

# THE JOURNAL

OF THE

## Chemical, Metallurgical and Mining Society

OF SOUTH AFRICA.

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### Proceedings

AT

### Ordinary General Meeting, May 16, 1908.

The Ordinary General Meeting of the Society was held in the Chamber of Mines, on Saturday, May 16th, Prof. J. Yates (President), in the chair. There were also present:—

51 Members: Messrs. F. F. Alexander, R. G. Bevington, W. R. Dowling, A. Heymann, A. McA. Johnston, A. Richardson, G. O. Smart, Prof. G. H. Stanley, A. Whitby, H. A. White, Prof. J. A. Wilkinson, W. A. Caldecott, S. Beaton, W. Beaver, G. D. Brown, E. H. Croghan, A. F. Crosse, Hon. W. Dalrymple, M.L.C., G. A. Darling, P. L. Edwards, A. D. Gilmore, J. Gray, B. J. Hastings, M. Inglis, J. P'Ons, H. R. Jolly, J. A. Jones, G. A. Lawson, Hy. Lea, Jas. Lea, W. P. O. Macqueen, G. Melvill, J. E. Metcalf, J. T. Milligan, A. Mitchell, P. Morrisby, B. P. Mosenthal, W. Nicklin, W. J. R. North, L. J. Robinson, W. H. Roe, O. D. Ross, C. E. Rusden, C. B. Saner, H. A. Scarf, A. L. Spoor, H. Taylor, J. E. Thomas, J. F. Walker and J. P. Ward.

14 Associates: Messrs. S. J. Cameron, J. Chilton, C. L. Dewar, A. L. Edwards, B. Ellis, J. H. Harris, C. B. Hilliard, W. H. Jane, H. Rusden, A. Thomas, W. E. Thorpe, W. Waters, L. J. Wilmoth and J. A. Woodburn.

10 Visitors and Fred. Rowland, Secretary.

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

#### NEW MEMBERS.

Messrs. G. Melvill and P. Morrisby were elected scrutineers, and after their scrutiny of the ballot papers, the Chairman announced that the candidates for membership had been duly elected, as follows:—

HENDERSON, JOHN McCLELLAND, P. O. Box 1146, Johannesburg. Mining Engineer.  
JOHNSON, HUGH MICHELL LUTTMAN, Rand Club, Johannesburg. Mining Engineer.

MATHER, THOMAS W., South American Development Co., P. O. Box 336, Guayaquil, Ecuador.  
SMITH, EDWARD, French Rand G. M. Co., Ltd., P. O. Box 92, Luipaardsvlei. Amalgamator.  
SUART, ARTHUR B., Johnson & Son's Smelting Works, Ltd., Finsbury, London, E.C. Bullion Refiner.

THOMAS, ABRAM, Simmer & Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Smelter.  
(*Transfer from Associate Roll.*)

WARWICK, D.S.O., JOHN ABRAHAM, Bushtiek Mines, Ltd., per private bag, Bushtiek Siding, via Bulawayo, Rhodesia. Mining Engineer and Mine Owner.

WILLIS, HARRY THOMAS, Champaign, Illinois, U.S.A. Mining and Metallurgical Engineer.

The Secretary announced that the following gentlemen had been admitted as Associates by the Council since the last general meeting:

GEPP, HERBERT WILLIAM, c/o de Bavay Treatment Co., Broken Hill, New South Wales. Chemist.  
TAYLER, WALKER ALFRED CHARLES, Simmer and Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Cyanider.

The following were then proposed and unanimously elected scrutineers for the forthcoming ballot for officers at the annual election:—

Messrs. Gray, W. E. Thorpe, Morrisby, J. Lea and A. L. Edwards.

**The President:** I do not know whether you have noticed the omission of one name from the ballot paper for the new members up for election to-night—the name of Mr. Lambert. I am very sorry to say that Mr. Lambert died a few days ago at Lydenburg from a chill, followed by pneumonia, and the end came in two or three days. Under the sad circumstances, I move that we send a letter of condolence to his relatives.

The motion was adopted unanimously in silence.

**The President:** The Council, at its last meeting, unanimously decided to make Prof. Hilary Bauerman, A.R.S.M., F.G.S., etc., an honorary member of our Society. He is an honorary member of the Institution of Mining and Metallurgy, London, and, in 1906, was awarded the gold medal of that Institution in recognition of his services to metallurgy. He is one of the

veterans of science, and the Council feels sure that in conferring the honorary membership upon him as a mark of our appreciation of his work, it has your unanimous and warm approval.

The Secretary will now read a letter received from the Kleinfontein mine, with reference to a certain accident which occurred recently in the cyanide works.

**The Secretary :** This letter was received in reply to an enquiry suggested by a member of Council, as to the cause of the accident.

Kleinfontein, Box 88, Benoni,

April 30th, 1908.

"In reply to your enquiry, I have pleasure in sending you the following account.

This accident happened in one of two 30-ton vats, which are used for receiving the fairly clear liquid decanted from the acid treatment tank. The liquid is allowed to settle in these tanks for about a fortnight before being decanted off and run to waste. The small amount of slime which settles is occasionally cleaned up, and on the 20th April, when the accident occurred, one of these vats had already been cleaned, and a boy who had been engaged in cleaning the first tank had started to clean up the second; things went quite smoothly for about half an hour until another boy was sent down to help, immediately after which the accident happened. The boy who was first in the tank feeling himself affected by fumes succeeded in getting out, but the other, who was following, showed signs of collapsing, and a white man and three natives went to his assistance. All the natives were overpowered by the fumes, and two of the native rescuers seem to have met with instant death. These two, with the boy they were trying to save, fell to the bottom of the vat. The white man, although partially unconscious, succeeded in getting away and in dragging with him the third native rescuer, who was brought round by means of ammonia and the application of cold water to his back. After the accident happened there was a strong smell of hydrocyanic acid in the vat, and there can be little doubt, that in some way the second boy had introduced cyanide into the vat, which was decomposed by the slightly acid sludge with the liberation of HCN. We are quite at a loss to surmise where the cyanide came from. As the vat is situated there is no possibility of leakage of solution into it, and the buckets used were specially marked buckets used only for this purpose. The fact that one boy had already been half an hour in the vat without feeling any bad effects would indicate that the cyanide must have been introduced at the time the second boy went into the vat.

The accident was an unusual and unexpected one, as under ordinary circumstances there should be no possibility of there being any appreciable quantity of poisonous gases in the vat in which it happened. The only precaution against the repetition of such an accident which suggests itself to me, in addition to the obvious one of exercising the greatest care in the handling and storing of cyanide, is that of having air pipes led to the vats so that in the event of the accidental liberation of poisonous gases these could be rapidly diluted to such an extent as to be rendered harmless. Since the accident happened we have had this done at the Kleinfontein cyanide works, so that we can blow air into the vats when men are at work in them."

J. K. ERSKINE, Cyanide Manager.

**The President :** I have just had placed before me a letter, addressed to the Secretary, which I will read.

#### METALLURGICAL TRIALS' COMMITTEE.

Dear Sir,

I beg to inform you that a committee has been formed of technical representatives of the majority of the mining corporations on the Rand, for the purpose of developing original suggestions, ideas and inventions (during the experimental stage), for improvements in the treatment of Rand ore. The following corporations are represented on this committee:—

Consolidated Gold Fields of S.A., Ltd., Messrs. H. Eckstein & Co., Rand Mines, Ltd., Farrar-Anglo-French Group, Messrs. S. Neumann & Co., General Mining and Finance Corporation, Ltd., Consolidated Mines Selection Co., Ltd., and Messrs. Lewis & Marks.

I have been instructed to request you to kindly notify the members of your Society of the formation of this committee, and of the opportunity thus afforded for bringing forward suggestions for improvements in ore treatment. The basis of remuneration or royalty payable for any suggestion experimented on will be arranged before the commencement of the trials.

Communications relating to proposed improvements should be submitted in writing to the Secretary.

J. P. LONG, Secretary

I must say that this idea of forming a committee to go into the merits of new inventions, and of proposed improvements, has my hearty approval, as I feel sure it has yours. To me it does seem, that the idea of taking new notions to this committee rather than to individuals is better calculated to secure full justice and consideration for such ideas. We have, on the Rand here, often been accused of indifference

to new ideas, and I remember well that Lord Selborne, in connection with the awarding of the trophy to the Gordon drill, in the recent competition, refuted that accusation. Now, I think, that you will agree with me that the idea underlying the formation of this committee is excellent, and I would remind you that all communications in connection with the committee mentioned are to be addressed to Mr. Long.

**The Secretary:** In connection with the annual election of Officers and the Council, the nominations must be in my hands by the 24th May, and I shall be obliged if members will send them in early.

**The President:** I have just been reminded that Dr. Moir is shortly leaving for home on a six months' holiday. I am sure that with me you wish him a very pleasant trip, and a safe return.

As you are aware, we visited the Nigel mine last Saturday, and I think I am right in saying that those who accompanied the excursion enjoyed themselves most thoroughly. I move that the Secretary be instructed to send a letter of thanks to Messrs. Line and Curnow for their excellent hospitality.

The motion was carried unanimously.

**The Secretary:** I am continually getting applications from men who want engagements as cyanide learners, assayers, etc., and I shall be glad if cyanide managers and others, who are in need of such men, will let me know, so that I can put such applicants into communication with them.

**The President:** I am very pleased to see that we have to-night two very old members here, Mr. J. R. Williams and Mr. Crosse. Your applause at least tells them that they are very heartily welcome. Mr. Crosse has a few words to say.

**Mr. A. F. Crosse:** With reference to Mr. Sinclair Brown's letter, read at the March meeting of the Society, combating my statement made at the September meeting, that nitric acid was used to dissolve the copper out of the base bullion, which, together with the silver dissolved, accompanied the copper down the creek, I wish to say that I fully accept Mr. Brown's statement that the copper and silver were duly recovered and accounted for, and did not go down the creek as I somewhat jocularly remarked. But I do wish to say a few words as to the method adopted by Mr. Brown. There is no possible advantage to be gained in treating gold precipitate containing copper and silver with nitric acid, which is very expensive—especially in an out of the way district. Mr. Scrymgeour, of the

Transvaal Gold Mining Estates, Ltd., Pilgrim's Rest, informs me that cupriferos precipitate is simply treated with sulphuric acid to remove the zinc, and then shipped to London, where it realises a very high value for the gold and silver contents, and a good value for the copper. The final result, from a commercial point of view, is infinitely better than the method adopted by Mr. Brown.

## THE ADAIR-USHER PROCESS.

By ALFRED ADAIR.

In response to the invitation to contribute a paper on the above, I have pleasure in submitting to the members of the Society, an account of the original experiments, and the ideas which sprang from them, judging they would prove of greater interest than a bare record of results.

The first experiments related to the use of umber in slimes treatment, and these shewed the possibility of reducing the time of treatment, which led to the Usher apparatus. The low residues obtained demonstrated the possibility of lowering sands residues by efficient tube milling, and as this material might contain a considerable amount of sand, it suggested a further advance in apparatus. Each of these subjects will be dealt with in sequence.

*Umbur.*—This mineral is found in most countries, and consists of a mixture of oxides of iron and manganese. It is usually soft and pulverulent. The German mineralogical name is Braunstein, or Umbra. The Transvaal mineral occurs all over the dolomite areas. The largest and nearest deposit is found N. of Krugersdorp—sandwiched between the dolomite and its underlying strata. It is a sedimentary deposit but not exactly stratified. The outcrop is 15—20 or even 30 ft. wide, and it extends for many miles both E. and W. As the dolomites in this locality contain 7—8% of ferrous and manganese carbonates, this occurrence of umber is readily explained. The mineral has the character of a chemical precipitate, is in an extremely fine state of division, and contains combined water. The chief impurity is silicious matter 10% to 40%, and it rarely contains less than 30% of moisture, which it retains with great obstinacy. The colour varies from light brown, through purple shades, to a dark rich brown. The depth of colour indicates the greater preponderance of manganese dioxide, but is only an approximate guide to the percentage thereof. No two samples agree exactly in composition, not even when they are in juxtaposition. The following analysis is not an average sample, the proportions of

silicious matter and manganese dioxide being high. The sample was a large one and was dried at 212° F., constant weight being obtained only after 48 hours' drying, the sample losing 31% H<sub>2</sub>O.

	per cent.
Silicious matter ...	38.93
Alumina ...	4.01
Manganese dioxide ...	26.40
Cobalt and nickel oxides ...	.05
Calcium carbonate ...	1.80
Magnesium ,, ...	1.47
Phosphates ...	traces.
Sulphates ...	traces.
Combined water ...	7.66
Ferric oxide ...	19.43
	<hr/>
	99.75

#### Analysis of the silicious matter —

Silica ...	30.29
Alumina ...	6.42
Ferrous and manganese oxides	traces.
Lime ...	.23
Magnesia ...	.56
Alkalis and loss ...	1.43
	<hr/>
	38.93

The variability in the composition of umber is shown by the following partial analysis, made in the Government Laboratories:—

	per cent.
Silica ...	29.54
Iron oxide ...	42.50
Manganese oxide ...	9.40
Loss on ignition ...	16.50
	<hr/>
	97.94

Transvaal umber is particularly sensitive to chemical reagents, which suggests its use in other ways than the one to be described. It absorbs sulphuretted hydrogen, and especially sulphurous anhydride with great avidity, forming the dithionate of manganese in the latter case. It is a powerful deodoriser, and it also yields a very pure spiegeleisen with ease on smelting with lime.

