (40 stamps) designed to cope with it, that the company converted their mills to wet-crushing plants.

Mr. Stokes states in his paper that the practice of keeping the amalgamation entirely distinct from the crushing department is peculiar to the Waihi Company, at least in Australasia. Perhaps it may be questioned if the Waihi people would have adopted that system, had not as I have before said, the two principal mills been built for dry-crushing, and consequently plate room was not taken into account; while in the Union mill the neighbourhood of the stamps was so dark that careful amalgamation would have been impossible except through the aid of artificial light. The advantages of the arrangement are many, for example, the entire absence of grease, the full efficiency of the plates and the doing away with the necessity of hanging up any stamps whilst the plates are being examined and periodically dressed and cleaned up.

While not maintaining that the Rand companies should alter their plants already built,

though a good case could be made out even for that, I think it can be proved that it would be economical for future mills to be built on that principle. For purpose of calculation I have taken a 200 stamp mill, expecting few, if any, of the deep deeps will be worked on a smaller basis. A conservative estimate of half an hour stoppage per plate per day, would represent on the stamping capacity above mentioned and taking a 5-ton stamp duty, a clear gain of 20.8 tons crushed per day, which allowing for an extraction by amalgamation of 4 dwt. per ton would equal 4.16 oz. of fine gold, value £16 13s. 4d. It would also be the means of slightly increasing the white labour on these fields as the extra labour required should only absorb £6, taking three shifts consisting of one amalgamator at 25s. per shift, and assistant at 15s. Against this £6 a saving might be effected by decreasing the number of hands required in what would be simply the stamping department. For the purpose of these few notes I have not taken into consideration what would probably take place, namely, an increased plate extraction, since the plates would have the undivided attention of the amalgamators.

I would like to mention here that, in at least one of their mills the Waihi Company use Muntz metal plates in preference to copper. In that particular mill they were treating the larger quantity of the sulphide ore then being obtained from the mine, and it was considered, and with good reason, too, that the Muntz metal plates were easier kept clean and in a bright condition, due possibly to the slight galvanic action set up by the zinc-copper couple. The Muntz metal, not absorbing the mercury like copper, requires rather closer attention and more frequent removal of the amalgam.

The plates were usually thoroughly cleaned up each month, and about once every two months steam was employed to remove as much amalgam as possible, for which purpose a light canvas frame was placed over the plate and steam introduced at one corner through a  $\frac{1}{2}$  in. hose and exhausting at the other corner diagonally opposite. It was usually found that, if due care had been exercised the plates would resume their normal condition within 24 hours of the operation being concluded. A weak solution of  $H_2SO_4$  was used for dressing the plates.

A further point to be noticed in Mr. Stokes' paper is the fact that all amalgamation is done after the tube mills and not as the Rand practice usually is before and after the re-grinding process. I consider that the amalgamating should take place when the ore is at its finest, otherwise I am inclined to think that a certain amount of scouring is inevitable in view of the coarse screens now

employed when tube mills are also in use. The present practice has the further disadvantage of having two amalgamating departments to control instead of one.

In concluding, I would like to state that the remark about the residues having already proved the snare and delusion of one adventurous syndicate is hardly fair to the business acumen of the men connected with it. I think I know the plant to which the writer of the paper refers, and it was erected with the idea of treating, not the current residues of the Waihi mill, but the old accumulation of tailings from the early days of the mine, before the cyanide process came into general use, when the course of treatment consisted of dry crushing with stamps and subsequent pan amalgamation with an extraction varying from 40 to 60 per cent. These old tailings have been undergoing a process of concentration due to the flow of the river for the past 10 or 12 years. If the syndicate had erected the plant with the intention of treating the current residues from the Waihi Company's mills they would not have placed it some considerable distance up the Ohinemuri River from the Waikino mill (200 stamps), seeing that all residues are discharged into the river.

The meeting then closed.

## Contributions and Correspondence.

### NOTES ON ROUTINE MINE SAMPLE ASSAYING.

Some eighteen months ago I devised a scheme whereby the green copper cupel stain could be utilised to obviate to a great extent the interchange of mine samples during the several processes undergone in assaying.

It was found that, by placing 25 mgm. of c.p. copper foil in a given fluxed sample before smelting in the wind furnace, a distinct green stain was noticeable on the corresponding cupel without practical detriment to the accuracy of either the bullion or fine gold determination of that sample, ordinary care in the manipulation of muffle temperatures having been observed, and magnesia cupels used.

We run our mine samples in lots of 28, in both wind and muffle furnaces, seven rows of four. We have four cupel trays which were made on the property, each tray having 28 partitions in seven rows of four. We also have sample trays on the same lines, but sufficiently large to allow of the fluxed samples being placed in them ready

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for carrying to the wind furnace. Cupel and sample trays are conspicuously marked I., II., III., and IV. respectively.

The fluxed samples are placed on the sample tray as under:—

1	2	3	4
5	6	7	8
9	10	11	12
13	14	15	16
17	18	19	20
21	22	23	24
25	26	27	28

No. I. TRAY.

All bags are closed with the exception of 1, 5, 9, 13, 17, 21, and 25, and into each of these seven open bags is dropped a piece of c.p. copper foil, weighing approximately 25 mgm.

No. II. TRAY.

2, 6, 10, 14, 18, 22, and 26 are coppered.

No. III. TRAY.

3, 7, 11, 15, 19, 23, and 27 are coppered.

No. IV. TRAY.

4, 8, 12, 16, 20, 24, and 28 are coppered.

So that when the four cupel trays are placed side by side with cupels in them after cupellation, their appearance may be described as under:—

I.	II.	III.	IV.
○ ○ ○ ○	○ ○ ○ ○	○ ○ ○ ○	○ ○ ○ ○
○ ○ ○ ○	○ ○ ○ ○	○ ○ ○ ○	○ ○ ○ ○
○ ○ ○ ○	○ ○ ○ ○	○ ○ ○ ○	○ ○ ○ ○
○ ○ ○ ○	○ ○ ○ ○	○ ○ ○ ○	○ ○ ○ ○
○ ○ ○ ○	○ ○ ○ ○	○ ○ ○ ○	○ ○ ○ ○
○ ○ ○ ○	○ ○ ○ ○	○ ○ ○ ○	○ ○ ○ ○
○ ○ ○ ○	○ ○ ○ ○	○ ○ ○ ○	○ ○ ○ ○
○ ○ ○ ○	○ ○ ○ ○	○ ○ ○ ○	○ ○ ○ ○

It will be seen that stained cupels are represented in thicker type than those not coppered.

An experiment was made to determine the cupellation loss when varying weights of gold bullion (10 per cent. silver), 20 gm. lead foil, and 25 mgm. c.p. copper foil were taken. This furnishes a charge practically identical with the average coppered lead button obtained by us in the assaying of mine samples.

Now, against this surcharge I found 4.5 dwt. silver (weighed collectively) in 400 gm. lead foil. In order to obtain these figures I was obliged to run some 35 cupellations with 25 mgm. copper foil (and under the same conditions as in tabulated experiment above) before I could get 20 prills together, owing to the smallness of these prills which in many cases went into the cupel. Taking into account this 4.5 dwt. silver, the gain is converted into a loss of .5 dwt. on 198.5 dwt., i.e., approximately .25 per cent., and as only one

1 oz. = 1 mgm., weighed to .5 dwt.

No.	Rough Weight of Bullion before Cupellation.	Actual Weight of Bullion Collectively before Cupellation.	Collective Weight of Prills after Cupellation.	Gain.
	Dwt.	Dwt.	Dwt.	Dwt.
1	3.5	15.5	16.5	1.0
2	3.5			
3	3.5			
4	1.5			
5	4.5			
6	6.0	30.5	31.5	1.0
7	6.0			
8	4.0			
9	6.0			
10	6.0			
11	10.5	51.0	52.0	1.0
12	10.0			
13	9.5			
14	11.5			
15	10.0			
16	20.0	101.5	102.5	1.0
17	19.0			
18	21.5			
19	20.0			
20	20.0			
TOTALS	...	198.5	202.5	4.0

sample in four is coppered, the average loss through copper dwindles to the negligible percentage of approximately .06.

The main point claimed for the copper stain check system is, that few mistakes occur when assistants know that blunders are apt to advertise themselves, and further, that interchange of trays is rendered obvious.

Implicit confidence in one's own work may be good, but unnecessary trust in others is bad.

H. MONCKTON.

Village Deep, Ltd.,  
July, 1907.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

ANALYSIS OF WOLFRAM AND HÜBNERITE.—“The coarsely powdered mineral is heated with three times its weight of sodium hydroxide in a silver crucible over a spirit lamp. Half the alkali is first melted and allowed to cool. The substance is then added, and afterwards the remaining alkali. Fifteen seconds suffice to complete the reaction, and the mass is then

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taken up with water and filtered. The alkaline liquid is oxidised by bromine and acidified with nitric acid. Sulphur is then determined by adding barium nitrate, the excess of barium is removed by sulphuric acid, and the acid liquid evaporated to dryness to remove most of the silica. Arsenic, molybdenum, and tin are next removed by sulphuretted hydrogen. The liquid is rendered ammoniacal, again treated with sulphuretted hydrogen, and the tungsten precipitated as sulphide by cautious addition of hydrochloric acid, the precipitate being calcined and weighed. Iron and phosphorus are determined in the filtrate. The residue remaining unattacked by the sodium hydroxide is readily decomposed by hydrochloric acid. It contains iron, manganese, cassiterite, and gangue (quartz), but never tungsten. The freedom of the barium sulphate precipitates from silica, and tungstic, columbic, and tantalic acids must be ascertained, as also that of the silica from the acids and that of the tungstic anhydride from silica."—P. NICOLARDOT, *Compt. rend.*, 1907, *144*, 859—861.—*Journal of the Society of Chemical Industry*, May 31, p. 530. (A. W.)

**METHOD OF WASHING LEAD SULPHATE PRECIPITATES.**—"When lead is determined as sulphate, the precipitate may be washed with a solution containing 0.7 per cent. of ammonium sulphate, instead of using alcohol, as is customary, or dilute sulphuric acid followed by alcohol. The small quantity of ammonium sulphate retained, in no way interferes with the subsequent incineration of the filter paper."—L. L. DE KONINCK, *Bull. Soc. Chim. Belg.*, 1907, *21*, 141—149.—*Journal of the Society of Chemical Industry*, May 31, 1907, p. 555. (A. W.)