*Initial Experiments with Umber.*—Solutions of ferrous sulphate were prepared, to which lime was added in excess, and then umber, or powdered pyrolusite. The solutions to which umber was added shewed a marked decrease in ferrous hydrate, while the pyrolusite tests shewed no decrease due to the reagent. Cyanide solution to which umber was added shewed no decrease in strength, but became slightly yellow, due to dissolved manganese. After some small tests on slimes, which were favourable, a test on a large

scale was made by adding 1 ton to 140 tons of accumulated slimes. The result showed an extraction of 80% on first transfer, as compared with similar charges of 66%, and the loss of cyanide nil, as compared with 15% without umber. These tests were not considered as quite conclusive at the time, and therefore a rather long series of parallel experiments, 26 in all, was made, for the sake of experience, as well as for the primary object of proving the value or otherwise of using umber.

*Parallel Experiments on Slimes with and without Umber.*—These experiments, illustrated by the accompanying diagrams 1 to 8, were made in two small hand churns, which were placed side by side with the handles inward, so that each received the same amount of agitation. Both received exactly the same treatment, so the results are strictly comparable. Owing to the smallness of the charges—15 litres of pulp in each—it was not possible to make many estimations. The estimation of FeO in the presence of cyanide was attempted, but the results were only comparable with each other, and, being imperfect, are unfitted for publication. The estimation of very small percentages of FeO in slimes, in the presence of cyanide, ferrocyanide, ferric oxide, manganese dioxide, ferrous sulphide, lime and alkali, etc., offers quite a pretty problem in analysis. A reliable method, which could be readily performed, would be very useful.

*The Assays.*—The assays in many cases involved small differences, so the method is given. Each sample for undissolved gold was washed twice, each time in four gallons of water. Each pair of assays was done in duplicate and fused, scorified, and cupelled together, 4 to 5 A.T. being used for each assay, and a button of about 70 gm. lead reduced. This was scorified to 10 gm. before cupellation. If this scorification is adopted, the final results are more trustworthy.

*Cyanide Estimations.*—The samples were filtered off, and tested immediately, 100 c.c. usually being taken.

The results of the more interesting experiments are shown in diagrammatic form. The ordinates give percentages and the abscissæ the hours of treatment. The percentage loss of cyanide is shown by the dotted lines, and the percentage extraction of gold in full lines. The light lines refer to the unnumbered charges (A) and the heavy ones to the umbered charges (B).

*Diagram I.*—Accumulated slimes.  
Acidity 0.193% H<sub>2</sub>SO<sub>4</sub>. Fe as FeSO<sub>4</sub>. 0.474%.  
Lime to each in slight excess. Umber 1½%.  
Ratio of solution to dry slimes 1.2 to 1.  
Ratio of cyanide to dry slimes 0.36 to 100.  
Strength of cyanide solution 0.3%.

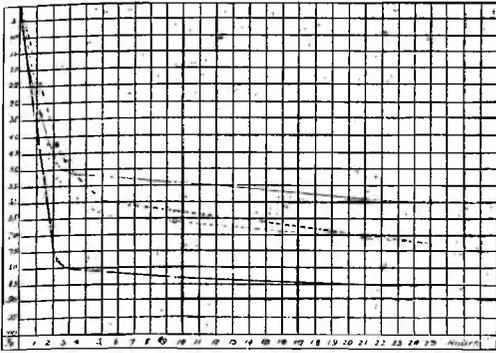


FIG. I.

Diagram II. — Accumulated slimes, same sample as I., but exposed longer to air.  
 Acidity.....? Fe as  $\text{FeSO}_4$  .0081%  
 Lime to each in excess. UMBER 1%  
 Ratio of solution to dry slimes 2 to 1  
 Ratio of cyanide to dry slimes .18 to 100  
 Strength of cyanide solution .045.

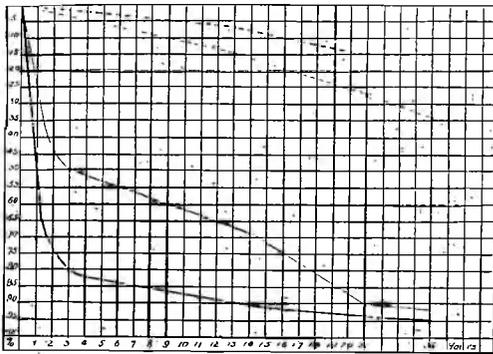


FIG. II.

The solution of the gold, up to three hours, is approximately equal in each case, notwithstanding the difference in the cyanide ratios.

Diagram III.—Accumulated slimes.  
 Acidity .0594%  $\text{H}_2\text{SO}_4$ . Fe as  $\text{FeSO}_4$  .056%  
 Lime to each in excess. UMBER 1%

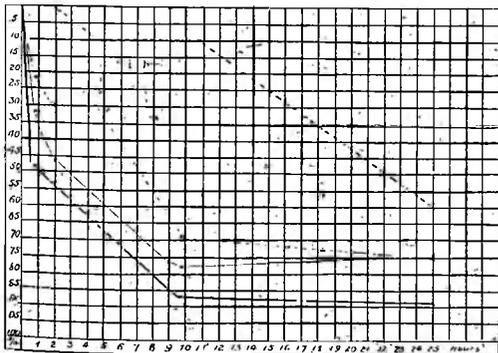


FIG. III.

Ratio of solution to dry slimes 2.27 to 1  
 Ratio of cyanide to dry slimes .0454 to 100  
 Strength of cyanide solution .02%.

The consumption of cyanide in A is apparently due to formation of prussiate between 5th and 9th hours more particularly.

Diagram IV.—Accumulated Slimes.

Faintly acid. Dark grey in colour. Fe as  $\text{FeO}$  .24%  
 Lime to each in slight excess. UMBER 1%  
 Ratio of solution to dry slimes 2.05 to 1  
 Ratio of cyanide to dry slimes .041 to 100  
 Strength of cyanide solution .02%.

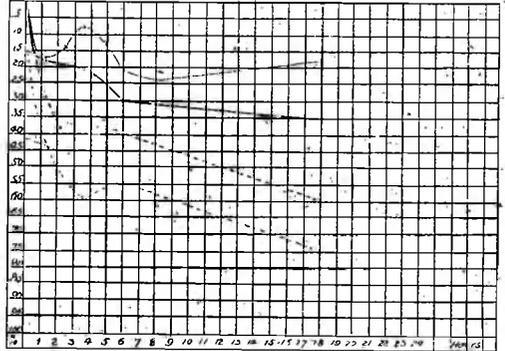


FIG. IV.

This is the most interesting experiment in the series. The amount of cyanide was far too small, but it was fortunate that it was so, as it furnishes direct proof of the reprecipitation of gold in the un-umbered charge.

Another experiment with the ratio of cyanide to dry slimes of .14 to 100 gave the following results:—

hours.	A Cy. loss.	Gold extr.	B Cy. loss.	Gold extr.
1	?	3.4	0	20.38
3 3/4	25	6.23	9	25.47
4 1/2	30	12.27	15	33.02
6	35	13.40	17	35.06
10	45	13.21	25	43.40

The same sample of slimes as the above, was broken up and allowed to oxidise in the air for 14 days, the oxidation being almost perfect. Slight excess of lime was added and 1% umber. Ratio of solution to dry slimes 1 to 1  
 Ratio of cyanide to dry slimes .05 to 100.

Treatment	Extraction.
After 10 hours' treatment 1 to 1	93 %
After 14 hours' treatment	95 %
After 25 hours' treatment	95 1/2 %
After 3 washes, total 4 to 1	96 2/3 %
After 4 washes, total 5 to 1	96 3/4 %

Diagram V.—Current slimes neutral.

Fe as  $\text{FeO}$  .0137%  
 Lime in slight excess to each. UMBER 1%  
 Ratio of solution to dry slimes 2.3 to 1  
 Ratio of cyanide to dry slimes .069 to 100  
 Strength of cyanide solution .03%.

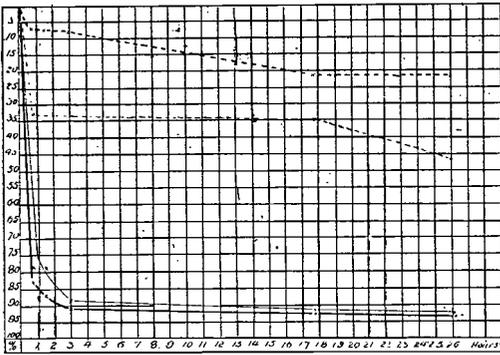


FIG. V.

Diagram VI.—Current slimes neutral.

Fe as FeO. .0384  
 Lime in slight excess to each. Umber .3%  
 Ratio of solution to dry slimes 2 to 1  
 Ratio of cyanide to dry slimes .04 to 100  
 Strength of cyanide solution .02%.

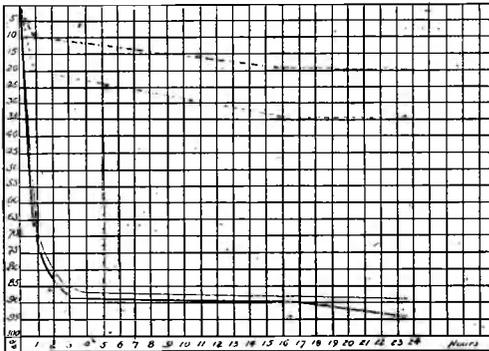


FIG. VI.

Diagram VII.—Current slimes alkaline and oxidised.

FeO traces  
 Lime already in excess. Umber 1%  
 Ratio of solution to dry slimes 2 to 1  
 Ratio of cyanide to dry slimes .032 to 100  
 Strength of cyanide solution .016.

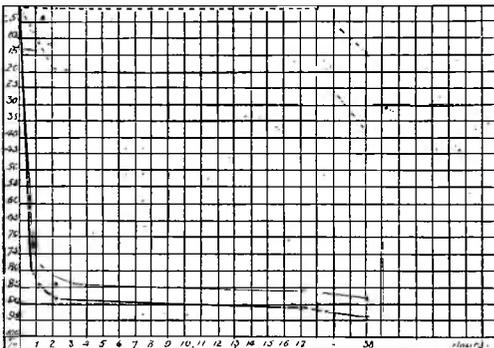


FIG. VII.

As might be anticipated, umber is most useful with the weakest solutions.

Diagram VIII.—Usher apparatus. Current slimes, neutral and oxidised.  
 Fe not estimated  
 Lime in slight excess to each. Umber 1/2% to B.  
 Ratio of solution to dry slimes 2 to 1  
 Ratio of cyanide to dry slimes .04 to 100  
 Strength of cyanide solution .02%.

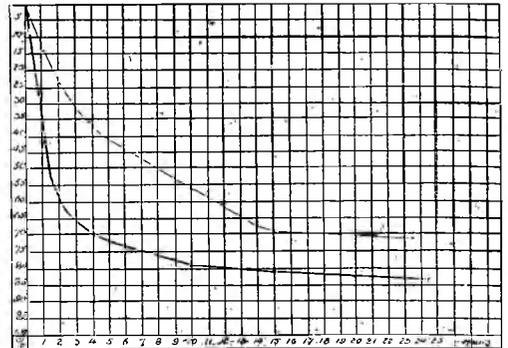


FIG. VIII.

After 15 hours' treatment washed with 4 to 1 of water.

In taking the foregoing results as a whole, whatever the reasons may be, it is clear that the umbered charges have invariably given the best results, and such has been found to be the case in every instance. The addition of umber is beneficial to the extent of from 3d. to 1s. 6d. per ton in gold and cyanide, according as slimes are low or high grade, and as the cost, including royalty, does not exceed 2d. per ton, its adoption will yield a greater profit and add to the security of the process. It is especially useful in reducing the time of treatment, which is now claiming the attention of those who desire to be abreast of the best practice. In attempting an explanation of the reasons for the action of umber, much more research work requires to be done. It has been generally observed, that ferrous compounds always have a detrimental effect on extraction by cyanide, and that aeration, or the addition of oxidising agents, such as  $KMnO_4$  and  $Na_2O_2$ , distinctly improves the process, but at very considerable cost. The ferrous salts in the presence of lime and alkali, form ferrocyanides; in sand solutions of .2 KCN or thereabouts, this is not serious, but with slimes solutions one-tenth of that strength and the iron compounds in a much finer state of division, it is no uncommon experience to find the whole of the cyanide converted, and the gold accordingly mostly reprecipitated, or not dissolved, and a second addition of cyanide becomes necessary.