**TINOL, A NEW SOLDER.**—"The author reports on a new solder, tinol, its conductivity and strength. Tinol consists of a soft solder metal, containing tin and lead in varying proportions. This composition is disintegrated to a very fine powder, and mixed with dry powder soldering fluxes. It is then worked up into a paste or viscous fluid with glycerine, alcohol, or other such media. This paste was spread over the test joints, and the joints made either by simply warming or by passing over them hot soldering irons. It was found possible to join all sorts of junctions, which otherwise would have had to be made by other methods, such as screw-clamps, etc. The conductivity experiments show a 4 to 5 per cent. higher value for tinol, while a strength test gave similar results to the older methods of soldering."—M. CORSEPSIUS, *Verhandl. des Ver. zur Beförd. des Gewerblisses*, Berlin, 1906, *237—244*; *Proc. Inst. Civil Eng.*, 1906-7, *167*, 72.—*Journal of the Society of Chemical Industry*, May 15, 1907, p. 472. (A. W.)

**QUANTITATIVE DETERMINATION OF FLUORINE.**—"The author has obtained satisfactory results in the determination of fluorine in fluorides by a method devised by W. Lenz. The substance is weighed into a small platinum crucible and mixed therein with a small quantity of pure calcium oxide. The remainder of the crucible is then filled with calcium oxide, and the whole is inverted in a larger platinum crucible, the annular space between the two being also filled with calcium oxide. The whole is then heated gradually to incipient red heat, cooled, the contents of the crucibles cautiously slaked with water, and dilute acetic acid added so long as evolution of gas occurs. The liquid is now treated with one-tenth of its volume of alcohol, and after several hours the calcium fluoride is filtered off and washed with dilute

acetic acid containing some alcohol, until the washings no longer give an immediate precipitate with ammonium oxalate. The calcium fluoride is then dried and weighed, and may afterwards be converted into calcium sulphate and again weighed. Check experiments with pure sodium fluoride and ammonium fluoride showed that the method uniformly gives results about 0.8 per cent. too low, which, considering the difficulty of fluorine determinations, represents but a comparatively small error. Commercial ammonium fluoride was found to consist of the normal fluoride,  $\text{NH}_4\text{F}$ , and not, as is frequently assumed, the acid salt,  $\text{NH}_4\text{FFH}$ . The author has also examined the iron fluoride obtained by dissolving metallic iron or ferric oxide or hydroxide in hydrofluoric acid in presence of nitric acid, and finds that instead of being ferric fluoride,  $\text{Fe}_2\text{F}_6 \cdot 9\text{H}_2\text{O}$ , as is usually stated, it is a ferro-ferric compound of the composition  $(\text{FeF}_3)_2 \cdot \text{FeF}_2 \cdot 10\text{H}_2\text{O}$ ."—E. DEUSSEN, *Monatsh. Chem.*, 1907, *28*, 163—172.—*Journal of the Society of Chemical Industry*, May 15, 1907, p. 490. (A. W.)

**DETERMINATION OF TIN AND TUNGSTEN.**—"The mixed tungstic and stannic oxides are heated in a covered porcelain crucible with zinc dust or zinc filings, for a quarter of an hour, and after cooling, the mass is heated with 1 : 2 hydrochloric acid till no more hydrogen is evolved, and all the metallic tin is, therefore, dissolved. To the cooled liquid, potassium chlorate is cautiously added till the blue colour has entirely disappeared, and the tungsten is entirely in the form of dioxide.\* The liquid is now diluted with at least 1.5 times its volume of water, and allowed to stand for 24 hours, when the whole of the tungstic oxide is deposited. It is filtered off, washed first with water containing nitric acid, and finally with a hot dilute solution of ammonium nitrate, dried, ignited, and weighed. In the filtrate the tin is precipitated as sulphide."—E. DONATH.—*Engineering and Mining Journal from Zeit. f. angew. Chem.*, 1906, vol. xix., March 23, 1907, p. 573. (A. McA. J.)

**SOME EXPERIENCES IN ASSAYING TELLURIDE ORES.**—"It has long seemed to the writer that too little is known regarding the losses in the assay of telluride ores, and while the following notes may cover ground that has been more thoroughly gone over by other assayers who may have worked out a method of testing to their satisfaction, yet he feels justified in publishing them for several reasons.

*Oré No. 1.*—This sample was brought to the writer some seven years ago, with instructions to assay for gold. The ore was of a lead-grey colour, with metallic lustre, and free from quartz or gangue of any kind. Tellurium was at once suspected, and its presence was determined by the sulphuric acid test. It was also noted that sulphur was present, and a little later bismuth was detected.

The sample was prepared as usual, and charged into crucibles with a heavy litharge flux, such as was in common use on Cripple Creek ores. Owing to the small amount of pulp at hand, only  $\frac{1}{2}$  assay ton charges were used. Silver was added. Resulting lead button was slightly brittle, but not enough so to be considered detrimental. On cupelling no button was found. Another charge was made up, more litharge and a larger piece of silver foil added. Still no button. The cupel was then ground up and assayed. Result, no button. The cupel was run again and this time a very small button was obtained.

\* ? trioxide. [Ed. Com.]

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Prof. Furman was greatly interested, and advised a combination method, consisting of treatment with nitric acid to oxidise the tellurium, precipitation of the silver with sodium chloride, and scorification of the residue for gold.

This was followed out with the astonishing result of some 50 oz. of gold and 3,000 oz. of silver per ton. The button was found to be pure silver. The sample was then submitted by Prof. Furman to a mineralogist of Denver, who classified it as an un-named mineral—a sulpho-tellurate\* of bismuth. This gentleman stated that he had recently had a sample of the ore, and advised the same treatment for it as already used, adding that the precaution must be taken to precipitate the silver with salt and not hydrochloric acid, as otherwise a combination would take place between the acid and the tellurium and re-dissolve the silver. Owing to lack of ore no further tests were made.

*Ore No. 2.*—A little later in the same year, a control sample was received from Cripple Creek with instructions to assay for gold. The sample was pulped when received, but tellurium was suspected and litharge charge run. Result, no button. Ore was tested and selenium, as well as tellurium, was found. Remembering the former experience, the ore was treated with nitric acid as before, and residue scorified with addition of silver. A gold button amounting to about 7½ oz. per ton was recovered.

The sender of the ore was notified as to its character and method of testing with instructions to tell his assayer how the ore was run 'in case he found no gold.' Much to the surprise of the writer, he received a polite letter from the assayer against whom the work was done (a man of wide experience) requesting that the assay be repeated, as he had obtained results of over 100 oz. per ton. He kindly explained his method of assay, which consisted in scorifying 1.20 assay ton of ore with an excess of lead, re-scorifying the lead button twice, and then cupelling.

This was done and his results checked (and this in direct contradiction to the published statements of at least two authorities on assaying that scorification is not applicable to tellurium ores). Considerable correspondence with the Cripple Creek assayer brought out the fact that ore from this particular mine always gave this trouble, while other ores of the district yielded their values to the ordinary litharge flux. The writer brought up the point that in all probability the acid treatment had resulted in some combination with the selenium by which the gold in the ore went into solution—the filtrate not having been tested for gold. The other party inclined to the opinion that the gold was probably rendered so fine by treatment with the acid that it passed the pores of the filter paper. At any rate, the acid method did not seem to work on this particular ore.

Having considerable pulp for experimental purposes, tests were made to determine the exact amount of gold in the ore by dissolving and precipitating. The result was 156 oz., or 50 per cent. more than what was obtained by the method used.

At the suggestion of a well-known chemist, 1.20 assay ton of the ore was wrapped in tissue paper and dropped into a scorifier containing lead that had started to scorify. The result was about 140 oz., or close to the actual value of the ore. This would seem to indicate that some loss was due to the volatilisation. No further tests were made.

*Ore No. 3.*—This was a sample of float from a grazing section of Colorado, and was not suspected

to contain any values in gold or silver. Testing it showed a very large amount of tellurium. The gangue was clay, or some variety of shale. The ore was run with the heavy litharge flux recommended by Hillebrand and Allen, and gave results of 8.80 oz. gold and 58.80 oz. of silver per ton.

One-tenth of an assay ton was scorified with an excess of lead and the resulting lead button was re-scorified and cupelled. Result, 20 oz. gold and 124.60 oz. silver per ton.

The acid treatment was used on another sample of the pulp, and gave results of 20 oz. gold and 109 oz. silver per ton. No further tests were made on account of lack of ore. The scorification method seemed to be the best for this ore.

*Ore No. 4.*—This ore was very similar in character to No. 1, although run several years later. A large sample was secured and the tests made by different methods, with the idea of obtaining the best one. One-tenth of an assay ton was run with the heavy litharge charge. The lead button was malleable. Result, 3.20 oz. gold and 1,534.00 oz. silver per ton. The silver button was dull in colour, and the cupel showed the tellurium stain with some minute beads of silver around the principal bead.

The resulting silver button was dissolved, and the silver precipitated with salt. Silver recovered weighed at the rate of 1,180 oz. per ton, against 1,534 oz.; showing that the button had consisted of something besides the silver. The filtrate was tested and showed bismuth in large quantity.

The cupel from the first button was ground up and tested. Litharge flux was used. Lead button was malleable, but resulting silver button was flat and dull in colour. Cupel still showed minute beads of silver and tellurium stain. Results from cupel, 6.40 oz. gold and 4,907 oz. silver per ton. Dissolved silver as before and precipitated. Resulting silver button weighed at the rate of 3,985.20 oz. per ton. Filtrate again showed bismuth in large quantity, and tests proved it to be bismuth beyond all doubt.

The second cupel was run again as before. Result, 412 oz. silver and 0.40 oz. gold per ton. The button of silver was round and bright, and proved to be pure silver. This cupel was again run. Result, 40.40 oz. silver and trace of gold per ton. The silver button was pure.

Total recovery by the treatment was 10 oz. gold and 5,617.60 oz. of pure silver per ton.

One-half assay ton of the same sample was treated with 100 c.c. of water and 50 c.c. of nitric acid. Allowed to stand in warm place until action ceased, 50 c.c. more nitric acid added, and after all action ceased, diluted with 500 c.c. water. Filtered and washed. The silver was precipitated with salt and lead acetate, and sulphuric acid added. Collected on filter, washed, dried and scorified. Residue was scorified for the gold. Both lead buttons were malleable, and the results were, 14.32 oz. gold and 6,465 oz. silver per ton.

One-tenth of an assay ton and 0.05 assay ton of the ore were then scorified with an excess of lead and the resulting lead buttons re-scorified with more lead until the final lead buttons were hard but not brittle. Results from 0.1 charge, 7,042 oz. silver. From 0.05 charge, 7,354 oz. silver. The gold values were neglected. The silver buttons were flat and dull in colour. They were dissolved and re-precipitated with salt, and weighed as follows: One-tenth, 5,664 oz.; 0.05, 6,230.40 oz. per ton. The filtrate showed bismuth on testing.

Results of all tests were by using litharge flux and crucible assay, re-running cupels three times and

\* ? sulpho-telluride. (Ed. Com.)

obtaining the pure silver by dissolving and precipitating; 10 oz. gold and 5,617.60 oz. silver per ton. By the acid treatment and precipitation of the silver, 14.32 oz. gold and 6,465 oz. silver per ton. By scorification, taking special pains to get rid of the tellurium by repeated scorification, 5,664 oz. silver and 6,230.40 oz. silver per ton.

**Conclusions.**—From these tests and others, the writer is inclined to believe that all telluride ores will not yield their values to the same treatment. A straight telluride ore, even of high percentage, will apparently give good results by the scorification method, but where other metals, such as bismuth or selenium are present, it fails to work well. The crucible method with excess of litharge has not proved to be very good in the writer's case, although other assayers seem to get satisfactory results. Whether they have had the same ores to contend with is not known to the writer. The combination method should yield good results if properly carried out.

The principal losses seem to be in absorption by the cupel and not by volatilisation. This will depend somewhat on the ores, however. The presence of other metals than gold and silver in the button in the cupel is something that was new to the writer, and not generally known. He has thoroughly satisfied himself that bismuth at least will be retained by the silver.

The most important feature of the results is that some of the ore tested was considered of no value, and if tested by the ordinary crucible method with the usual flux for sulphide ore would give no button. Is it not possible that some good bodies of ore have been overlooked, because the camp assayer reported in all good faith that the ore carried no values?—J. N. McLEOD.—*The Mining World*, May, 1907, p. 622. (H. A. W.)

**THE ESTIMATION OF FREE ACID IN THE PRESENCE OF METALLIC SALTS.**—"The author in stating that the end point of titrations of free acid in the presence of metallic salts when caustic alkali is used is frequently obscured by the tendency of the hydrates to precipitate, suggests the use of bicarbonate of soda, his method being dependent on the following conditions:—

1. The bicarbonates of these metals are slightly but sufficiently soluble for the purpose of keeping the solution clear, and are neutral to methyl-orange, whereas bicarbonate of soda is strongly alkaline to the same indicator. 2. The red colour of an acid solution of methyl-orange and the blue colour of a sulphate of copper solution are complementary, and when mixed in the proper proportions give an almost colourless solution. For example, if to 100 c.c. of a 3 per cent. solution of pure copper sulphate four drops of a methyl-orange solution be added, a pure yellowish green colour is produced. If to this green solution a  $N/2$  solution of sulphuric acid be carefully added, it will be noticed that after the addition of two drops or 0.1 c.c. the solution becomes practically colourless. At this stage the further addition of one drop of  $N/2$  acid produces a distinct pink coloration, whilst the addition of one drop of  $N/2$  bicarbonate of soda produces a distinct green coloration, the original green of the neutral solution only being obtained after the addition of two drops or 0.1 c.c. of the  $N/2$  bicarbonate. The further addition of bicarbonate produces no marked change in colour, and no precipitate until a considerable excess has been added.

The colourless stage of the solution and the remarkably sharp change to pink or green according

to whether an excess of acid or alkali is added, gives an excellent indicator in the titration of the free acid in solutions of sulphate of copper, even when containing large proportions of the sulphates of nickel, iron, zinc, etc.

The titration is carried out in the following manner:—Three grammes of the sulphate of copper crystals or such a quantity of the solution to be tested as would contain about that weight of sulphate of copper, is put into a 10 oz. conical flask, and diluted with 100 c.c. of distilled water, and four drops of methyl-orange solution added. If the colour produced is pink, the solution is titrated with  $N/2$  bicarbonate of soda until the colourless stage is reached; the amount of free acid in the material tested is equivalent to the number of c.c. of  $N/2$  carbonate used, plus 0.1 c.c. If, however, the colour produced on the addition of the methyl-orange is green, then the acidity of the solution cannot be greater than the equivalent of 0.1 c.c.  $N/2$  bicarbonate, and in this case the solution is titrated to the colourless stage with  $N/2$  acid; the amount of free acid in the solution is therefore equivalent to 0.1 c.c. minus the number of c.c. of  $N/2$  acid used.

The determination of the free acid in solutions of sulphate of copper by this method is accurate to 0.04 per cent. of free acid on the weight of sulphate employed, but involves a constant correction of 0.08 per cent. or 0.00245 gm.  $H_2SO_4$  above the actual amount shown by titration with  $N/2$  bicarbonate to the colourless stage.

The end reaction is still sharp in 5 per cent. solutions, and if preferred  $N/10$  solutions of acid and bicarbonate may be employed, but for all practical purposes the above strengths will be found most suitable for works or laboratory use.

The method is also applicable to salts of nickel or iron, etc.; such, for example, as galvaniser's 'pickle,' in which case stronger solutions may be employed than with coloured salts, such as sulphate of copper or nickel.—EDGAR SIMS.—*Chemical News*, May 31, 1907, p. 253. (A. W.)

**ON THE DIRECT QUANTITATIVE SEPARATION OF CHLORIDES AND BROMIDES.**—All methods recommended for the direct separation of Cl from Br depend on oxidation applied to solutions of bromides and chlorides, but the oxidants used hitherto have possessed a higher oxidising potential than that of an aqueous solution of Cl, and hence the success of these processes have depended on interrupting the process exactly at the point when all the bromine is liberated. Bugarsky first pointed out the necessity of using oxidants whose oxidising potential should be intermediate between that of chlorine and bromine, and the one selected was  $HIO_3$ . This method consists in adding to the Cl' and Br' solution, sulphuric acid and potassium di-iodate, and then distilling

$$KH(IO_3)_2 + 10KBr + 11H_2SO_4 = 11KHSO_4 + 5Br_2 + I_2 + 6H_2O.$$

The  $Br_2$  and  $I_2$  pass off with the steam, and the solution contains the HCl and excess of iodic acid.

This method was modified by the author by increasing the amount of iodate taken, replacing the  $H_2SO_4$  by  $HNO_3$ , and removing the excess of iodic acid by a single reagent which would introduce no sulphate or other injurious compound, and the excess of which would require no removal. This method in use for six months gave uniformly satisfactory results, and also with the determination of Cl in crude bromine.—L. W. ANDREWS.—*Journal of the American Chemical Society*, vol. xxix., 3, March, 1907, p. 275. (J. A. W.)

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**BICHROMATE TITRATION FOR IRON.**\*—“L. Brandt states (*Zeit. anal. Chem.*, 1906, xlv., 95-99) that in the determination of iron by titration with bichromate solution, diphenylcarbazide (CO(NH.NH.Ph)<sub>2</sub>) has the advantage over other indicators of being used in the titrating vessel, and not outside it. The method is as follows: To about 1.5 litres of water in a capacious dish are added 60–80 c.c. of dilute hydrochloric acid and 100 c.c. of a solution containing 10 (sp. gr. 1.12) and 55 c.c. of dilute sulphuric acid, 1:3, and 5 c.c. of phosphoric acid of sp. gr. 1.7. The iron solution which should contain 0.2–0.5 gm. of iron, and 5 c.c. of 0.1 per cent. solution of the indicator are further added, and the titration with bichromate performed. There is, at first, a bright red-violet colour which finally changes sharply to the green of chromic chloride. The indicator may also be employed in the determination of chromic acid, an excess of standard ferrous ammonium sulphate solution being added, and the process completed as above described. The solution of the indicator in acetic acid should be prepared fresh every few days.”—*Engineering and Mining Journal*, April 6, 1907, p. 667. (J. A. W.)

**NEW PROCESS FOR WHITE LEAD MANUFACTURE.**

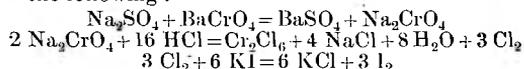
“The production of white lead in such a way as to avoid as far as possible injury to the health of the workers has long been a problem, with which inventors have been busy. We learn from the *Western Electrician* of April 13 that Mr. Clinton P. Townsend, of Washington, has perfected an electrolytic process which appears to avoid the evils hitherto encountered in producing electrolytic pigments—namely, the irregularity in composition and quality and deficiency in covering power. He has accomplished this by a circulation of the electrolyte successively past the respective electrodes, and preferably in the direction from the anode past or through the cathode. Any type or form of cell may be employed, but the sections must be so formed as to provide between them and the sides of the cell a series of channels. The method of arranging the anode and the cathode with provision for free circulation is described, and a filter-press is proposed for dealing with the electrolyte, which must be made to move freely in the requisite directions. It is pointed out that not only does the electrolyte circulate from the anode past or through the cathode, but the pigment or insoluble compound formed at the anode detaches itself therefrom and is carried upwardly by the electrolyte past or through the cathode and thence to a region on side the field of electrolysis. In operation a lead anode is employed and an electrolyte consisting of any salt, the acid radical of which forms a soluble compound with lead. To this solution is added a soluble carbonate. It is stated that a mixture of sodium nitrate or acetate and sodium carbonate, in proportions approximating ten to one, constitutes a satisfactory electrolyte. The current density depends in some measure on the speed of circulation of the electrolyte; and it may be increased as the flow becomes more rapid.”—*Times Engineering Supplement*, May 8, 1907. (J. A. W.)