The ferrous compounds existing in alkaline slimes are the hydrate or the sulphide; the latter is usually small in amount, but present in all the samples, even of current slimes that I have examined. It is the greatest foe the cyanider has to deal with, as shown in experiment 4, where there is a very clear indication of the reprecipitation of gold. These particular experiments were carried out on pure black slimes, a very small proportion of which usually enters the treatment vats. Here umber shows a distinct improvement and its capacity to deal with the usual smaller amounts of sulphide. On ferrous hydrate in the presence of lime the action of umber is to oxidise it to the ferric compound, being itself reduced to  $Mn_2O_3$ , which does not affect the process.

If a charge be thoroughly aerated, it might be expected perhaps that the addition of umber would be without any effect, but such is not the case. Several of the experimental charges were thoroughly aerated, No. 8, for example, and yet the umbered charge not only gave a better extraction, but also conserved the cyanide better. The explanation offered of this effect is, that on the addition of cyanide to an umbered charge, a portion of the manganese dioxide is instantly dissolved. If a 1% cyanide solution be shaken up with umber, it instantly becomes yellow, forming a soluble double compound, which instantly oxidises ferrous salts. This double cyanide is not a manganic but a manganous, or perhaps sesqui-cyanide compound.

*Manganese Double Cyanide.*—I propose to add a few remarks on the manganese double cyanides, as they are of interest, yielding several compounds. By excluding air, a colourless solution is obtained from cyanide and a manganous salt, which on partial aeration turns pink, on further oxidation brown, and finally yellow. On oxidation with chlorine, bromine, or  $Na_2O_2$ , it becomes reddish-brown. On acidification, the colourless solution gives a whitish precipitate with ferrous sulphate, and each of the others blue. The last solution only gives a brown colouration with ferric salts, and all the others blue; it is apparently a mangani-cyanide, corresponding to ferricyanide.

A further examination of these compounds would yield some interesting chemical results, and possibly modify our present conception of the formulæ of double cyanides.

#### THE USHER WASHING APPARATUS.

From the foregoing experiments it is clear, that the time of treatment for slimes can be considerably reduced; but by the old decantation process, it is not possible to do so, hence the necessity for a more rapid process, which has been realised in the Usher plant.

This modification of the decantation process not only shortens the time of treatment, but what is perhaps of more importance, it permits of more perfect washing, and therefore gives lower residues.

In its simplest form, the plant consists of a pipe, leading from the top to the bottom of the ordinary tank, and there branching out into numerous small perforated pipes, which should spray every part of the bottom of the vat. These pipes may be 1 in. in diameter and 2 ft. apart. The holes are so placed, that a slight rotary motion is given to the pulp.

The original form of the apparatus was a central down pipe, 9 in. in diameter, with four arms of 5 in. pipe flattened, with 50  $\frac{1}{8}$ -in. holes distributed along the arm, according to the sectional areas of the vat, and driven by a rope, which passed round a large pulley, fixed to the arms and supported by tie rods. The rope passed round pulleys at the required height, and over a driving pulley placed at the top of the vat.

The simpler arrangement, being cheaper, has been the form usually adopted, but misses one or two advantages of the other system, viz., saving in power, and more efficient treatment. As the object of this paper is to show the experimental side of the question, I leave the large scale experiments to be described by those who have personally conducted them; as I am only responsible for the laboratory experiments.

The first Usher experiment was made in a tub of about 60 litres capacity. The apparatus consisted of a  $\frac{3}{4}$  in. central tube with two horizontal arms, perforated with  $\frac{1}{16}$  in. holes, which were joined to the upright pipes, and extended to the sides at the bottom of the tub, and were revolved at a very slow rate for the first 24 hours, and more rapidly for the remaining 12 hours; 30 lb. of slimes with 30 lb. of .02% cyanide solution were then charged into the tub and the arms revolved. Cyanide solution .02% was then passed through for 36 hours. The results were as follows:—

Original slimes	...	3.34 dwt.
Washed residue	..	.5 "
Unwashed residue	..	1.49 "

The suspension in this experiment was imperfect, and led to channelling, but the main feature aimed at was to get clear solution at the top, while the arms were in motion and solution in circulation; in this we were quite successful. It was evident that the general idea was right, and with some modification the extraction could be improved.

*Usher Experiment, No. 2.*—The same stirrers as in last experiment were used, but stirrers were soldered to the arms.

Charge 1, solution .027% KCy to 1 of current slimes

$\frac{1}{2}\%$  lime and  $\frac{1}{2}\%$  umber.

Time in circulation,  $23\frac{3}{4}$  hours,  $\frac{1}{2}$  to 1 of .025 solution added—

Washing, 3 to 1 of dry slimes in 8 hours.

Results—

Original slimes...	...	...	3.37 dwt.
Washed residue	$\frac{3}{4}$ hour	.94	72% extr.
Washed residue	$23\frac{3}{4}$ hour	.12	96.3% „
Washed residue	final	.10	97.0% „
Unwashed residue	final	.18	94.6% „
First solution drawn		1.115 dwt. per ton	
Last solution drawn		.18 dwt. per ton	

Total solution,  $4\frac{1}{2}$  to 1.

In this case complete success was achieved, as one may say, at a single bound.

*Usher Experiment, No. 3.*—Similar to No. 2.

*Usher Experiment, No. 4.*—

Charge 1 of .025 solution to 1 of dry slimes

$\frac{1}{4}\%$  lime and  $\frac{1}{4}\%$  umber

1 of solution, .025% KCy added in 10 hours }  $30\frac{1}{4}$

4 of wash water " }  $20\frac{1}{4}$  „ } hours

Total solution and wash, 6 to 1

Solution drawn off, 4 to 1.

Cyanide in first solution drawn	...	.024
Cyanide in last solution drawn	...	.003
Alkalinity in first solution drawn	...	.02
Alkalinity in last solution drawn	...	.002
Gold in decanted solution, 4 to 1	...	89%

Assays—

Original charge	...	3.35 dwt.
Unwashed residue	$28\frac{1}{4}$ hours	.18 „ 94.6%
Washed residue	$28\frac{1}{4}$ hours	.10 „ 97.0
Unwashed residue	$30\frac{1}{4}$ hours	.10 „ 97.0

The value of the final wash is very marked bringing the actual extraction up to 97%. The next experiment suggests further possibilities.

*Usher Experiment, No. 8.*—The charge was made up to 3.4 of solution to 1 of dry slimes,  $\frac{1}{2}\%$  lime,  $\frac{1}{2}\%$  umber, and at once set in agitation. After a short time the solution began to clear, and as it was decanted off, it was again put through the pulp continuously for 18 hours.

The charge was then washed with 5 and 3 to 1 of dry slimes in 7 hours. Total solution 8.7 to 1.

Assays—

Original slimes (current)	2.0 dwt.	
Washed residue	12 hours	.20 „ 90.0% extr.
„	18 „	.12 „ 94.0 „
„	$25\frac{1}{2}$ „	.10 „ 95.0 „
„	$30\frac{1}{2}$ „	.025 „ 98.75 „
„	$32\frac{1}{2}$ „	.02 „ 99.0 „
Unwashed residue	$30\frac{1}{2}$ „	.08 „ 96.0 „
„	$32\frac{1}{2}$ „	.03 „ 98.5 „

In all, some 12 experiments were completed. The following notes have been deduced from the experiments:—

1.  $\frac{1}{2}$  solution of .04 gives much quicker results than 3 to 1 of .02 cyanide solution.

2. The time of treatment is reducible to 24 hours, if after withdrawing the decanted solution the balance is discharged to a dam, subsequently recovering the bulk of solution.

3. The less the amount of original solution, the quicker and more satisfactory is the recovery of the gold.

4. The rate of washing depends on the slimes, and the bulk of solution in the vat, also on alkalinity and temperature; in the Usher process the alkalinity is of more moment than the temperature. Other things being equal, the rate of washing an original 2 of solution to 1 of dry slimes pulp:—

$$\frac{\text{Units (in tons) of wash per hour}}{7} = \text{Rate of tons}$$

of wash solution per hours at 3 to 1.

$$\frac{\text{Units (in tons) of solution}}{4} = \text{Rate of washing in}$$

tons per hour solution 4 to 1.

$$\frac{\text{Units (in tons) of solution}}{2} = \text{Rate of washing in}$$

tons per hour.

Thus, if the bulk of pulp contained 400 tons solution to 100 tons of slimes the rate of washing would be 50 tons per hour. The efficiency of the treatment is affected by the temperature. To obtain proper replacement, the temperature of the feed should be some degrees below that of the temperature of the vat.

*Practical Results.*—The figures for the practical work are not in my possession, but I understand that the general result of the partial adaptation of the process shows a distinct gain in time and lower residues, at a less cost.

*New Continuous Process, all Sliming.*—

In our great industry there can be no finality to any process, and one desideratum has always been simplicity, and ever more simplicity, especially in the elimination of labour wherever possible. To achieve this a continuous process eliminating Kafir labour, gives the best hopes of success. Several earnest trials have been made, both here as well as in Australia and America. I need only quote the E. T. Rand and Clancy processes as proof.

The method which I am here bringing forward, differs from any other in some essential particulars. As it has been shown in the experiments, and in actual practice, that a much better recovery is obtainable from slimes than from sands, it follows if the cost of reduction of sands to fine sands or slimes can be effected for the difference in cost between the two processes, then the extra gain in gold would be just so much extra profit.

My proposition is to put in more tube mills, grind finer, and treat the resulting pulp in one

plant instead of two. The present difference in working costs between the sands plants and the slimes plants, is approximately 9d. to 1s. per ton, due largely to native labour. The present price of tube milling, reducing the sands to 90 mesh or under is variously given at from 4d. to 9d. per ton. I take it that 60% of the pulp can be reduced to 90 mesh for 5d. per ton, therefore it would cost 7½d. per ton to reduce the whole, but it might cost a 1s. per ton. Without tube milling, the sands residues were on the whole 4s. per ton, and by partial tube milling, reduced to something like 2s. 6d. per ton.

Before the introduction of the Usher process, the slimes residues would average on ordinary mines 1s. 9d. to 2s. per ton, so that there was little encouragement to alter the existing plant, but by the Usher plant the slimes residues of 1s. 9d. to 2s. could be reduced to 6d. per ton, hence a saving of something like 2s. per ton on sands becomes a possibility, even a certainty.

But that possibility is not to be achieved without considerable changes in plant. The Usher plant has very good points, but it has one important defect, it will not treat slimes laden with 75% of sands, nor can it be regarded as a continuous process. I have, therefore, been gradually led to devise a plant to secure the double advantage of continuity and capacity to treat the product referred to—containing, say, 75% fine sands and 25% slimes. This plant consists of the ordinary decantation vats, provided with cone bottoms, and supported on cast iron columns, instead of a solid bed of mason work or concrete. The cost of construction would not be more than the former, if the excavations are taken into account, as no excavations are here required, and the outflow of slimes would be at least 6 ft. above the foundation line. Each vat is provided with an outflow at the bottom, and an overflow at the top. The entering sands and slimes are led into a distributor near the top of the vat, while the solution enters by a solution distributor placed about the middle of the vat. The vats are erected about 2 ft. above each other, rising from the first treatment vat to the last. The number of vats would depend on the richness of the ore, 4 vats in a row would deal with 2½ to 3½ dwt. pulp, 3 with poor slimes.

In operation, we suppose each vat charged with pulp which has settled in the cones, with solution above. The solution circulating from the top vat, say, No. 4, through each vat in succession to No. 1, whence it overflows to the boxes and after passing through, is continuously returned to No. 4. Operations would start by drawing pulp from the settling vat, by means of an injector, mixing it with the required amount of cyanide and distri-

buting the pulp below the surface of No. 1 vat, here it meets with a slow uprising solution from the distributor, which is fed from the overflow of No. 2 vat. As it falls to the bottom it is robbed of some of its gold, and passing the distributor gradually descends in the cone, as the injector transferring to No. 2 vat withdraws the thick pulp at the bottom. The same process is then gone through in the other three vats, and in the final vat the thick pulp can be diluted with water, and washed to the dam. The peculiarity about the process is the settlement in the cone by which the pulp is greatly freed from water, and thus carries over the least possible amount of dissolved gold to the succeeding vat. The water consumption can be reduced to a minimum. I have had no opportunity of testing the process on a large scale, but the small experiments point to a successful issue. The plant would be arranged so as to vary the conditions, as required, and I expect to reduce the number of vats to three 30-ton vats, which would treat 150—200 tons per diem. The sands plant would be entirely done away with.