**SULPHUR IN COKE.**—“The appearance of the coke offers no characteristics as to its percentage of sulphur. A short description of a method of analysis, unfortunately little known, but both simple and reliable, may therefore be useful:—

Into a steel, or better, nickel, crucible of about 50 c.c. capacity, 11.5 gm. of pulverised peroxide of sodium and 0.7 gm. of coke finely pulverised, or

16 gm. of Na<sub>2</sub>O<sub>2</sub> and 0.7 gm. of coal should be weighed. The materials should be mixed thoroughly, and the crucible placed on a small tripod in a beaker, so that between the bottom of the crucible and that of the glass there is a distance of 1 cm. Then the beaker should be partly filled with distilled water, so that the crucible is about half covered by water. In the lid of the crucible there is a small hole through which a glowing wire is conducted to light the contents. After about three minutes, the crucible should be put aside, and the contents dissolved. Thereupon the crucible, lid and tripod should be rinsed well, and taken from the glass. So much concentrated hydrochloric acid should be added that the solution just reacts acid; this is seen at once when the acid is slowly added, as soon as a light green colour appears, and the fluid is quite clear. Then the solution should be boiled, and some drops of ammonia, just in excess, should be added and mixed with 15 c.c. of chromate of barium (solution 23 gm. of BaCrO<sub>4</sub>, 80 c.c. of HCl concentrated, 920 c.c. of distilled water). One should make sure if there is ammonia in excess, and boil until no smell of ammonia can be perceived; the solution must then be boiled down to a quantity fit for titrating. Then one should filter it, wash the sediment with water, and add 1 gm. of iodide of potassium to the filtrate. Subsequently one should cool down to 30 deg. C., add 5 c.c. of hydrochloric acid and starch paste as indicator, and titrate with a decinormal solution of thiosulphate of sodium.

The course of the analysis after Na<sub>2</sub>SO<sub>4</sub> has been formed from the mixture by combination with Na<sub>2</sub>O<sub>2</sub>, is the following:—



Each atom of sulphur releasing three atoms of iodine. Special attention, in this analysis, several of which can be carried out within an hour, should be principally paid to two points:—Firstly, that the solution is filtered thoroughly. The filtrate must be absolutely clear; only finest filters should be used and the washing done carefully. If one is not in a great hurry, one should let the solution stand over a little before filtering, so that the sediment granulates a little. The second important point is the preparation of the solution of chromate of barium. This preparation should be made—as chromate of barium to be had on the market mostly has an excess of chloride of barium or of chromate of potassium—and washed well. Traces of either of these impurities mentioned affects the result of the analysis.

It is not advisable to carry out this analysis with coal, as coal mixed with peroxide of sodium causes an explosion when lighted.”—A. THAU.—*Iron and Coal Trades Review*, May 24, 1907, p. 1855. (A. McA. J.)

**ON FERROUS AND FERRIC SULPHIDES.**—The main conclusions are stated as follows:—

1. Precipitated FeS does not react with Zn(OH)<sub>2</sub> in the cold or at boiling temperature and only slowly at 160 deg., the products being colourless ZnS and Fe(OH)<sub>2</sub>.

2. In general, the statement that ferric salts give a mixture of FeS and S, when precipitated in alkaline solution by an alkaline sulphide is erroneous. The product is ferric sulphide which reacts with Zn(OH)<sub>2</sub> rapidly in the cold, instantly on boiling to form ferric hydroxide and ZnS.

3. Alkaline polysulphides precipitate ferrous salts as ferric sulphide.

\* See this *Journal*, vol. vi., p. 376.

4. Ferric sulphide readily hydrolyses into ferric hydroxide and hydrogen sulphide at 100 deg.—H. N. STOKES.—*Journal of the American Chemical Society*, vol. xxix., 3, March, 1907, p. 304. (J. A. W.)

### METALLURGY.

**SUCCESSFUL TUBE MILL LINING.**—"The accompanying illustration depicts what has proved in practical work to be the most successful tube mill lining yet tried. The lining proper consists of a number of hard cast iron segments each 4ft. long by 1 ft. in width with its base properly curved to fit the shell of the given diameter (66 in. in this case).

The base of the shell is of 1 in. thickness while the radial riffles extending longitudinally are  $3\frac{1}{2}$  in. apart at their tips, and  $3\frac{1}{4}$  in. at their bases. The segments are laid end to end and bolted to the mill shell of  $\frac{1}{2}$  in. boiler iron in the manner shown. When all the segments are in place the usual flint pebbles are fed in and quickly jam so tightly in the riffles that they can be removed in most cases only by breaking out with a hammer.

The illustration shows the interior of a 24 ft.  $\times$  5 $\frac{1}{2}$  ft. tube mill of Allis-Chalmers make fitted with such a lining into which the pebbles have jammed, thus effectively protecting the metal. This mill is one of four in use at the Los Estrellas plant, El Oro, Mex., where the lining was invented. The neighbouring El Oro Co. operates its own foundry where the linings were cast, and this foundry accommodates the other mines of the surrounding region.

The original idea of the inventor was to provide riffles or cogs in the shell which would catch up the pebbles and carry them nearly to the top before allowing them to drop, thus greatly increasing their crushing action on the ore in addition to that of their regular grinding action. Much to his surprise, he found on opening his experimental tube, after several

the new riffle linings modified from the original design A to the design B now in use. In like manner the other mills of the El Oro region discarded silex linings, and the new lining is now in use at them all. For 8 months, these Los Estrellas linings have been in use in the mills, each grinding 100 tons per day to a pulp, of which about 50 per cent. is finer than 200 mesh and not more than 5 per cent. is 100 mesh.

A recent examination of the original lining at the time of taking the photograph disclosed that the wear on the metal riffles was perceptible only on the rounded outer edges. This slight wear is taken to indicate that the probable life of the metal segments will be from 2 to 3 years, a record yet to be broken."—*Mines and Minerals*, June, 1907, p. 520. (A. McA. J.)

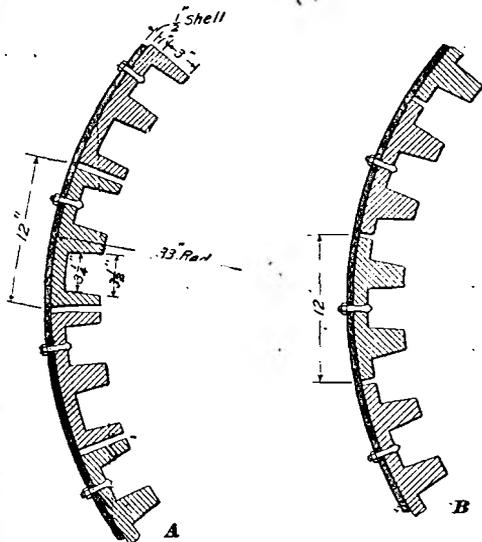
**CYANIDE CONSUMPTION DECREASE.**—"In regard to saving cyanide in the treatment of gold and silver ores, we have been enabled by a simple method to reduce our cyanide consumption from 4.5 to 1.5 lb. per ton of ore treated, and the results are so interesting that I herewith tender them for publication.

When we first commenced treatment in October, 1905, our protective alkalinity in the working solutions was carried at 0.04 per cent. (in terms of caustic soda), and the cyanide consumption varied from 4.3 to 4.5 lb. per ton of ore.

The ore is valuable for its silver, which occurs as a sulphide, and carries from 12 to 20 oz. of this metal per ton, together with a small quantity of gold. We find that 5.2 lb. of lime per ton is sufficient to keep the alkalinity at 0.04 per cent. Our extraction has been good, averaging 93 per cent. of both gold and silver, and, accepting the fact that a certain amount of cyanide will of necessity be constantly tied up as a double cyanide of zinc and sodium (using sodium cyanide), we have not considered the amount abnormal, though the advisability had been discussed of installing electrolytic precipitation to obviate this loss of potential free cyanide. However, in reading over 'The Cyaniding of Gold and Silver Ores,' by Julian and Smart, I noted what was said in regard to the decomposition of the double cyanide ( $\text{Na}_2\text{ZnCy}_2$ ) by an increase in the free alkali. W. R. Feldtmann is there quoted as saying: 'The addition of alkali to working cyanide solutions which have become somewhat weak in alkali, brings up the strength by regenerating, *i.e.*, decomposing, the zinc cyanide so that, as a matter of fact, when the solutions are pretty strongly alkaline they contain no zinc as cyanide, but only the hydrate dissolved in alkali.'

With the object of ascertaining what results could be obtained by increasing the alkalinity, I commenced adding lime, and kept gradually increasing the same until the solutions tested 0.2 per cent. alkalinity. Our working solution for slime, which we carry at 0.125 per cent. KCy, at once began to gain in strength, and kept growing gradually stronger until it showed 0.3 per cent. KCy; the alkalinity was then allowed to fall to 0.09 per cent., when cyanide strength also fell. After many experiments with various strengths, I found 0.135 per cent. to be the least alkalinity with which I could regenerate, and consequently I have kept the alkali at that strength ever since.

As a result of this regeneration, our cyanide consumption has not exceeded 1.5 lb. per ton treated for more than five months; as a matter of fact, no cyanide was added to the slime-treatment solutions for nearly 13 weeks, and the amount used to bring



hours of grinding, that the pebbles had wedged tightly into the riffles, thus giving it a lining of pebbles automatic in originally placing itself and in replacing worn pebbles with new ones.

This lining was thus found to need neither attention nor repairs. Its usefulness and great efficiency having been demonstrated by severe tests, the former silex linings of all four tubes were replaced by

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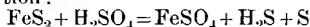
the leaching-plant solutions up to strength was small; this, of course, was due to the large excess of zinc cyanide then existing in the system.

In regard to the testing for alkali, we first determine the free cyanide by titrating with standard silver nitrate solution to the first faint opalescence, without the use of the potassium iodide indicator; we then add a few drops more of the silver nitrate to make the opalescence well defined, and titrate with  $N/5$  oxalic acid solution using phenolphthaleine as indicator. This is the method recommended by J. E. Clenell, and as it has been found to check very closely with L. M. Green's ferrocyanide method, and also with that of Gerald Williams,\* it has been adopted by preference on account of its simplicity.

Finally, I would say that up to date there is no indication of increase in our cyanide consumption, and as this work has been going on for the past five months, I have come to the conclusion that the regeneration has assumed constant proportion."—L. McN. B. BULLOCK.—*Mining and Scientific Press*, June 8, 1907, p. 719. (A. McA. J.)

**TREATMENT OF MATTE FROM THE CYANIDE MILL.**—"It may be of interest to some to know of a quick and complete way in which to extract the metals from a rich matte as obtained at a cyanide clean-up. It often happens that the yield from this source adds considerably to the month's bullion.