*Notes on the Assay of Slimes from results by J. A. Jamieson, lately Assayer to the Adair-Usher Process Co.*—The experimental work in connection with the introduction of the Adair-Usher Processes, necessitated the careful assaying of slimes, particularly residues. In spite of every care, differences did occur, and therefore a series of experiments was projected, to find out the cause of these discrepancies, and some of them were carried out. Unfortunately, the removal of the laboratory prevented the experiments from being completed. The results will, no doubt, interest assayers generally, and they are given here with due apologies for their incompleteness.

The questions to be solved were:—*A.* the loss in fusions, *B.* the cupellation loss, with special reference to slimes residues.

Perhaps some assayer may have an opportunity to check some of the following results and to supplement them. There is no doubt, that an important loss does occur in both operations, which, if it could be recovered, would be an acceptable addition to shareholders' dividends.

The estimation of the cupellation loss presents little difficulty, but the loss in fusions, as far as I am aware, has not been determined with any accuracy, but that such loss does occur, may be verified by re-fusing the slag with charcoal, etc. Possibly the following wet method might give satisfactory results with slimes. It has been successfully applied to rich ores, and to the analysis of cyanide solutions containing gold, yielding in both cases higher results than are

obtainable by fusion. The process for ores is as follows:—

Weigh off 2 A.T. of the very finely powdered ore (if much pyrites is present roasting is necessary). Transfer to a 1,000 c.c. stoppered mixer, add 30 c.c. concentrated HCl and 5 c.c. bromine, allow to stand 1<sup>h</sup> hour, fill up to mark with water and add water to make up for volume occupied by ore

$$= \frac{W \text{ in grammes}}{\text{specific gravity}}$$
 Shake up, allow to stand 15

minutes, shake up again, and filter through a folded filter (thick paper), take 750 c.c., boil off bromide, add Zn powder to precipitate the gold, allow to stand until all the zinc is dissolved, filter off the residue on to a small filter with the filter pump. The small filter should be prepared by previously passing through it 5 gm. finely powdered litharge suspended in water. After filtering off the gold, wash and aspirate until almost dry, fold the filter into small bulk, place on a small scorifier, surround with assay lead, and scorify to 5 gm., cupel and part as usual.

If the bromine is mostly absorbed in the first hour by excess of pyrites, 5 c.c. more should be added and allowed to stand another hour. Low results are obtained, if sufficient time is not given to allow the gold to be soaked out of the ore. If the pores of the small filter be not filled with PbO, finely divided gold passes through and even if the filter is good and PbO is not added, the results are liable to variation, as some of the gold surrounded by carbon (from the filter) escapes absorption by the lead.

The first four of the following experiments were made to show, firstly, if any difference occurred in using 2½ and 4 A.T. charges, and secondly to find out what the loss or gain would be in reducing small or large beads of lead. The fifth batch to find out the advantage or otherwise of quick fusion. A very rich sample of slimes was used in the expectation that differences would be more obvious than on ordinary slimes.

The assays were carried out as follows:—

The whole of the charge was weighed out and transferred to an enamelled can (the popular miners' tea can), provided with a close fitting wood lid, lined with rubber. Half a dozen rapid twirling shakes of the can suffices to obtain a uniform mixture of the contents without loss. New pots only were used, and these were damped inside and rinsed out with borax. Each batch was started in the cold and slowly raised to incipient fusion and allowed to remain in a pasty condition for at least 15 minutes. The furnace was then raised to full heat, and the pots allowed to remain until the contents were perfectly quiescent and transparent. The cupellations were made without previous scorification in morganite cupels,

sufficient silver being added to ensure clean parting, but without breaking up. The partings were completed in 15 minutes with 30% acid free from haloids, the prills washed with distilled water, and annealed. All the fluxes were dried and in fine powder, and as far as possible, the conditions were kept alike with each batch. The fluxes used as also the lead buttons obtained are given in grammes.

#### BATCH I.

No.	Soda.	Borax.	PbO.	Carbon.	Ore.	Pb button	Dwts. per ton.
1	36	12	45	2.5	1 A.T.	27	
2	"	"	"	"	"	28	8.60
3	"	"	"	"	"	30	8.80
4	90	30	"	"	2½ A.T.	38	9.00
5	"	"	"	"	"	34	8.88
6	"	"	"	"	"	34	8.72
7	144	48	"	"	4 A.T.	40	8.70
8	"	"	"	"	"	35	8.65
9	"	"	"	"	"	35	8.85

Time of fusion, 75 minutes.

#### BATCH II.

1	36	12	45	2.5	1 A.T.	37	8.50
2	"	"	"	"	"	39	8.80
3	"	"	"	"	"	37	8.70
4	90	30	"	"	2½ A.T.	39	8.80
5	"	"	"	"	"	39	8.80
6	"	"	"	"	"	36	8.80
7	144	48	"	"	4 A.T.	27	8.62
8	"	"	"	"	"	36	8.75
9	"	"	"	"	"	34	8.75

Time of fusion, 65 minutes.

#### BATCH III.

1	36	12	88	4.5	1 A.T.	71	8.30
2	"	"	"	"	"	72	8.30
3	"	"	"	"	"	74	8.50
4	90	30	"	"	2½ A.T.	70	8.96
5	"	"	"	"	"	67	9.04
6	"	"	"	"	"	69	8.96
7	144	48	"	"	4 A.T.	62	9.00
8	"	"	"	"	"	60	9.00
9	"	"	"	"	"	63	8.90

Time of fusion, 65 minutes.

#### BATCH IV.

1	36	12	88	4.5	1 A.T.	79	8.70
2	"	"	"	"	"	80	8.50
3	"	"	"	"	"	79	8.80
4	90	30	"	"	2½ A.T.	76	9.00
5	"	"	"	"	"	74	9.08
6	"	"	"	"	"	70	8.80
7	144	48	"	"	4 A.T.	68	8.80
8	"	"	"	"	"	67	8.67
9	"	"	"	"	"	67	8.90

Time of fusion, 50 minutes.

BATCH V.							
1	90	30	58	2.5	2½	A.T.	50 8.64
2	"	"	"	"	"	"	46 8.76
3	"	"	"	"	"	"	47 8.64
4	"	"	"	"	"	"	boiled over
5	144	48	88	4.5	"	"	71 8.60
6	"	"	"	"	"	"	boiled over
7	"	"	"	"	"	"	71 8.68
8	"	"	"	"	"	"	72 8.96

Time of fusion, 25 minutes.

These results confirm the general experience that a long fusion with large leads gives the highest and, therefore, most correct results, also that 2½ assay ton charges are best. For very low residues, however, the 2½ A.T. leads should be scorified in pairs, otherwise the prills are difficult to weigh correctly. Further experiments would be required to show why the 2½ A.T. charges come out highest.

*Cupellation Losses.*—In the cupellation of bullion only a small amount of lead is used to a comparatively large amount of gold, and the cupellation loss is insignificant. But when the amount of gold is minute as compared with the lead as in assaying stones, then the cupellation loss affects the results sensibly.

The following tests were made in the attempt to determine the loss, but are incomplete for the reason given above.

*Preparation of the Test Pieces.*—It is not practicable to weigh off pure gold in the minute quantities required, and evaporation of gold solution on lead foil gave very unsatisfactory results. Some fine gold was therefore dropped into molten lead, brought to the melting point of gold, stirred and quickly poured on to a flat iron plate, and this operation of melting and pouring repeated (perhaps better results would have been obtained by further repetitions of this process to obtain a perfectly uniform alloy). Two such alloys were prepared, 5 gm. being used for each test.

ALLOY A.

	Nominal weight.	
1	5 gm. cupelled and parted	1.2 = .06 mgm f. gold
2	"	1.2
3	"	1.2
4	"	× 1.1 assumed mean
5	"	1.2 1.2
6	" + 25 gm. lead	1.2
7	"	1.1
8	"	1.1
9	"	1.0
10	"	1.1 1.1
11	" + 65 gm. lead	× 1.2
12	"	1.0
13	"	1.0
14	"	× 0.8 1.0

	Nominal weight.	
15	5 gm.	1.0
16	" + 65 gm. lead	× 1.0
17	" scorif. to 10 gm. then	1.1
18	" cupelled and parted	1.2
19	"	1.2 1.16
ALLOY B.		
1	5 gm. cupelled and parted	3.3 = .17 mgm f. gold
2	"	3.4
3	"	3.2
4	"	3.4 assumed mean.
5	"	3.4 3.4
6	" + 25 gm. lead	3.3
7	"	3.3
8	"	3.3
9	"	3.4
10	"	3.2 3.3
11	" + 65 gm. lead	3.2
12	"	3.1
13	"	3.2
14	"	3.1
15	"	× 3.3 3.15
16	" + 65 gm. lead	3.3
17	" scorified to 10 gm. and	3.4
18	" then cupelled & parted	3.3
19	"	3.2 3.30

As before stated, these results are not quite satisfactory, but they suffice to show that there is a cupellation loss, which is by no means insignificant. Thus: assuming the figures as correct then the loss on 1 A.T. charges allowing .02 for 5 gm. lead would be on 30 gm. lead buttons: .12 dwt. per ton; and on 70 gm. lead buttons: .24 dwt. per ton. On 2½ A.T. and 4 A.T. charges this loss would be divided by these figures respectively.

As shown in the pot results, the higher results are obtained by using larger amounts of lead. Hence, if the difference of the cupellation loss is added as a correction between an assay yielding 35 gm. lead and one giving 70 gm. lead it may quite easily amount to 6d. per ton or more on residues.

**The President:** I am sorry to say that Mr. Adair is not very well, otherwise he would have been here to read his paper himself. Years ago I know that quite a number of us, at any rate, were under the impression that the last word had been said in connection with the cyanidation of gold ores. We were all mistaken, and even nowadays new ideas are coming forward very rapidly indeed. This Adair-Usher process is a case in point, and I think both Mr. Adair and Mr. Usher are much to be congratulated upon their persistency, and the courage which they have shown in developing and pushing this process to a successful issue. Now, this particular process is being quite largely used on the Rand, and I do hope

that those of you who are using it will come forward, and benefit not only our members, but everyone along the reef, and off the reef, by letting us have your views and experiences. In conclusion, I move a very hearty vote of thanks to Mr. Adair for his very valuable paper on the new process.

**Mr. J. R. Williams :** I fear I am something like the ancient mariner, in that I have been on the sea for a very long time. It is now 15 months since I have had an opportunity of visiting the mines of the Rand. At the start of the Usher process I took a very considerable interest in it. It gave me the idea from the original drawings, that it was a step in the right direction, and I have no hesitation in saying that it is a very valuable addition to our knowledge. I should like to ask one or two of the workers a question, and then to reserve my remarks till a later stage. My reason is this, that at the starting of the cyanide process, long before the slimes treatment was dreamt of, everyone had some idea of the use of an oxidising agent, and one gentleman rushed across to Pretoria and patented every possible agent. That is a patent that Mr. Crosse and myself have found very useful. We call it the Pharmacopœia patent, because it covers everything. Our own experience on a large scale was very disappointing. When making laboratory experiments, we found that we got results far better with oxidising agents than without them, but in no instance on the large scale did I find that any of these oxidising agents had any effect. Not even to the extent of 0.1% could I trace any advantage. I should like to ask those gentlemen, before making any criticisms on the paper, whether they have been able to get any advantage to extraction by the use of umber?

**Mr. G. Melvill :** I have had a great deal of experience with this particular process, and experiments were carried out on a large scale on the Crown Reef, under the supervision of the Adair-Usher Co. When I saw the results of their experiments on a small scale, I certainly thought them marvellous. They used umber all through their experiments and certainly got some very good results, but after I took over the running of the plant I stopped the umber, and my results were quite as good as theirs. So I came to the conclusion that it would be cheaper to work without the umber. Having about the worst accumulated slimes on the Rand to treat, I tried the umber on a small and large scale, but found I got better results by the ordinary circulation with pump compressed air. I, therefore, came to the conclusion that no benefit was derived by using it.

**The President :** Did you visit the Nigel, Mr. Melvill?

**Mr. G. Melvill :** No, I was unable to attend.

**The President :** I asked that question because I noticed down there they were using permanganate of potash in connection with the treatment of bulrushes. Just alongside the Chilean mill, which they were using to grind up these bulrushes, was a bowl of permanganate of potash, and I presume that they find the addition of permanganate useful as an oxidiser of organic matter.

**Mr. G. Melvill :** I may say that there is a large percentage of bulrushes in the accumulated slimes at the Crown Reef. I have tried permanganate, but it would take such a quantity to deal with the organic matter that, though the extraction would be improved, it would not pay.

**The President :** What were your values?

**Mr. G. Melvill :** About 3 dwt.

**The President :** I may say that the value down at the Nigel is very much greater, about 7 dwt., and it has been as high as 1 oz. In the old days they had 5 oz. ore in their mills on certain occasions, so this high value would allow them to use more permanganate than you could. With regard to this matter, I had better explain that the old accumulated slimes of the dam are overgrown with quite a thick growth of bulrushes, and you cannot get at the slimes without taking the bulrushes, and I assure you, to see the mess that comes up in those cocoa-pans is enough to frighten any ordinary cyanide manager. The process is quite simple. They tip it into a Chilean mill, at the bottom of which is a grating; a constant stream of water carries off the slimes as it is disintegrated by the rollers and passes it through a screen of  $\frac{1}{8}$  in. or  $\frac{1}{4}$  in. mesh. There are boys standing by these screens, and every now and again they pull them out and knock them clear of the disintegrated bulrush matter. It was in this Chilean mill that the permanganate of potash was being used.

#### CYANIDATION IN THE MALAY STATES.

By H. F. LOFTS (Associate).

Perhaps some account of a plant which happens to be the only one in these States, and is also situated on the only working gold mine there, and has also the peculiarity of having shiftmen who are coloured men, may be of some interest to your members. It is attached to the Raub Australian gold mine, situated at Raub Pahang F.M.S.

The reef may be described as a quartzitic, lenticular, interbedded vein, and contains scheelite, antimony, bismuth and arsenic in the form of arsenical pyrites. The current tailings assayed 1.12 dwt. and they were too low grade and too full of antimony to treat. Both weak and strong solutions were tried after the tailings were water washed, and in each case no result was obtainable. However, as shown below, when the concentrates were separated by Wilfley tables, they were treated as slimes and showed a 78% extraction. In fact, the greatest trouble had been experienced in obtaining a payable extraction from the higher grade sands, which had been allowed to accumulate. It was found that the chief cause of trouble was antimony, of which sufficient remained attached to small particles of quartz to make the sands and concentrates very refractory.

The higher grade tailings were successfully treated under the conditions mentioned below:—

Buddlings: Assays, 16.35 dwt.; extraction, 70.5%; residue, 4.81 dwt.; treatment, 4 days; cyanide used, 1.05 lb. per ton; number of solutions run on, 34. This sample consisted of finely crushed sandy concentrates. It showed no acidity or refractory material. The strongest solution used was .07%, and the average considerably below this. The treatment consisted in running through weak washes, which produced 66%, and with second treatment this was increased to 70.5%. The last three washes carried no gold. It should be noted that solution values went up after double treatment was used.

Another sample of buddlings gave the following result:—

Assay value, 16.80 dwt.; tons, 1.71; tons of solution run through box, 82.6; lime used, 90 lb.; strength of KCy solutions, .05, .13, .18%; sump, .21% KCy; the precipitation was very good, sump often being nil on assay; final residue, 3.04 dwt.; extraction, 82%.

#### Results on Concentrates from Wilfley Tables.

—These are obtained from the ore now being crushed by battery. The greater part of the sample would pass 120 mesh screen and is black in colour, consisting of a large proportion of heavy metallic particles and much iron. The original assayed 22.40 dwt. Treatment was carried on as under:—

Sept. 26, water wash (neutral), Sept. 27, second wash (neutral). Agitation carried on with 1% KCy solution showed the presence of sulphides with acetate of lead test. This was continued until Sept. 29, Sept. 30, aerating followed by solutions .09% and .06% KCy. Continued until Oct. 2, then, second treatment and oxidation from 7 a.m. to 10.30 a.m. Solution then came off free from sulphides. The strong solution was then run off and a weak solution added.

Weak washing was carried on till Oct. 6, when two samples of slimes were taken and washed, the result being a residue of 8.17 dwt. It was found that by ordinary agitation 32.5% was extracted up till the 30th. Then the extraction stopped. A new system was started, which included aeration, oxidation and agitation, similar to that produced by a centrifugal pump. This brought the second residue down to 9.18 dwt. or 59% extraction, and finally to 8.17 dwt. or 63.5%. On a working scale, this extraction was brought up to 68%.

I now append details of another experiment on concentrates:—Assay value, 22.96 dwt.; length of treatment, 7.33 days. The product consisted of concentrates from the present mill pulp and contained a large quantity of antimony. The first two solutions carried no gold, showing ferrous hydrate or some other oxygen absorbing substance. Continuous washings with weak solutions and afterwards with .35% KCy were very effective and gave an average of 2 dwt. of gold per ton until extraction stopped. A second treatment was resorted to, and the final extraction was as high as could reasonably be expected. The solutions came off clear, and the leaching was good. As there was no acidity no lime was added. I think caustic soda might with advantage be used to dissolve organic matter and avoid a heavy loss of cyanide.

This experiment was a sands treatment on 100 lb. of ore. Final residue, 1.14 dwt. = 95% extraction. The extraction obtained from 372 tons of concentrates and buddlings treated equalled 77.1%, average number of days, 7.77; average charge, 9.41 dwt. The highest extraction being 93% and the lowest 69.8%. These results were obtained from a small plant with a capacity for slimes equal to 200 tons per month. The above plant, through anything but large, is sufficient to show what results could be obtained with a plant of greater capacity.

*Precipitation.*—The solutions in use on concentrates which assayed from 10 dwt. to 38 dwt. and carried from .18 to .05% KCy after passing the boxes varied from a few grains to nil. With lower grade slimes, which contained free and fine gold, solutions were used down to .001% KCy, which gave complete precipitation in the zinc boxes. In these cases the zinc was coated with acetate of lead. In dealing with stronger solutions no acetate of lead was necessary, as good extractions were obtained. The acetate of lead had the effect of making the bullion very impure. Sodium cyanide was used, the average consumption being 1.60 lb. per ton. Large quantities of a white precipitate occasionally formed in the boxes. When this occurred it was found to be necessary

to increase the strength of the solutions by adding solid cyanide at the head of each box. This dispersed the precipitate, which only formed when a weak solution was used. The cyanide solution became at times very foul, and experiments showed that it was advisable to increase the alkalinity by adding caustic soda to the sump and solid cyanide at the head of the box. To ensure a good precipitation an average of 80 lb. of lime was placed in the vat with each charge of 10 to 12 tons. With solutions of the strength mentioned in the preceding paragraph, the boxes were kept fairly free from white precipitate.

**Antimony.**—In this connection it is interesting to notice how quickly when powdered, this is acted upon by caustic soda. The gold slimes were of low grade in the extractor. This, I think, is the usual experience, when ores of this nature are dealt with owing to the precipitation of base metal with the gold. Owing to the minute quantities of almost invisible slimes which were not settled with the lime and which coated the zinc and stopped precipitation, it became necessary to introduce sand filters, which were very effective.

**Agitation.**—It was found that a long agitation was not an advantage; a short agitation lasting one hour producing a solution as rich in gold as one which lasted five hours. The usual proportion of slimes to solution was 1 to 3.

**Tonnage Estimates.**—I found that, if after decantation I measured the cubic feet of wet slimes in the vat and divided by 30 (= cubic feet per ton of slimes) and deducted 33% of moisture, it gave an excellent agreement with samples taken when the vat was full and in agitation.

**The President:** I move a vote of thanks to Mr. Lofts for his paper, and in doing so, I would say that it always gives us special pleasure to receive papers from members oversea.

## NOTES ON TIN MINING IN CAPE COLONY.

(Read at December Meeting, 1907.)

By HARRY D. GRIFFITHS, A.R.S.M., Wh.Sc.  
(Member).

### DISCUSSION.

**Mr. J. R. Williams:** I would like to make a remark on this paper. I have heard it repeatedly stated, especially lately, that the assayers of the Witwatersrand, in fact in the whole of South Africa, are very indifferent about tin, and that a lot of them fail to find tin, when there is plenty. I remember a case when a sample of so-called tin was brought to me. The man was a Cornishman, at least, I think so, and he told me that the sample

contained 70% tin, and that he had two certificates to the effect that there was no tin in it. I thought that a Cornishman should know tin when he saw it, and so I asked my assistant, who told me there was no tin present. I thought I would try it myself, and I applied a method which, I think, never fails to give you tin, when present in the ore. I have spoken lately to one or two assayers concerning this method, and strange to say, it had escaped their memory. What this Cornishman took to be tin was undoubtedly a form of titanite iron. When crushed and on the shovel it certainly looked very much like tin.

The method is simply one of reducing to metal, by ignition with gas or hydrogen. Then it is very easy to dissolve it with hydrochloric acid. Many years ago I was asked to examine this method, and I did so by adding tin stone of a known value to all the samples. It always gave very accurate results. Mr Caldecott informs me that Beringer gives the method, and I think it will prove a good check on any method they may use for these obstinate samples.

**Mr. J. Gray:** With reference to this method of assaying, I have tried it and find that the essential part consists of getting the ore into a state of very fine division. Parry's book deals with this matter very fully, and he recognises it as the best method, though there is some doubt about it.

**Mr. A. Whitby:** This method has always been looked upon as the final court of appeal. It has been supposed to be the most perfect method of tin assaying. I have found hydrogen more effective than local coal gas in reducing the tin. After reduction and solution in hydrochloric acid the process branches in several directions, some operators precipitating the sulphide and weighing the oxide from it; others titrating with iodine, and others again reducing with zinc to a sponge of metallic tin, oxidising with nitric acid, and weighing the resultant oxide.

## THE ORIGIN OF THE GOLD IN BANKET.

(Having reference to Prof. J. W. Gregory's paper on the same subject, read before the Institution of Mining and Metallurgy.)

(Read at January Meeting, 1908.)

By J. S. CURTIS (Member).

### REPLY TO DISCUSSION.

**Mr. J. S. Curtis:** In regard to Mr. Milligan's observations of February 15, I am quite of his opinion that the quartz veins could have no influence on the deposition of the gold.

I have to thank Mr. T. Lane Carter for his flattering criticism of my paper, and I agree with him in what he says about Prof. Posepny of Bohemia. Still, able as he is, he can scarcely be called "the father of the modern science of ore deposits," as that title undoubtedly belongs to Bernhard von Cotta (Professor of Geology in Freiberg), who, in the year 1853, published *Die Lehre von den Erzlagertstätten*. I had the pleasure of attending his lectures at Freiberg in 1867 and 1868, and can bear witness to the great reputation he had as a mining geologist. I believe he was the first writer, who combined the science of geology with a practical knowledge of mining.

I can quite understand that Prof. Posepny, from the limited information he had at hand when he published his "Ore Deposits," formed the opinion that the gold of the Rand was placer gold. I had also formed the same opinion in Barberton before I came to Johannesburg in 1887, but soon had reason to change it. In 1886 I had the pleasure of corresponding with Prof. Posepny in reference to my work on the "Silver Lead Deposits of Eureka, Nevada" (VII. Monograph of the U.S. Geological Survey), and regret very much that after I came to South Africa I was unable, through want of time, to keep it up.

As regards "the bigger the pebble the richer the ore," what I wished to convey was that, as a rule, a gold bearing banket reef was richest in those places where the pebbles were largest, not that a reef carrying large pebbles should necessarily be a good gold producer.

I quite appreciate Dr. Moir's remarks on the instability of gold compounds, and also what he says about the "critical temperature" of water, but I think that pressure would have a very marked influence in keeping the numerous compounds in solution, even at a temperature above 330° C. The rock at the time the depositions took place could be compared to a leaky boiler in which it might nevertheless be possible to keep up a very considerable pressure.

ASSAY OF BATTERY CHIPS AND SCREENS.

(Read at February Meeting, 1908.)

By LESLIE J. WILMOTH (Associate).

DISCUSSION.

Mr. H. R. Jolly: In offering a few remarks on this paper, it is not my intention to theorise, but to give a few notes embodying the practical results of some experiments carried out at the Robinson Deep on this class of material.

The samples of battery chips received, were divided into three grades:—

1. Fines — 20 mesh.
2. Intermediates + 20 - 10 mesh.
3. Metallics + 10

The last measured in some cases about 1 in. in diameter, and were only slightly coated with a film of oxide, which could be easily removed by rubbing between the fingers. I may state that we have tried the author's method of assay, and find it to be very unsatisfactory indeed, in spite of the fact that we followed his directions, and were careful to wash the slags down with litharge and carbon. It is only fair to state that his slags were fluid enough, but the real difficulty presented itself when we tried to detach the lead buttons from the matte. In many instances the lead was so brittle, that it broke away from the matte in pieces, and the cupels showed considerable quantities of iron oxide after cupellation, leading to the conclusion, that there must have been a large quantity of iron or iron sulphide carried down with the lead. The mattes, from six assays of our fines, were then run down together and were found to contain 72.0 dwt., equal to a loss of 12.0 dwt. of gold per assay. The slags only contained sufficient gold to account for a loss of 2.0 dwt. per assay. Three other assays by this method showed losses in the mattes only equal to 1.0 dwt. per assay, but the slag loss amounted to 6.0 dwt. per assay. In these cases, however, the mattes formed were very small.