Sulphur, the matte-forming element, comes from zinc sulphate ( $ZnSO_4$ ) remaining with the precipitate after acid treatment. The amount of  $ZnSO_4$  present will depend, of course, upon the number of water-washes used in the final acid treatment. The more the precipitate is washed, the less will be the matte formed on a gold button. I believe that in our case (treatment of concentrate) it is possible at times for extremely fine pyrites to escape from the filters, and be carried in suspension to the zinc boxes where it is bound to remain with the other precipitate. The amount, however, is exceedingly small. If such be the case after treatment with acid, ferrous sulphate ( $FeSO_4$ ) will be contained in solution, while free sulphur separates out with the precipitate according to the reaction:



There is a possibility then, unless a complete roast is given to the acid-treated precipitate, of sulphur being present as a matte-forming material upon melting the final precipitate. The amount of sulphur obtained from this source to form matte will in all probability be slight. Experiments now point to the fact that the amount of matte obtained on a gold button depends principally upon two things, namely, the completeness of the final water-washes after acid treatment and the extent to which the final roast has been carried (whether merely a drying of the precipitate or a complete roast).

The matte that I am treating contains zinc, lead, iron, gold and silver. When using the zinc-lead couple for precipitation, the presence of lead in the matte, if the gold slime is acid treated and then smelted in a graphite pot, is practically inevitable. Lead in the matte may also be due partly to the dissolving (by the cyanide solution) of lead salts from oxidised concentrate followed by the precipitation of lead on the zinc. It may originate from the lead acetate used in some solutions. There are two sources by which iron may enter the matte, either by iron pyrite collecting in the zinc boxes and finally appearing in the acid treated precipitate as  $FeSO_4$ ,

or by some iron scale gathered from the roasting pan with the gold precipitate. Zinc seems to enter the matte in considerable amount, especially when the acid treated slime contains some undissolved zinc or has been only partly roasted, so as to contain zinc sulphate.

I believe that a small amount of matte on a button is desirable, for the slag above will, as a general rule, assay less in silver, especially when the matte contains lead. No nitre is used by us to keep down the matte upon fusion of the slime in crucibles. Its damage to the pots amounts to more than the cost of treatment of the additional matte. Previously to using my present method of reduction, we certainly did not welcome the presence of any considerable amount of matte at a clean-up, since the old method of treating small quantities with nitre or scrap iron was slow and incomplete. It is not safe to allow any amount of this rich matte to accumulate (it assays about \$20 per lb.), and for that reason the best plan is to treat it immediately at the end of the clean-up, melting the bullion with the gold buttons into bars.

Potassium cyanide acts as a powerful reducing and desulphurising flux, and for this reason it was used for decomposing the matte. Our particular matte is unusually high grade, assaying as much as \$45,000 per ton. It contains zinc, lead and iron as sulphides, besides the precious metals. At the monthly clean-up about 800 lb. precipitate is obtained, producing nearly 3,000 oz. bullion, and from the treatment of this we obtain from 30 to 40 lb. matte. The amount varies from month to month. The method of treatment for the matte is as follows:—

The matte, borax and cyanide are put separately through a small rock-breaker and crushed fine. Borax and cyanide are used as fluxes and are put with the matte into three No. 60 graphite pots in the following proportions and order: There are alternate layers of borax, matte and cyanide throughout, until the pot is full, and finally covered with borax. The crucibles are now put into the furnace and a white heat maintained for two or three hours. As a rule, it takes a little over two hours with a good hot furnace. It can be seen when the action is complete, for the charge will subside and the bubbling cease. The action and burning of the sulphur will be violent at the end of the first hour. Also the slag will become quite thick when the action is complete, being removed with a skimmer; this is necessary before pouring the contents of the crucible into a conical mould. If an excess of cyanide be used, it will be found just above the gold button and can be broken off by a blow with a hammer. The matte will now be entirely decomposed, leaving only a light porous slag and the gold button.

This method shows an extraction of 85 to 94 per cent. of the total value of the original matte, depending upon the richness of the material treated."—A. E. DRUCKER.—*Mining and Scientific Press*, May 18, 1907, p. 638. (A. McA. J.)

**CLEANING OF FILTER PRESS PLATES.**—"At the Kalgoolie mines, where the 'all roasting and sliming process' is in operation, it is found in the subsequent filter-pressing of the pulp that, owing to the presence of hydrated calcium sulphate in the cyanide solutions, there is a tendency for the filter-press plates to become coated with a very hard and extremely adhesive cement, which in time may completely fill the fittings and choke the channels of the plates. The removal of this incrustation has been found difficult and costly, says a report of the Chamber of Mines, Western Australia. The use of

\* See this *Journal*, vol. iv., p. 288.

pneumatic tools and of 'sand blast' has been resorted to for the purpose. With badly incrustated plates, the latter is the more effective and less costly method of the two, but, costing as it does approximately 20s. per plate in addition to the cost of removal from, and replacing in position, the expenditure is one that causes serious consideration.

Mr. A. Wauchope reports that at the Great Boulder Main Reef mine early this year it was decided to take advantage of the fact that dehydration of the hydrated calcium sulphate (gypsum) cement would have the effect of removing its cementive properties for the time being, and would leave it in such a condition that its removal would be comparatively easy. To this end an oven was constructed, wherein the plates—about 25 at a time—were vertically suspended, leaving a space of three or four inches between each plate. The roof of the oven was removable, to facilitate the introduction and withdrawal of the plates, and consisted merely of sheets of iron. No fire grate was used, as the object was to produce a comparatively low and uniform temperature, lest strong heating would buckle or crack the plates. All cocks were removed. After 24 hours' heating the plates were removed from the oven and attacked with suitable tools and brushed with brushes made of stout steel wire rope. It was found that the scale was so softened as to permit of its easy removal, both from the fluting and the channels. In addition, all the old rubber rings, blanket strips, and P. and B. paint were removed at the same time. No buckling or other damage resulted to the plates. The total cost of removing the plates from the presses, conveyance to oven, firing, cleaning and replacing in presses, amounted to 10s. 6d. per plate.—*Australian Mining Journal*, June 19, 1907, p. 568. (A. McA. J.)

A STANDARD FOR THE FINENESS OF GROUND POTTERY MATERIALS.\*—An elutriation process is considered by the author of this paper to be the only simple method for estimating the amounts of particles of different sizes in a mass of finely ground material. For this purpose, instead of the Schoene apparatus, the following, used by Krehbiel, is recommended. The apparatus comprises three cylindrical copper cans, the bottom of each terminating in an inverted cone of 60 deg. A stream of water is fed into a thistle funnel, which passes to the bottom of the first can, and the overflow from this passes into a similar thistle funnel, and to the bottom of the second can, and similarly from this to the third can. The diameters of the second and third cans are arranged so that the velocity of the stream of water as it rises in the cans decreases, and, consequently, smaller and smaller particles are carried over. The stream is supplied to the first can from a receptacle, provided with an overflow, to ensure a constant pressure. The following table shows (1) the height, (2) the diameter, and (3) the speed at which the water rises in each can, and (4) the size of grain removed:—

	1.	2.	3.	4.
	cm.	cm.	mm. per sec.	mm.
First can ...	15	5.00	1.50	0.04—0.025
Second can ...	15	7.32	0.70	0.025—0.015
Third can ...	15	14.58	0.18	0.01—0.00

\* See this *Journal*, vol. vii., Sept., 1906, p. 72.

The stream from the third can, carrying the milky washings, is allowed to flow away. 20 gm. of the finely-ground material are agitated with water and introduced into the first can, and the stream of water started. Particles whose diameters are above 0.04 mm. are left in the first can, those above 0.025 mm. in the second, and those above 0.01 mm. in the third, and these fractions are afterwards taken out, dried, and weighed, the fraction whose particles have diameters less than 0.01 mm. being estimated by difference. After obtaining the fractions the grains of which have diameters within the limits just mentioned, the fineness of the sample can be expressed as a numerical value, by the 'surface factor' suggested by Jackson, and modified by Purdy.

[The 'surface factor' is obtained in the following way:—Invert the mean diameter of each group. Multiply the 'surface factor' of each group, thus obtained, by the fractional amount of the quantity of the sample having particles of these mean diameters. The sum of these products is the total 'surface factor' for the sample.]

The following examples may be quoted to show the order of the figures of 'surface factors' of some American ground materials:—

Felspar, Eureka Co. body spar	...	...	1028.
„ „ lawned 160	...	...	1047
„ „ calcined	...	...	1198.
Flint „ French flint	...	...	1031.
„ „ „ lawned 160	...	...	1111
„ „ „ re-ground	...	...	1324.

—C. F. BINNS, Transactions American Ceramic Society, 1906, p. 244.—*Journal of the Society of Chemical Industry*, April 15, 1907, p. 321. (W. A. C.)

PANS *v.* TUBE MILLS.—“Some tests carried out at the Dolores Mill, in Chihuahua, Mexico, shew the relative merits of Pans *versus* Tube Mills as sliming machines.

In Test No. 1, the ore was crushed with cyanide solution by 15 stamps, weighing 900 lb., through a 2 mesh screen, the pulp passing to the two Bryan mills with 60 mesh screens, then to four pans in series, thence through two settlers, and finally, through two cone-classifiers. The overflow from the classifiers went to the agitation plant, and the underflow was returned to the tube mill for re-grinding, the re-ground product joining the stream from the settlers at the head of the cones.

In Test No. 2, the pulp from the Bryans passed directly to the tube mill, thence to the cones, the overflow as before to the agitation plant, the underflow being returned to the head of the pan series, thence through settlers to cones as before.

In both tests samples were taken every hour, over periods of twelve hours, from the heads of the tube mill and heads of pans, and also from the discharge of same. The bulk 12-hour sample was determined from the proportion of sand and slime in each sample, the difference in slime content between heading and tailing being taken as the sliming power of each machine.