While treating a sample some 12 months ago we discovered that sodium bisulphate was very useful in fluxing iron, and conducted a number of experiments with it, the following fluxes being the nett results of our investigations:—

	1	2	3
	Fines.	Intermediates.	Metallics.
Charge	1 A.T.	1 A.T.	1 A.T.
Bisulphate	$\frac{1}{4}$ "	$\frac{1}{2}$ "	$\frac{3}{4}$ "
Soda	$\frac{2}{2}$ "	1 "	1 $\frac{1}{2}$ "
Silica	1 $\frac{1}{2}$ "	1 $\frac{1}{2}$ "	1 $\frac{1}{2}$ "
Borax	1 "	1 "	1 "
Litharge	1 $\frac{3}{4}$ "	1 $\frac{1}{2}$ "	2 "
Nitre	1 gm.	2 gm.	4 gm.
Wash	$\frac{1}{2}$ A.T. PbO	$\frac{1}{2}$ A.T. PbO	$\frac{1}{2}$ A.T. PbO

carbon  
Wgt. of Pb. 50 gm. 40 gm. 60 gm.  
Average A.V 26.0 oz. 23.0 dwt. 18.00 dwt.

The various charges were run down in H pots, well packed round with coke, the furnace being urged to its fullest extent. Immediately the fusions settled down, the washes, as stated, were added, and the pots kept at a high temperature for other 20 minutes. When the furnace is going well, one hour is found to be sufficient for the fusion. Bubbles of gas come up before the wash is added, but disappear afterwards. The slags

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were all very fluid, clean and glassy, and in all cases the lead buttons were soft and malleable, and broke away from the slags nice and clean. The cupels showed no signs of iron having been present.

The reason for only using litharge as a wash with the intermediates and metallics is that, in these cases, there is generally sufficient sulphur reduced from the bisulphate to act upon the litharge. At the same time, we have found it advisable not to use too much nitre, as an excess does not reduce the size of the buttons but merely acts corrosively. Some nitre, however, is essential in preventing the reduction of the bisulphate to sulphide, which tends to pass sulphur into the buttons. Our slags from the fines were found to contain gold equivalent to an average loss of 2.0 dwt. per assay, which appears to be a very decided improvement on the matte method of assay. As a severe test of our method we ran two fusions, one of which consisted of six pieces of iron, and the other of eight, to the assay ton and found no difficulty in getting good fusions with homogeneous slags.

It is advisable to put on a trial assay, as slight alteration in the fluxes may be necessary for different samples, mainly in the litharge and soda. The litharge we use is in a higher state of oxidation than the ordinary yellow kind, and is really a mixture containing red lead. The formation of a matte at any time in gold assaying is, in my opinion, bad practice and should be guarded against. In fact, I do not quite see how it is possible to form a matte, which will be gold free. The simpler the products of a fusion are the better.

With reference to the assay of battery screens, I quite agree with the author's preparation of the sample by oxidation and crushing of the oxide formed, as more concordant results will be obtained by that method than by assaying direct. I disagree with him, however, in his practice of forming a matte in the assay of the oxide. The fluxing of the oxide is a very simple matter indeed, and one which can be treated almost like an ordinary assay. As far as I am aware, sodium bisulphate has never been put to any use in the assay of gold before, and I think that there is a possibility of its finding a more extended application as a flux.

In conclusion, I wish to draw attention to the fact that there are small quantities of the metals of the platinum group present in the samples of battery chips, as also in black sands.

**Mr. H. A. White:** There is one point in connection with Mr. Jolly's contribution to this discussion, that I should like to draw your attention to. The peculiar flux recommended contains bisulphate of sodium and sodium carbonate in

the same charge. This cannot be defended. If the intention is to add sodium sulphate (neutral), this round about method is not scientific, for the carbon dioxide evolved can be of no service and merely increases the chances of loss through boiling over or spitting. The appended equation will render my meaning clear

$$2\text{NaHSO}_4 + \text{Na}_2\text{CO}_3 = 2\text{Na}_2\text{SO}_4 + \text{CO}_2 + \text{H}_2\text{O}.$$

I do not undertake to say that  $\text{Na}_2\text{SO}_4$ , which apparently remains unaltered during the fusion, will have any useful effect in such assays as are quoted.

**Mr. A. Whitby:** It was not my intention to contribute anything to this discussion, for the reason that not having had any opportunity to carry out experiments with the author's method of assaying, I did not feel justified in actively criticising it. But there is a somewhat interesting point brought out in Mr. Jolly's contribution, which may be worthy of remark. It may be said that the author's and Mr. Jolly's methods represent the two extremes of our conception of fluxing, the former removing the iron as a sulphide and the latter as a sulphate. Although the first does not say how much litharge he uses, it is evident from his remarks that he adds a quantity merely sufficient for his button and, in the added sulphur, the iron of his charge, the nail and the charcoal, has a great preponderance of reducing agents. The second uses no charcoal except in his wash for fines, and, with the sulphuric acid of the bisulphate the added nitre and the oxygen of his litharge, appears to have an excess of oxidising agent, merely relying on the iron of his charge for the production of a button of lead. These methods are so diametrically opposite in character that one finds difficulty in conceiving that they should arrive at similar conclusions. Many years ago in this Society,\* one assayer recommended a flux of carbonate of potash, litharge, borax, salt, excess of reducing agent, and an excess of iron in the fluxing of pyritic ores as a means of preventing the carrying of silver and gold values into the slag. This was on the assumption that a ferrous potassium sulphide was formed, but I have not found any further reference to this conception of fluxing, and my own opinion about the assay of moderately pyritic ores is quite antagonistic to such a method. And this inclines me to think that in the generality of cases the author's process must fail, for the following reasons.

I wish to definitely state here the position taken up by me with regard to this matter. Whether it is the carrying of iron or of sulphur into the slag, any condition which disposes the slag to retain alkaline sulphides seems to me to

\* See this Society's Proceedings, vol. ii., p. 601.

imply the retention of gold, since the metals of the platinum group share, in some measure, the property possessed by those of the tin group of forming sulpho-salts. By observation and analogy, it is rare to find a matte entirely free from the precious metals. It is difficult to so proportion the reducing agents, that each shall fulfil its own particular duty without interfering with the others, an excess of sulphur, for example, implying lead sulphide in the button whilst the iron nail, according to my experience, when working at high temperatures, has an inconvenient habit of melting into the lead button with disastrous results on cupellation. Further, it is the exception rather than the rule to find a well defined line of demarcation between the button and the matte.

Mr. Jolly's method seems to offer a more reasonable solution of the difficulty presented by this class of material, and is quite in accord with the principle of assaying ores containing pyrites, where the sulphur, in my opinion, is best oxidised to the sulphate state. A little comparison will show, that in this case also, practically all the lead is reduced.

In reference to Mr. White's remark, one would certainly premise, that he is correct in his statement, but the facts appear to be that Mr. Jolly has tried the neutral sulphate without the beneficial effect of the acid salt. In making acid sulphate fusions of iron ores, although this is scarcely parallel, I have observed, that solution of the iron only takes place at a temperature when presumably all the extra sulphuric acid has been driven off, whilst even with a full red heat the fumes of  $\text{SO}_3$  are still noticeable. It might, therefore, be assumed, with some little justification, that the acid sulphate scarcely enters the field until the alkaline carbonate has been, to some extent, satisfied by earlier reactions.

From this point of view I cannot agree with the author's treatment of his roasted product from battery screens, and should certainly prefer to carry the iron into the slag, as silicate and borate, to going out of my way to form a matte with it. With particular reference to the matte assaying of cupriferous gold-bearing ores, I would call the author's attention to Peters' observations on this point in his principles of copper smelting. As there expressed, it is the opinion of the great majority of metallurgists that the existence of very small amounts of copper in the ore is sufficient to carry the gold values into the matte in matte smelting, since copper possesses a remarkable affinity for gold. I do not here mean to imply that the author did not recover the whole of his gold in his experiment with copper ore, but merely to point to the possibility of failure under dissimilar conditions.

## WESTRALIAN WET-CRUSHING PLANTS, WITH SOME NOTES ON LABOUR EFFICIENCY.

(Read at February Meeting, 1908.)

By GERARD W. WILLIAMS, F.C.S., A.I.M.M.

### DISCUSSION.

**Mr. F. F. Alexander:** The author's description of the general practice obtaining in Westralian wet crushing plants is certainly most interesting to most of us, who knew him here, but I certainly consider him unkind, to say the least, in recommending for our notice such time honoured abuses as inside amalgamation, grinding pans, and 3 ft. 3 in. tube mills, to say nothing of the apparent undermanning of the plant at the expense of a reasonably good extraction. I believe inside amalgamation is still practised on some of our out of the way plants, and the advantages claimed for it is a myth. In other words, by neglecting adequate plate dressing for the constant feed of mercury and contaminated gold from the mortar boxes you are losing the substance for the shadow through faulty amalgamation. I cannot agree, that a tube mill is essentially an impact grinder. That impact assists within small limits is a reasonable assumption, but I certainly class the tube mill with attrition grinders, I would go further, however, by classing a 3 ft. 3 in.  $\times$  13 ft. mill as a worthless installation—the load in so small a mill, not being heavy enough to give full crushing capacity. Neither is the inclined plane or race of pebbles long enough to do the work, and I feel sure a 6 ft.  $\times$  18 ft. 6 in. mill with peripheral discharge would yield results which would astonish the advocates of grinding pans. I will not attempt to defend a possible misuse of unskilled labourers on these fields, but I certainly believe in a judicious admixture of such labour, as we have at our disposal. A numerically increase of the total number employed on Westralian plants would tend both to reduce costs and improve extraction. It is a well-known fact that our Kafirs are quite useless if left to their own initiative on work that is not absolutely routine, and knowing this, the skilled workman is saddled with the responsibility of his own work plus that done by a most inefficient gang of irresponsible Kafirs. These facts have to be taken into account, and sorely trouble foremen and shiftmen alike on a large reduction plant. Is it not a penny-wise policy to expect your assayer to light fires, fetch and carry, when more important duties have to suffer? The same might be said of cleaning and sweeping by the extractor house man to the neglect of proper classification of

solutions, etc., affecting cyanide consumption and extraction generally. In conclusion, I am pleased to note the author allows, that our costs are low and extraction high, and it is most gratifying to know that these results are obtained with labour far below the standard of Westralian efficiency, according to his showing.

**Mr. E. M. Weston :** Regarding the author's remarks on the number of men employed firing boilers on the Ida H. G. M. Co., he says one man is sufficient to fire four boilers, but he omits to state that these boilers are fired by wood. There are few ashes to remove compared with the 30% here, and a man can fire with wood much more easily than with coal. Comparisons are useless, when all the facts relating to them are not stated.

### NOTES ON RAND MINING.

*(Read at March Meeting, 1908.)*

By **TOM JOHNSON** (Member).

#### DISCUSSION.

**The President :** Mr. Richardson will kindly read a contribution by Mr. Phillips on Mr. Johnson's paper. I have here a letter from Mr. Phillips, stating that this is only the first instalment.

**Mr. J. M. Phillips :** As our President has suggested, the author has certainly introduced nearly all the contentious points of mining in his paper, and on each one a good "paper" could be written. He covers such a large field, and generalises so much that one hardly knows where to begin in discussing the paper. In the first place, he contends that there are too many shafts and winding engines, etc., for the amount of rock hoisted. I agree with him; that there have been more shafts sunk than are absolutely necessary at the present time, but when the greater majority of those shafts were sunk, or in process of sinking, the true value or dip of our reefs were not so accurately known, as at present. Certainly no one can accuse us of sinking too many shafts on the one property at the present time. In fact, I contend that we are not sinking enough for proper ventilation and development. The idea at present appears to be, that one large shaft is more economical than two smaller ones, but I fail to see where the true economy comes in. In the one shaft system all the tramping, hoisting and ventilation is concentrated on one point. There is generally a congestion of traffic underground in consequence, and in the event of a serious accident happening to any one of the compartments, in all probability it will affect the whole shaft, disorganising both

hoisting and milling. All the necessary engines, three or four, are located under the same roof, which may easily result in the confusion of shaft signals, especially when two or more are being given at the same time. In my opinion, the adoption of two smaller shafts is more economical and advantageous, for if started simultaneously the development is better and more speedily accomplished, more faces, as the author says, being attacked at one time and ventilation more quickly secured.

The two shafts should not be so far apart as at present, as in the event of an accident occurring to the one shaft, the second one would be easily accessible for all purposes. Whilst I am in favour of the two smaller shafts as against the one large one, yet I do not believe, as the author evidently does, that three-compartment shafts are large enough. His contention that 5,000 tons can be hoisted through two 6 x 5 ft. compartments in 24 hours is feasible enough for, say, one 24 hours, but I doubt very much if the rate can be kept up day after day the whole month through, or even for one week. I work it out at approximately, with 4-ton skips, 200 tons per hour, or 25 trips with each skip, which means a trip every 4 minutes; this, I own, is not an excessive speed for any one trip, but when it comes to having to maintain this average the whole month through I think you will find that it will tax the energies of the skip man, at least, to his full capacity. Further, there is no allowance made at this rate for unavoidable loss of time, such as is needed in changing from station to station, examining the engines, shafts and ropes, etc. Then again, presuming that this rate can be maintained, how, may I ask, are the men, tools, etc., to be raised, which are necessary to break the 5,000 tons; surely, not by the same engine and through the same compartments, for I take it for granted that the third compartment is reserved for pumps, ladder way, etc. I have been on mines here, where it takes a separate engine and compartment for this work alone. In my opinion, nothing smaller than a four-compartment shaft will meet the requirements; two compartments for hoisting rock, one compartment for men, tools, etc., one compartment for pumps, ladder way, etc., and, of course, for the men, tools, etc., an extra engine is necessary.

As regards the shape of the shaft, I am in favour of the rectangular as against the circular. True circular shafts are stronger and safer in weak ground, but on the Rand—with few exceptions—we have no "heavy" ground to contend with in sinking, and when the shaft is sunk across the formation it is seldom necessary to carry "full" sets the whole depth. The hoisting in rectangular shafts certainly retards the ventilation to some extent, but not to the same amount

as is generally supposed, as one will find if they stand close to the shaft at one of the levels when hoisting is in full swing. It is also said that in other countries the circular shaft is easier and cheaper to sink, but on these fields that point is open to great doubt, and I believe that in some parts of Scotland and the United States of America the rectangular is used on the coal fields in preference to the circular. Circular shafts were sunk for some little depth on the Langlaagte Royal and Primrose mines here. It would be very instructive at this point to know why they were not sunk to the depth originally intended. The costs, as compared to the rectangular, of same capacity, could, no doubt, be ascertained from the companies mentioned. Seeing that the circular shape has not been adopted in any of our later shafts, I take it that they were a failure.

I do not quite understand the author when he says—in referring to rail guides—“guides put in in this manner would prevent the use of a bend.” I fail to see where a well laid out bend is any detriment to the use of steel rails; surely, the author has seen skips of all designs working around bends on steel rails here. Bends are bad things but sometimes they are unavoidable. In incline *v.* vertical shafts—for great depths—I fail to see where the advantages of the former over the latter come in. For incline shafts (*a*) it takes more footage to reach a given vertical depth; (*b*) consequently more timber, ropes, etc.; (*c*) the wear and tear on ropes is greater; (*d*) extra costs of rolls for rope; (*e*) heavier and closer timbering; (*f*) loading stations at every two or three levels at the least; whereas for vertical shafts (*a*) less footage; (*b*) less timber and ropes; (*c*) less wear and tear on ropes, guides, etc.; (*d*) no rolls; (*e*) lighter timbering; (*f*) loading stations can be placed farther apart; (*g*) quicker hoisting. Against the vertical shaft we have the extra cost of the cross cuts to intersect the reef, but then again an incline shaft is more subject to damage from subsidences, of which we have had several severe instances.

As regards the author's contention that too much timber has been used in our shafts, this to me, is a point which cannot be answered by a “yes” or “no.” It is a pity that this point was not particularised, for unless one personally examines the shaft one cannot say whether too much or too little timber has been used. Neither can one say that too much timber has been used in any particular shaft simply because the one adjoining has used less, for although the formation here is pretty regular, yet in sinking, the strata in one shaft might be more broken and heavier than in the one adjoining, thus necessitating more and even heavier timber.

I am struck with the inconsistency of one of the author's suggestions. In one place he advocates the sinking of the two shafts in close proximity to each other to facilitate ventilation, development, etc., then farther on he puts down 7,000 ft. as about the distance apart. Surely, 7,000 ft. cannot be regarded as a short distance in mining; neither can I see where shafts so far apart facilitate ventilation or development.

**Mr. Chas. B. Saner:** The author, as usual, has presented the members of our Society with any amount of food for thought. He enunciates various theories, which, although different to one's own ideas, are difficult to controvert, and to discuss in detail would take too long.

I heartily agree that much can be learnt from other districts and industries and applied to our own; it is very foolish for anyone to crow from his own perch before finding out if there are any better roosters in the vicinity to knock him out. We can all learn by interchange of ideas, from the methods of others, and by observation how not to do things and so improve on bad ideas. On the surface, given a modern plant, the running costs should not alter materially from month to month. But underground there are so many constantly varying factors, that a close comparison even between two adjoining mines is impossible, hence each mine should be judged upon its own merits. Even parts of the same mine differ greatly.

In some levels there may be only one narrow reef which is much faulted and cut up with dykes. In another section you may have two reefs close together, or a wide reef with good hanging and foot. All these conditions make a vast difference in working costs. Water may be cut, which at once upsets all calculations. Take the sinking of a vertical shaft, in good average ground speed is good, costs are low, and everything runs smoothly; then strike water or dyke or generally both together. Speed is greatly reduced sometimes by 50%; costs rise terribly and everything seems to get in a mess. As an example, I may mention that at the Turf mines shaft I was sinking for five months in hard dyke, which brought in some 80,000 gallons of water per 24 hours; this quantity is not great, but as it was continually following us down through fissures in the dyke we had to bale out every gallon from the shaft bottom and sink as well. Water is a curse when free, a blessing when under control. During the five months in the dyke the average sinking was 77·8 ft. per month at an average cost of £38 16s. per foot. During the last nine months, although we have passed through three small but dry dykes, our average has been 147 ft. and costs £26 5s., a reduction of £11 5s. per foot, but we

had good bedded quartzites and the water caught up.

The author remarks "truly there are too many shafts, etc.," but in the early days our knowledge of the reefs and their extent was not what it is to-day, hence small properties and numerous shafts; as we come to learn more of these wonderful banket beds the laying out of properties is being considerably modified; fewer shafts are being sunk to attack larger areas. Look how much better the first row of deeps are laid out than the outcrops, and the deeper deeps than the first deep.

If it is quite sufficient to have only a  $6 \times 5$  ft. shaft to haul 5,000 tons of rock or coal from 3,000 ft., how is it that in other countries the same terrible blunder is being made, as the author credits us with? From the last journals of the Institute of Mining Engineers, I see at the Bentley colliery, near Doncaster, two pits 20 ft. internal diameter were put down, designed to haul 2,000 tons up 600 ft. At the Polmaise collieries, near Stirling, two pits, 18 ft.  $\times$  6½ ft. and 19½ ft.  $\times$  6½ ft. and two others 1¼ miles from these (9,240 ft.) 22 ft.  $\times$  6½ ft., 16 ft.  $\times$  6½ ft., were sunk about 500 ft. and 1,250 ft. respectively to haul not half the amount and from a much less depth. Owing to the reef conditions generally to haul the quantity stated by the author is, in my opinion, an impossibility. If all the rock were to be hauled from one level or station the shaft would naturally be much smaller, but with an incline reef and various levels to haul from, fast pulling is not possible. There is comparatively a small quantity of rock to be obtained from each level when it is worked out and abandoned. For instance, Kimberley as a comparison with the Rand for fast hauling is absurd, and you may as well compare Park station with Euston, as regards handling passenger traffic; when only hauling from one station money can be spent to advantage in making good bins, station and other facilities, but when a station is to be abandoned in a year or so, surely it is folly to spend money in elaborate arrangements.

At the Village Deep No. 1 shaft a large bin was put in at the bottom of the vertical into which the broken ore from three levels was dumped through "ore-passes" cut in the solid; this bin holds 800—1,000 tons. A "Kimberley" chute was put in at the bin mouth, which holds the exact quantity to fill a skip. By this means the skip is filled as quickly as the one above is emptied into the surface bin, and by means of an ingenious lip hinged on the skip, little or no fines fall into the sump and good fast hauling is done; and before the electric pumps were installed at the No. 2 shaft, water was hauled all night and the mill was fed from 1½ levels; as well as all steel,

tools, men and boys being handled. In giving us examples of fast hauling, details at the same time should be supplied of the conditions, width of seam or reef, dip, facilities of underground handling, etc.

It is strange that the author should accuse the Rand of being so far behind the front rank in mining methods when we have amongst us some of the best mining men in the world. We can easily haul the rock up the shaft, but the difficulty is to get the rock broken in sufficient quantities, and when broken get it to the shaft bins. Mining banket is not mining coal, and similar methods are not applicable to both.

I quite agree that the driver of an engine should be made comfortable while on duty; with an independent distance bell, not connected to the engine indicator, but one rung by the ascending skip or cage 80—100 ft. down the shaft, and a good headgear, giving plenty of room above the collar, accidents should be few or none with good drivers. Telephones underground are a great boon in large workings, and the old excuse that the banksman would immediately warn all that the boss is on the move is weak. Opinion is divided as to the best form or shape of shaft, whether round or rectangular, although I notice English collieries adopt both. Shape depends upon the formation sunk through.

I do not agree with the author that two shafts should be sunk close together; if dykes or faults or water are cut in one, the trouble will be in both. When the shafts are far apart, that is, dividing the property so that each shaft works half the area, there is no necessity to put the permanent pumps in each shaft. Equip one with good pumps and gravitate all the water to, is the best and present-day practice. Unless the ventilation is guided, the drives from the shaft will be no cooler than under the present system of initial development. With the pump and ladder compartments, or half the shaft well bratticed, excellent ventilation is obtained and can be guided into the development faces. Much faster development can be done by this method than with the two shafts close together, for at each level, four faces will be available against two. With the latest developments, arrangements are being made with neighbouring companies for each company to put down only one shaft; and the latest idea, which is excellent, is to erect all the necessary machinery, permanent headgear and engines at once, complete this, withdraw the construction staff and go ahead and sink at full speed. Until lately temporary hoist and headgear were put up, and sinking attempted under great disadvantages and delays while the permanent gear was erected over your head.

*Cost per Ton* should not be the be-all and end-all, but profit per ton should be the aim. By all means pack as much waste as possible underground; by this means safety is increased, and there is a saving in shovelling, tramping and hauling, and that is now being realised. If the new electrical heating furnace, at present being expérimented with, proves a success, there will be a great saving in raising and lowering steel and in the quantity used; the great advantage of this invention being that an even temperature can be maintained, so a better class steel will be used with advantage.

*Shaft Guides.*—Two most interesting short papers appear in the Transactions, Institute of Mining Engineers, vol. 33, pp. 104-8, on various forms of shaft guides. Rope guides are much used in England, but have not found favour here, their place being taken by wooden guides, until comparatively lately; in section pitch pine 8 in. × 4 in. or 4 in. × 4 in. I prefer to have the joint between the dividers with a "backing piece," this "gives" a little and makes for smoother running. Steel rails of heavy section and slotted steel guides are now being put into the large shafts. These have the great advantage of giving great rigidity to the shaft timbers, and if rollers are put on the skip in place of steel shoes friction should be reduced and smooth running ensured. It would be most instructive to get comparative figures as to actual cost and life of the different styles. In one shaft I was connected with, wooden pitch pine guides were put in for sinking with coach screws. When the shaft was finished sinking these were taken out and 60 lb. rails put in; this was done by means of a 5-deck cage 35 ft. long, the corners of the compartment acting as guides; five white men on a single shift took out all the guides from two compartments and put in eight rails, or a total of 16,400 ft. in three weeks without in any way interfering with the work in the other compartments of the shaft. In another shaft three 8-hour shifts, each five men, were put on and 10,300 ft. of 60 lb. rails were placed in position and all wooden guides taken out and shaft examined and put in thorough order in 14 days. In the one case the job was contract, in the other day's pay and bonus. In putting down the steel slotted guides, my method is to bore the bolt holes in the dividers on the surface, and when a 30 ft. guide is needed, punch the guide on surface from measurements taken from the timber in position. We put a pair of guides in place by this means in half an hour. High lift electrical pumps in stations in the end of the shaft are doing good work in the latest sunk shafts.