The following are the results obtained, from the average of five samples:—

TEST NO. 1.		TEST NO. 2.	
Percentage of slime.		Percentage of slime.	
Tube mill heading 8.20	Pan heading .. 11.02	„ „ heading 8.20	„ „ heading .. 11.02
„ „ tailing 42.64	„ „ tailing ... 52.54	„ „ tailing 42.64	„ „ tailing ... 52.54
Slimed by tube mill 34.44	By pans .. 41.52	Slimed by tube mill 34.44	By pans .. 41.52

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TEST No. 2.		Percentage of slime.	Percentage of slime.
Tube mill heading	4.66	Pan heading	... 7.36
" " tailing	44.08	" " tailing	... 51.88
Slimed by tube mill	39.42	By pans..	... 44.52

Taking the average of both tests, to bring the machines under the same conditions, gives the following result:—

Percentage slimed by tube mill	... 36.93
" " " pans	... 43.23
" " in favour of pans	... 6.09

The pans used are of the Wheeler type, with plain flat miller and dies; they have a diameter of 5 ft. The speed is 65 revolutions per minute. The power required to drive the four pans is 50 h.p. The tube mill is a machine specially constructed by Allis-Chalmers, made in sections for mule transport, and I believe is one of the first, if not the first sectional tube mill made. It has a length inside of 16½ ft. by 3 ft. 6 in. diameter, and runs at a speed of 35 revolutions. It is lined with white iron, and carries a load of 4½ tons of flint pebbles. The power required to drive it is 20 h.p.

Although the above tests showed a small margin in favour of the pans for sliming, yet when the questions of power and cost are considered the tube mill is undoubtedly the more economical machine. The wear and tear in the case of the pans is considerably more than that of the tube mill, and the difference in power required for driving, by itself would more than minimise any advantage the pans have in sliming capacity."—ROBERT CLARKE.—*Mining and Scientific Press*, April 6, 1907, p. 431. (T. L. C.)

MINING.

THE DEVELOPMENT OF THE HAMMER DRILL.—“No doubt, those who are interested in mining or the improvements in mining machinery are aware that during the past three years many changes are noticeable in the methods used in breaking ground. Since the introduction of the first piston drills, many years ago, improvements have, of course, been made from time to time, but the general construction has been the same. While admitting that the cost per foot broken was, in many cases, higher than with hand drilling, operators have continued using the larger 3 in. to 3¼ in. machines in their cross-cuts and shafts, and 2½ in. to 2¾ in. machines in their drifts, raises and stopes. At the same time, there have been many complaints made against the operating and maintenance expenses.

Many changes have been inaugurated in the past three years in the methods of rock breaking, and from the subject of this paper you will naturally infer that I intend to show what results have been obtained by the use of the air hammer drill.

It has been proved conclusively that with the improved type of hammer there is very little shock or vibration to inconvenience the operator, and that the remedy does not lie entirely in the improved machines. The conclusion is that, with a gad made with the proper taper, at each blow of the piston the tool is forced into the crevice and remains there until the next blow is struck. This leaves the operator with but the natural vibration of the hammer to contend with. In the later types of machines the piston is made to cushion on live air on the back stroke, and no excessive jar occurs.

A difficulty is developed in the construction of the drill bits. These were made of 7/8 in. hollow staybolt iron, with 6 in. of hollow steel welded to either end for a shank and bit. It was found that crystallisation would take place at the welds and either the shank or bit would jump off in a very few minutes. We turned our attention at first to the steel, and after careful experiments found that the bits with six cutting edges cut faster and held their gauge better than any other style. The cross bit was tried, but was discarded on account of its tendency to ‘rifle’ a hole and ‘hang up’ in crossing slips at an acute angle. We also found that with this shape of bit it was hard work to rotate the drill by hand in soft ground, and, while it cleared better, the six-tooth bits gave much better results. Sharpening tools were made for the six-tooth bits, and little difficulty was experienced in drill sharpening after the blacksmith had acquired the knack of spreading the gauge.

Hollow steel was necessary in drilling down holes or flat holes in sticky ground, as it was essential to force part of the exhaust air through the steel to clear the hole of cuttings. The welded bits would not stand in hard rock, and we were in a quandary as to how we would overcome this difficulty, until it was discovered that a 3/4 in. hole could be drilled through a 4 ft. piece of steel in a gun-barrel lathe. We used this class of steel for some little time, until we found that we could purchase a brand of hollow rolled steel in Sheffield, England.

With these changes we were able to drill 4 ft. holes in any ground at a speed varying with the formation of from 1 to 10 in. per minute. In testing the machine thoroughly, however, we found that a cast-steel cylinder would crystallise easily, and we were forced to change the material used in its construction to the best grade of steel shafting, forming the complete cylinder out of the solid metal.

We then found that it was next to impossible to get men who would hold the machine over their heads and drill uppers. One could not blame the drill runners, as it was certainly an awful task to set a man at, and only the very strongest could stand it for any length of time. The screw-feed machine mounted on a light drift column was tried, but finally discarded, when one of the manufacturers constructed an air-feed attachment. With this machine it was possible to ‘set up’ and ‘tear down’ in less than a minute, and the drilling capacity of the machine was increased 100 per cent. Machines of this type were placed in all of the larger mines of the country; and it was soon found in competitive tests that they would out-drill any of the 2¼ in. machines.

In the large cross-cut or railway tunnel, the large piston drill still holds the advantage, as it is possible to drill deep rounds and break the ground far more economically and rapidly than would be possible with the small hammer type. Even in this class of work, however, there is a place for the small drill, as they can be used most successfully in block holing and in taking up bottom or trimming up the walls where projections are left, either on account of missed holes, or because the drill runner has miscalculated the load he had on a certain hole. They can also be used for drilling uppers in the roof for pipe hangers, or for putting in short relief holes in the heading, where they will give the cut holes a better chance to break.

In shaft or winze work, however, they can be used for drilling the full round of holes, and even in the larger sized shafts, records show a saving which is

simply astounding. To those who are familiar with this class of work, the reason is obvious. With piston drills, even in the hands of the best of runners, more time is lost in setting up, tearing down, lining up, cranking in and out of holes and in changing steel, than is consumed in drilling. Where the ground is even and breaks well, it is not so bad, but where slips are encountered frequently, and time is consumed in preventing a 'hang up,' or where the formation is such that a great many holes must be drilled to break the ground, the air-hammer drill has all the best of the argument.

The machine, steel and hose, can all be put into one bucket, and the drill runner can be at work in five minutes from the time he goes on shift. The steel being loose in the chuck, he loses no time in changes, and he can start a new hole while a piston drill runner would be cranking out. Holes can be drilled in any direction and pointed to take advantage of any slip. No holes need be lost, as it is practically impossible to get a 'fitchered' hole even in the worst kind of ground. All of the holes can be drilled to any depth desired, up to 4 ft., except in very heavy sulphide ore, and the sump holes can be put in any part of the shaft.

When timbering commences, the small hammer drill is indispensable for cutting hitches and taking out projections where they interfere with the sets, and for these purposes alone, will pay for itself in 30 days.

In a drift, a drill runner can either mount the air-feed drill on a column or, where it is possible to pull the cut out of the top, he can work with the air feed alone.

There is a diversity of opinion as to whether it is advisable to use a 2½ in. piston drill or an air-hammer drill in this class of work, and a decision, from an economical standpoint, can only be obtained by actual tests. As a usual thing, however, the hammer drill can be used advantageously in all drifts where a 4 ft. round, bottom 1½ in. will break without leaving any 'guns.' With a piston drill of the size mentioned or larger, it is possible to make more rapid progress, but the cost per foot will probably be greater.

In ground which breaks short, however, the piston drill usually has the advantage, as a large hole is drilled, and the powder can be got to the bottom of the holes where it belongs.

In stope work the 2½ in. machine does not have a chance against the air-hammer tool, and the records on some of the Cripple Creek properties show that their stoping costs have been reduced one-half since the introduction of the small machines. Where the values lie in small streaks of very high grade ore, the saving is enormous, as the stopes can be carried even narrower than by hand, as the drill can be run in any place, a runner can get his body.

Not only can more work be done with hammer drills than with piston machines, but the saving in repairs is well worth considering. In nearly all of the small hammer drills, there is but the one moving part, and when properly made, this should last for months. The rotation being accomplished by hand, there are no pawls or springs to break, and no rifle nuts to wear out. As the machines are fed by air, no feed screws or nuts are required. Taken as a whole, with a properly made air-hammer drill, the repairs are less than 25 per cent. of those of a 2½ in. machine.

Working at a drill pressure of 100 lb., a 2½ in. machine requires 100 cub. ft. of free air per minute,

while an air-hammer drill of the valveless type uses but 25 cub. ft. at the same pressure.

Another advantage possessed by the hammer drill lies in the fact that it is not necessary to employ a first-class drill runner, as a good miner, who knows how to point his holes, will do just as much work after two or three days' experience, as the best man one could hire."—H. L. SINCLAIR.—*Compressed Air*, May, 1907, p. 4,450. (H. A. W.)

**SPECIAL METHODS OF SHAFT-SINKING.**—"Under the head of special methods of shaft sinking, I propose to consider the methods that have to be resorted to, when ordinary methods cannot be applied economically on account of an excessive influx of water. Such complications are mostly confined to coal mining, and arise but rarely in metalliferous mining, mineral deposits usually lying in the older, less heavily watered rocks.

Year by year, as the coal seams available within the area of the exposed coalfields are more extensively drawn upon, it becomes increasingly necessary to sink down to the so-called concealed coal measures, overlain by more recent formations, many of which carry vast quantities of water. It is a question what quantity of water may be regarded as necessarily involving the use of special methods. The largest amount that I know to have been sunk through successfully was at Horden Colliery, where 9,250 gallons of water per minute at a depth of 540 ft. were dealt with. But in many cases far less than this amount has proved an insuperable obstacle to sinking in an ordinary way. It is possible that in the future ordinary methods of sinking may be found applicable in cases in which they have heretofore been considered impracticable, by the use of suspended tubbings, as introduced by Messrs. Haniel & Lueg, of Diisseldorf, a system which has never been tried in this country, although it has been used with much success in several deep sinkings on the Continent.

The special methods to be used will differ, according as the water-bearing strata to be traversed are firm or running ground. In the former case the Kind-Chaudron method is mostly resorted to. This system was first employed about the year 1854, since when about eighty shafts have been sunk by it, of which five have been in this country. A pair of shafts were put down by it in 1876 at the Cannock and Huntington Colliery in Staffordshire, but the operation failed through the breaking away of the ground at the moss-box in each shaft. Sinkings by this method were also commenced, almost simultaneously, in 1877 at the Whitburn Collieries, and were completely successful. Finally, the shaft of the Consolidated Kent Collieries Corporation, Limited, was sunk by this method at Dover in 1904-5. A depth of 1,095 ft. had been reached by ordinary methods of sinking, and there still remained some 90 ft. of Oolitic and Liassic rocks to sink through in order to reach the coal measures. This was done by the Kind-Chaudron method, and a column of tubing 1,120 ft. high was put in. The modern continental practice in such cases is not to put in a column of tubing the full depth of the shaft, but to lower a length of tubing corresponding only with the depth bored out, and fitted with a false cover as well as a false bottom, the economy being obvious.

Mr. Riemer, one of the leading German authorities on sinking, considers that it will not be possible to execute sinkings by the ordinary Kind-Chaudron method at depths exceeding 2,000 ft., because it would be impracticable to handle cylinders thick

enough to withstand the pressure corresponding to that depth, the limit of thickness according to him being about 5 in. Various proposals for modifying the method, so as to render it available at these great depths, have been put forward by Mr. Riemer and Mr. Thomson, but have not yet been put into practice.

When the ground to be sunk through is running as well as water-bearing, the Kind-Chaudron method is no longer applicable. In such cases three groups of methods are recognised—namely, (1) driving down annular sheet piling, (2) forcing down continuous cylinders of brickwork or iron, and (3) the Poetsch freezing process. Annular piling is the oldest method, and is only applicable when the stratum of soft ground to be passed through is not very thick, and is comparatively near the surface. As the piling is usually in lengths of 15 ft. to 20 ft., and as each new ring of piles has to be driven inside the one above, it follows that the original diameter must be taken very much greater than the finished diameter of the shaft. The method is not often seen now, but a good modern example may be quoted—namely, the recent sinking (September, 1906) at Bowburn Colliery Co., Durham.

It may be here suggested that ferro-concrete sheet piling might be used. Piles of this material, 60 ft. in length, have been driven, and it has also been found practicable to add fresh lengths to the top of piles already driven, and thus to get down to almost any desired depth. Furthermore, the adjacent sides of ferro-concrete piles can be grooved, and when the piles are down, cement can be run into the spaces thus left, making the piling practically watertight. As far as I know, this system has never been tried for shaft-sinking, but there appears to be no reason why it should not be perfectly successful.

The method of sinking continuous cylinders, and excavating the ground from inside them, has only been used in this country for moderate depths. There have been several good examples of shallow sinkings by it in Scotland, where sand, mud, and silt on the seashore have been sunk through for depths of about 100 ft., the sinkings at Musselburgh (1901–2), and at Bridgeness (1878), being cases in point. In a more recent sinking of a pair of shafts at Ardeer (1905), about 80 ft. of silt had to be traversed, and a similar process was adopted, using, however, the somewhat antiquated device of the air-lock (first used by Triger in 1839) for the actual excavation, the men working in compressed air at the bottom of the cylinder. Modern practice seems generally to favour the use of some form of dredge for excavating the material from the inside of the cylinder.

This principle has reached its greatest development in Germany, where it has been extensively used for sinking down to firm ground through the overlying thick masses of superficial deposits. The modern practice consists in sinking a cylinder, usually of masonry, built upon a massive iron cutting shoe, until skin friction prevents its further descent. The bolts, attached to the cutting shoe, are built into the brickwork, and by means of these a heavy iron anchor ring is secured to the upper part of the shaft. A cylinder of iron is then built up inside the first cylinder, and is forced down by powerful hydraulic presses, which work against the anchor ring. The ground inside the cylinder is excavated by some form of dredge; grab dredges, bucket dredges, bag dredges, and the air lift all having been made use of.

The freezing method was devised by F. H. Poetsch in the year 1883. Although originally intended for sinking through wet quicksands, it has also been applied to sinking in solid but broken strata, carrying

much water. Although it had been used repeatedly with success in both France and Germany, the first applications of this method in Britain have been comparatively recent, and have all been confined to the Durham coalfield. The first example was the sinking of a pair of shafts at Washington in 1902. Freezing was subsequently employed in two very difficult sinkings at Easington and Dawdon. At the former place the operation was a failure; the yellow sands here are exceptionally thick—namely, over 100 ft.; they lie at 465 ft. beneath the surface; the water is brackish, and the water level is affected by tidal movements. At the neighbouring colliery of Dawdon, however, the operations, commenced about the same time, were perfectly successful.

Another very interesting sinking by this method is now in progress at the famous Wearmouth Collieries, originally sunk in 1826. It is now proposed to sink a new shaft close to the site of the old ones. If this were done by the ordinary method, the necessary pumping would relieve the tubbing in the old shafts of the hydrostatic pressure under which it now stands, and when this pressure comes on again, which it would do as soon as the new shaft were tubbed off, the old tubbing would in all probability be destroyed. Hence it has been decided to use the freezing method, which does away with the need of pumping, and, therefore, does not affect the hydrostatic level of the district.

Finally, reference may be made to a method that has only been used once or twice in the north of France—namely, the forcing of cement slurry through boreholes into soft, fissured strata, and thus forming a wall of concrete within which sinking can be performed; this method was used successfully a few years ago at the Lens Collieries.—HENRY LOUIS.—*London Mining Journal* (from Institution of Civil Engineers), July 6, 1907, p. 30. (A. McA. J.)

**BREATHING APPLIANCES FOR MINES.**—Dr. Boycott begins by referring briefly to some physiological considerations which bear on the efficiency of breathing appliances. The common object of such apparatus is to enable the wearer to remain for a varying time in air which either contains insufficient oxygen to support life (*e.g.*, black-damp), or is poisonous (*e.g.*, after-damp) or irrespirable (*e.g.*, irritating vapours). This is attained by connecting the mouth to a closed space into which oxygen is delivered and from which the carbonic acid is removed. The apparatus must be air-tight as far as the passage of air from outside to inside is concerned. In most circumstances met with in mines, a very small leak does not necessarily render the apparatus useless. The outside air must not be breathed, either because it does not contain enough oxygen (black damp), or because it is directly poisonous (after damp). In the former case a small leak is obviously immaterial; in the latter the occasional leakage of a few cubic centimetres of average after-damp (containing 1 or 2 per cent. of carbon monoxide) into an air space of 4 or 5 litres would cause no material symptoms for many hours. If, however, irritating vapours, such as thick smoke, are present even the smallest leak must be avoided.

The amount of oxygen required varies enormously with the degree of bodily activity, and to a less extent with the individual concerned. A man at rest uses about 0.3 litres per minute, while severe exertion raises the consumption to 2 litres per minute; a degree of activity corresponding to walking 3 miles per hour requires rather more than 1 litre per minute.

The volume of air breathed cannot be indefinitely increased, and the breathing of air which is deficient in oxygen is an imperfect stimulus to deeper or more frequent respiration. It is, therefore, necessary that the percentage of oxygen in the air breathed should be maintained as well as the total quantity available for use. It is undesirable that the percentage of oxygen in the air breathed should fall much below that in outside air (21 per cent.), and it is definitely dangerous to breathe a mixture containing as little as 10 per cent. oxygen, especially while doing muscular work. There is no considerable physiological advantage in increasing the percentage of oxygen in the mixture breathed beyond 20 per cent.

The symptoms resulting from breathing air deficient in oxygen are insidious and indefinite, and vary very much with different individuals. The first obvious symptom, however, especially in persons not particularly directing their attention to the matter, may be sudden loss of consciousness. It is, therefore, absolutely necessary that some definite warning of the failure of oxygen, not dependent on personal vigilance or sensations of giddiness or the like, and preferably of an emphatic mechanical kind, should be given to the wearer of a rescue apparatus before the percentage of oxygen has fallen to a dangerous limit. This is best obtained by arranging and working the apparatus so that it becomes filled with a mixture containing 60 or 70 per cent. of oxygen. Before the oxygen percentage has been reduced to the limit of safety, the apparatus will then become so empty and respiration so impeded (or impossible) that the attention of the wearer cannot fail to be attracted mechanically. Any apparatus which permits the possibility of the user being unknowingly led into breathing a mixture containing less than 12 per cent. of oxygen must be looked upon as extremely dangerous.

In all cases a moderately distensible bag is necessary to act as a reservoir and as a buffer or cushion for the ebb and flow of inspiration and expiration. Such a bag must be so large that the deepest inspiration or expiration may be contained in it quite easily. Expiration or inspiration against even the slightest pressure (as into a full or out of an empty bag) is extremely distressing, and not free from danger if persisted with.

The physiological effects of breathing air containing an excess of carbonic acid are usually obvious. Air containing 5 per cent. causes definite panting, which at 6 or 7 per cent. becomes distress; even 4 per cent. produces unpleasant increase in respiration (hyperpnoea), and greatly reduces the capacity of doing hard work. It may be stated in a general way that 3 per cent. is the maximum permissible, while it is desirable that the percentage should not rise much above 1 per cent. Definite symptoms of poisoning do not appear—at any rate in the course of a few minutes—until the percentage rises to 10 or 15.

At the end of each expiration the mouth and larger air passages are filled with the last and most impure air from the lungs; the air contained in this "dead space" between the lips and the lungs forms the first part of the succeeding inspiration. Any breathing apparatus attached to the mouth increases this "dead space" by the volume of air between the lips and the purifiers, or the inlet of pure air. In this way a large dead space may render the air actually inspired very impure. Thus with a dead space of 200 cub. cm. containing 6 per cent. carbonic acid, an average inspiration of 600 cub. cm. from air containing 1 per cent. carbonic acid contains, as a whole, 2.7 per cent. carbonic acid, while with a dead space of 400 cub. cm.,

the figure is raised to 4.3 per cent. Generally speaking, therefore, the dead space in the respiration apparatus should not exceed 200 to 250 cub. cm. at most.—Dr. BOYCOTT, Report of Royal Commission on Safety in Mines.—*Iron and Coal Trades Review*, July 5, 1907, p. 31. (A. R.)

## Reviews and New Books.

(We shall be pleased to review any Scientific or Technical Work sent to us for that purpose.)

HYDROMETALLURGY OF SILVER. By OTTOKAR HOFMANN. 336 pages, price \$4. (Hill Publishing Co.)

"While barrel and pan amalgamation have been steadily practised for the recovery of silver, and the lixiviation of silver ores as well, little has been heard of them outside of Mexico and one or two South American countries. Dating from Stetefeldt's hypsulphite lixiviation, very little if anything new has been added to the literature of the hydrometallurgy of silver, which makes this book on the subject decidedly welcome. The subject matter in this book is separated into two parts: the first dealing with the preparation of the ore, or chloridising roasting, and the second taking up the extraction of the silver from ore prepared for lixiviation or amalgamation. The synopsis and the method followed by the author in dealing with the subject, furnish an index of what can be expected in a book, and then the thought arises: will there be anything new or valuable in the work? A few brief remarks will show that the author is experienced, original, and learned in his subject, thus being qualified to teach others and add something new to the subject. At once, the reader finds that sulphide minerals are divided into three classes, according to the readiness with which they will combine with sodium chloride. This is both a new and valuable systematisation of ores. It has been the prevailing belief that certain kinds of coarsely crushed minerals permit of quicker percolation than finely crushed minerals. This the author, with certain qualifications, states is erroneous, and that increased filtration due to coarse crushing is so slight as to be of no practical value. He then shows that finely pulverised ore is in a much better condition for chemical reactions induced by chloridising roasting and lixiviation, and further that chloridised ore permits quicker filtration than raw ore. Professor Christy, some years ago, made experiments in chloridising roasting, in which the loss from volatilisation was found greatest if salt was added early to the roast. Mr. Hofmann says the proper time to add salt is during crushing operations, provided that the nature of the ore will permit, thus mixing it thoroughly. Reasons are given why lead and zinc sulphides require more salt for a chloridising roast than iron and copper sulphides. The author is not only practised in the subject on which he writes, but has made a special study of chloridising roasting, the results of which he has incorporated in his book. In addition to the valuable experience he relates, he gives practical suggestions that add materially to the value of the work from the standpoint of the man engaged or about to be engaged in carrying out the process."—*Mines and Mining*, June, 1907, p. 504. (A. McA.J.)

**THE METALLURGY OF THE COMMON METALS.** By LEONARD S. AUSTIN. First edition. £1 1s. (San Francisco, U.S.A. : *Mining and Scientific Press.*)

This excellent work is divided into ten sections, and deals with the general principles underlying the treatment of ores of gold, silver, iron, copper, lead, and zinc.

Although each, naturally, can receive only a limited amount of attention, the information is clearly put and admirably practical, and the work will be found particularly useful as giving information not obtainable in many more pretentious treatises. Perhaps one may take some exception to the section relating to the occurrence and treatment of gold ores, as it is scarcely so practically treated as the others, but the sections relating to copper and lead are full of useful details, and will prove valuable both to the beginner and to the fully qualified metallurgist.

In addition to the sections dealing with the metals individually, we have special and probably more generally useful sections dealing with generalities such as sampling, fuels, combustion, thermo-chemistry and refractory materials, roasting—a most useful section—and refining. Finally, a section is devoted to the commercial aspects of metallurgy, including notes and advice as to labour and general costs, and a useful schedule of prices ruling at various mining centres for the purchase of ores, concentrates and mattes.

Unfortunately, the work has one inexplicable and absolutely unpardonable fault. It possesses no index. — *London Mining Journal*, July 6, 1907, p. 13. (A. McA. J.)

**PRACTICAL COAL MINING.** Vol. II. By Leading Experts in Mining and Engineering, under the editorship of W. S. BOULTON, B.Sc., F.G.S., etc. Six volumes. First edition. Vol. II. 6s. (London : Gresham Publishing Company.)

Volume II. contains the following sections :—

- (4) Shaft Sinking (continued).
- (5) Breaking Ground.
- (6) Methods of Working and Timbering.

The description of the Ordinary Methods of Sinking are continued, and amongst them excavating by hand is briefly described, the absence of which we should have liked a little more practical detail, but this should, perhaps, be looked for in the pit rather than in the text book.

A separate chapter is devoted to Special Methods of Sinking, and fully illustrated. The special conditions which led to the adoption of these methods might have been more fully explained.

Breaking Ground is an eminently practical chapter on explosives, shot firing, and drilling machines ; but where is the miner's pick and all the other tools whose uses are generally given under this heading? Perhaps they will come later on.

A short and useful chapter is devoted to Driving Stone Drifts ; the subject is well worth it. It is an almost constant occurrence in the life of a colliery, and a very different thing from driving a coal road. Much money is often wasted through not realising this difference, and expecting the ordinary coal hewer to undertake it.

The last chapter in this section deals with Coal-cutting Machines. The main characteristics of the many types of coal cutters now on the market are

well and briefly described with illustrations, and the author's general remarks are excellent.

Section 6 begins with a description of the various Methods of Working, of which there are really only two, with endless modifications adopted to suit special circumstances. The general remarks are very much to the point, but no amount of book reading, without experience underground, can enable anyone to select the method best suited to a particular mine. The student has here, however, a number of actual instances, with the sections and conditions of the mine, and the special method adopted, which cannot fail to be of use to him.

Workings under the sea, beneath running sands, bogs, etc., have received special attention, as also boring against old workings ; both are attended with special dangers, and the author's remarks will be read with interest.

Timbering needs experience and much practice. The possibilities of saving expense without in any way increasing the dangers from falls, etc., are greater than appears at first sight. We realise the difficulties of explaining these details—small in themselves, but which mean so much—within reasonable limits (viz., notching trees and bars so as to avoid splitting under weight, the angle at which timber should be set to carry side or top weight ; in heaving ground, inclined seams, etc.) ; still we think the subject worthy of more notice in a work entitled 'Practical' Coal Mining.' — *London Mining Journal*, July 13, 1907, p. 66. (A. McA. J.)

## Selected Transvaal Patent Applications.

RELATING TO CHEMISTRY, METALLURGY AND MINING.

Compiled by C. H. M. KISCH, F.M. Chart. Inst. P.A. (London), Johannesburg (Member),

(N.B.—In this list (P) means provisional specification, and (C) complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)

- (C.) 298/07. A. H. Gibson. Improvements in coal mining machinery. 12.7.07.
- (P.) 299/07. D. Brazzi. Improved screen for use in gold bearing ore crushing stamp batteries. 15.7.07.
- (P.) 300/07. J. E. Smith. A new and improved locomotive smoke box. 16.7.07.
- (C.) 301/07. F. H. Maxwell. Improvements in stamper batteries. 16.7.07.
- (P.) 302/07. R. V. Driscoll. Improvements in safety devices for mine cages and like vehicles. 19.7.07.
- (C.) 303/07. H. D. P. Huitzer. Manufacture of a drinking vessel made of ice and apparatus therefor. 19.7.07.
- (P.) 304/07. S. S. Lawry. Improvements in apparatus for concentrating loose material containing precious stones or metals. 20.7.07.
- (P.) 305/07. A. R. Wojciechowski. An improved process and means for disintegrating and rendering friable diamond bearing ground. 23.7.07.
- (P.) 306/07. S. Fraser. Stamp battery mortar box ore feeder. 24.7.07.
- (P.) 307/07. J. T. Nicholson. Apparatus for taking finger prints. 24.7.07.

- (P.) 308/07. A. Gemmell. Automatic dry earth closet. 25.7.07.
- (P.) 309/07. F. Slade (1), C. R. Pratt-Smith (2). Automatically regulated gravitation fire escape. 25.7.07.
- (P.) 310/07. A. E. Pack. Improvements in rope jockeys. 25.7.07.
- (C.) 311/07. J. Gill. Improvements in rotary motive power engines applicable also to rotary pumps, air or gas blowers, compressors or exhausters, meters and the like. 26.7.07.
- (C.) 312/07. C. H. Fox. Improvements relating to railway wagons and like vehicles. 26.7.07.
- (P.) 313/07. J. W. Sutcliffe (1), E. W. Norton (2). An improved grip for attaching trucks to a cable used in mechanical haulage. 26.7.07.
- (P.) 314/07. P. T. Morrisby. Improvements in pulp distributors. 27.7.07.
- (C.) 315/07. B. Hall. Process for the extraction of gold and silver from ores. 27.7.07.
- (P.) 316/07. S. Fraser. Improvements in linings for stamp mortar boxes. 27.7.07.
- (P.) 317/07. J. F. Cook. Improvements in means for collecting crushed ore products. 27.7.07.
- (P.) 318/07. J. Conning. An improved joint applicable for use between soil pipes and closet pans or stop sinks and the like. 27.7.07.
- (P.) 319/07. P. A. Chiappero. Improvements appertaining to internal combustion motors or engines. 27.7.07.
- (C.) 320/07. R. Howe-Gould. Improvements in fluid meters. 27.7.07.
- (C.) 321/07. A. H. Warmesley. Improvements in apparatus for producing combustible gas from petrol or other spirit or oil. 29.7.07.
- (P.) 322/07. W. R. Jones. An improved form of brake with automatic powers to prevent the cage from dropping when the rope breaks. 29.7.07.
- (P.) 323/07. A. Harvey. Improved and portable shower bath. 30.7.07.
- (C.) 324/07. H. Undeutsch. Improvements in devices for stopping cages, vehicles and the like in case of rope rupture. 2.8.07.
- (C.) 325/07. C. H. Alston. Improvements in or connected with internal combustion engines. 2.8.07.
- (C.) 326/07. E. Surcouf. Improvements in motor road vehicles and trams. 2.8.07.
- (P.) 327/07. A. M. Bell. Ironing machine. 3.8.07.
- (P.) 328/07. J. Craig. Apparatus for treating diamond bearing deposits, ores and the like with liquids. 6.8.07.
- (C.) 329/07. H. W. Blaisdell (1), H. A. Brooks (2). Improvements in or relating to filter leaves. 7.8.07.
- (C.) 330/07. J. R. Brown. Improvements in or relating to lining for grinding mills. 7.8.07.
- (C.) 332/07. H. Poehl. Improvements in machines for washing coal and other materials. 9.8.07.
- (C.) 333/07. H. Poehl. Improved method of and apparatus for draining washed coal and other materials. 9.8.08.
- (P.) 335/07. J. McIntosh. Improvements in chucks of rock drilling machines and the like. 10.8.07.
- (P.) 336/07. J. McIntosh. Improvements in the valves of rock drilling machines or other reciprocating engines. 10.8.07.
- (P.) 337/07. S. S. Osborn. Improvements in centrifugal amalgamating apparatus. 10.8.07.
- (P.) 338/07. H. Pakeman. Machine for developing motive power with cold water and compressed air. 10.8.07.
- (P.) 339/07. J. E. Wheatley. Movable clamp-bar bracket. 12.8.07.
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