*Labour.*—This is a subject I could enlarge upon at considerable length, but as my remarks have already assumed such considerable propor-

tions I will only touch upon one or two points. To my mind, there is no better human muscular machine than the native, and in those parts where little brain is needed but steady work, he cannot be beaten, and it is impossible to replace three or four natives by one white man in the lower branches of mining. The European hewer of wood and drawer of water has little ambition, and is content to earn his small pay comfortably at home. If he comes to a foreign country he wants to rise. It is no good putting grown men in the mine, the ex-soldier, clerk or butcher, who cannot stand the dirt and rough work, and only takes it on as a very temporary billet; educate the youth, and there may be hope in years to come. The one great trouble with the native is his shortness of service; he is in a continual state of crude apprenticeship. I am in favour of establishing married quarters for natives, under proper control; I am sure by this means we should get a steady, contented and more efficient class of labour, and pay by results. Have well conducted eating houses, like mine boarding houses, and do no cooking in the compound. Let each native buy his food like the white man. If he will not work, then apply the motto: no work, no food.

**Mr. E. M. Weston:** One could easily write another paper of equal length, discussing all the points raised by the author, but I can deal only with a few points.

Our President has first already pointed out that it is quite a different matter to haul coal or rock from one station as compared with eight or ten. The author ignores the fact that owing to the laying out of the mine, due to faults, dykes, etc., it may be much cheaper, and indeed probably is in the cases the author has in mind, to employ two or three hoisting engines to haul 1,500 tons per 24 hours, than to employ one. I am glad to see the author has been reading up technical literature about controlling apparatus for prevention of overwinds. He has, however, forgotten to mention that designed by Futers, and in use in the Midlands. I am afraid the author will not convert many engineers to his idea of sinking circular shafts, especially if he were going to sink them without dividers. Personally, I would rather be out of such a shaft as soon as faulted ground or dykes were met with, and a good many miners would feel the same. It would be practically impossible to secure a circular shaft of large dimensions under such conditions, without the use of masonry or solid iron linings. If he is leaving dividers out, of course, bearers would be left out also. How is the author going to secure his rope guides while blasting in hard ground? The nice coal mining arrangements would soon be reposing on the bottom of the shaft. Bends in shafts certainly

do not seem an unmixed blessing, and I think fewer will be put in in future. Is the author serious when he advises us to put in endless ropes on an incline shaft of 45°? Has he thought for a moment of the consequence of a stone dropping out on to the track and derailling only one truck for a start out of the hundred or so? How long does he estimate it would take to extricate and disentangle the final mess at the shaft bottom? Go easier Mr. Johnson, you must not startle us out of our sleep of rigid conservatism with such radical proposals. You may overdo the dose and the results might be serious. Why not put belt and bucket elevators in our vertical shafts! Nearly all the same advantages can be claimed for them as you claim for your proposal.

It is sad to see that all these engineers who have been worrying over dense air transmissions, electric transmissions, etc., have been wasting their time, and that the author has the true solution, which is, to take boilers and, I suppose, economisers and superheaters with feed pumps, etc., underground. See what a lot of pumping might be saved also, as the water could be lifted as exhaust steam!

If some of the mines, whose property is divided into two halves by a dyke or fault with 500 or 1,000 ft. throw, had been developed by the author's double-barrelled shaft they would be in an unhappy way now. Here again, I do not think the author serious. Has he never heard of bratticing a single shaft to make the mine as "cool and comfortable" as possible? Does he really contend that development can be done as rapidly from such a double-barrelled shaft as from two shafts set 2,000 or more feet apart and provided with brattices? "Much felt in practice," I should pity the author if he had to redeem a pledge to develop an equal or greater amount of ore in a given time by his scheme as could be done with two shafts in the proper position. How about future ventilation? Perhaps the author will recommend a system of bratticing the drives, and take out a patent for a rapid repairing machine to put them in again when they are blasted out or when the boy pokes a cocopan through them. If one shaft only is to be sunk, I venture to put forward a scheme which should provide equal haulage capacity to the double-barrelled scheme with a lesser expenditure.

Briefly, it is this: a seven or eight compartment rectangular shaft, timbered with wood, would be sunk, but it would be about 2½ or 3 ft. longer than at present. Double dividers set 2 or 3 ft. apart would be put in between hauling compartments in the centre; the joints of wall plates would be put in between these dividers. While sinking, special hitches would be cut 50 ft. apart in this semi-compartment, and steel bearers of I

channel beams would be put in with ordinary dividers at suitable intervals. The shaft would be sunk to the required depth with a temporary air brattice, and the only extra expense would be due to extra length of shaft, and the special bearers and extra timber—not big items. After the shaft was sunk 8 to 12 old hauling ropes would be hung from the collar set between dividers in the centre of the 2 to 3 ft. space. They would be spaced 9 in. to 12 in. apart and form a line across the shaft. They would pass between two I channel beams every 50 ft., and would be clamped in tension above and below each set, and perhaps between them. Then beginning at the bottom, the centre 2 ft. would be partitioned off on both sides and filled with concrete from wall to wall. This would not take long, as there would be skips on either side. For a 4,000 ft. shaft about 2,600 yards of concrete would be required; but a lot of large filling could be used; 1,600 ft. of I beams chains and some old ropes would be needed. The cost of all the work would be very little compared to that of another shaft. Mining from such a single shaft might then be permitted. The centre ferro-concrete wall would be of immense strength and would form a secure solid continuous centre bearer under the wall plates at their weakest place, rendering any collapse of the shaft due to heavy ground impossible. It would make a perfect fire-proof screen right from top to bottom. One shaft might be burnt right out without injuring the other. It would form a perfect airtight brattice, rendering the mine "cool and comfortable." Stations could be opened out separately for each half shaft, or the dividing wall built in on stations. It would offer immense resistance to any explosion such as occurred in the Driefontein shaft. Such an arrangement would make two shafts out of one and would avoid most of the objections against mining from one shaft. Even if an explosion wrecked the wall in one place, the end hauling compartments should escape damage. By making the concrete division larger a passage way for pipes and wires could be left in the solid of the wall, preserving them from water, and the experiment of concreting the main air pipe in might be worth trying.

Again, we wonder if the author is poking fun at us, when he suggests putting 12 machines on a 200 ft. face. First we would ask him, if it does not now pay to install motors on levels with 250 ft. of backs removing the ore slowly, how is it going to pay better to put them in for a shorter time for the same tonnage? I can imagine the state of mind a poor trammer would be in who had to keep such a face clean, when one of his trucks ran off the line and spilled. Why could drives be any further apart with this system than they are now? Surely, more rock must somehow

be moved in a shorter time than is done now, and moved a longer distance. Why should brake roads, etc., help more than they do now, when they would have to be pulled down twice a day? What is our miner in charge of the face going to do if one lot of workmen drill much better than others? He must either sack the average man, or one lot of men would eat in ahead of all the rest, and the stope instead of assuming that ideal, so beautifully shown in the author's figure, would approach in shape the proverbial cow's hind leg. It would be quite a nice problem in transportation to get the dozen machines, bars, gear, hoses, pipes, etc., out of a 250 ft. stope on to the levels below or above twice a day; as obviously they would be scarcely safe in the stope itself anywhere. How long would the rigging up take next shift? One lot of men certainly are given a full shift of work which would take them the 12 hours, while the other men and the rigging gang have not enough to do. I am afraid even the policy of breaking rock until the stope was full of rock would not remove the objections to this scheme. If miners working two or more machines in a face are to break rock to greatest advantage, using their brains they must have lots of face to work on. Even three machines in a 160 ft. face are too many, and 80 ft. of face per machine pays every time. If the miners are to have good benches each time they set up, they must have room. There is more money lost to-day by the crowding of machines which is prevalent, than by not having enough at work.

I agree with the author that low costs are not always an evidence of good work, but I am afraid if some of his schemes were adopted we should have neither low costs nor good work. Speaking of grade, it certainly seems a peculiar method of keeping it normal, to spend the last week of the month hunting round the mine for any waste or unpayable reef to put in the mill to keep it down, if too high, or digging frantically at the rich stopes to whip it up, if too low. Such things are, I believe, known in certain high grade mines, and if you ask the mine captain, if anything below 16s. rock is left in the mine, he will tell you it is—sometimes.

I must now turn to the new system of placing stope holes. I am glad to see that the author believes in long holes. There are still mines not a hundred miles from here where the longest chisel that goes down the mine is 7 ft. With regard to the author's proposed arrangements of stopping holes, all I can say is, that, if a miner was working for me and I found holes like those shown struck in his stope, I should be obliged to sack him.

It would require several diagrams to go into this matter fully, and I have not time to draw them, but it can be easily shown that though the

author's holes would, no doubt, break some ground, they are not efficient. The mistake most rock drillers make is shown very evidently in the author's diagrams. Most men have no idea or do not take the trouble to think what their first hole is going to break. In ninety cases out of the hundred it does not break in the crater form beloved of theoretical investigators, but tears most of the reef out in front of it. Nearly all the reef here breaks in slabs or heads at right angles to the walls, and in the author's case, were he to fire his front holes first and then go back and look he would find that the second hole had about a quarter of the burden on it should have, and the same with the other holes. Even in a stope of 38 in., if the top hole is given 24 in. burden, the bottom hole will break from 30 to 36 in. from the face. The author apparently and incompetent stopers place it vertically below the other hole, and probably give it the same charge of dynamite. As Prof. Yates has pointed out, it is this habit that renders machine stopping so much less efficient than hand stopping, and efficiency in machine stopping can only be brought up to a similar level in efficiency by using a sane imagination to judge just what each hole will leave for the next one to break. Every miner on going to a new stope should fire only one of his four or five holes first time, and go back even in the smoke to see just what it has done before he fires the other. It will be time well spent. When I saw the drawing of the author's new cut I said, now we have something at last that is new and original. Some friends of mine on contract at the Village-Main Reef tried it. Their report to me reads somewhat as follows:—"We both made experiments with the new cut. We regret to say that owing to our limited financial resources and the necessity of breaking the round we have been temporarily obliged to discontinue them. Should, however, the author care to finance us for a few weeks and make us right with the mine captain, we will be pleased to resume them."

The author's remarks on the carrying of stopes appear to be based on experience. He recognises the fact that hammer boys are most useful in machine stopes. I have always contended that they should be used to do work in certain places where they are more efficient than machines. The author apparently thinks miners do not know their business when they tell him they object to using their machines under the bar in narrower stopes. If he thinks he knows better than those who have tried it, I am sure he is mistaken. I have tried it often enough myself, and it is an excellent method for wasting time, and should never be adopted unless in exceptional cases, where faults have thrown the reef down. I cannot agree with the author when he says that front

holes should of necessity be given more burden than back holes, and that back holes must of necessity look in. If the bar is rigged in the right place there is no need for this in a four-hole bench in a large stope or a five-hole bench in a small stope.

Surely it is the wrong kind of economy to try and use 2½ d. steel. Unfortunately here high carbon steel at 5d. to 6d. a pound does not get proper treatment, as the author says; but such low grade steel, as the author recommends, is far too soft. The author's idea of rounding the edges of his chisel bits to give more cutting edge where it belongs, is an amusing instance of doing things the wrong way. What the author should have done was to make up the shoulders of his bit quite square and not rounded at all, if anything, somewhat concave. A section of the bit taken half an inch back from the edge of the bit should be in the form of a rectangle with sharp corners. Then the bit has one cutting edge parallel to the face of the hole, and four cutting edges parallel to the direction of the hole, to ream it out properly and cut the rock where there is most to cut. Taking out the centre of the bit is no new idea. Its effectiveness in chisel bits is spoiled by its weakening the cutting edge too much. The author will find that for this reason everyone using hollow steel tries to get his hole down on to the side of the cutting face. With star bits it is effective if it could be done economically, as thereby a core of rock is formed which is jarred off and breaks in large pieces. In Gillette's book on rock excavation the author will find particulars of the Derby tubular drill bit, designed to work on this principle with water feed.

Now, what do I really think about the author's paper? Well, I think the author determined to stir us up a bit, so he wrote a paper bringing the best arguments he could for all the bad practices in mining he could think of, with some useful notes from personal experience, hoping, as the French say, that he would at least "give us furiously to think," and perhaps evolve a few useful and new ideas. If this was his plan and motive, I am sure he deserves our best thanks.

### SOME FEATURES OF SILVER ORE TREATMENT IN MEXICO.

By W. A. CALDECOTT, B.A., F.C.S. (Member).

#### PART II.—THEORETICAL CONSIDERATIONS.

##### DISCUSSION.

Dr. J. Moir (contributed): As regards this paper, I am mainly interested in the discussion by the author on the theory of the solution of silver sulphide in cyanide, more particularly the experiments described on p. 267. I wish to

point out that the true equation for the reaction is more probably that given in the footnote of p. 266, simply because, in dilute solution, alkaline sulphides do not exist but are almost wholly hydrolysed into hydrosulphides. This also is borne out by the author's Table A, according to which the amount of sulphide solution required to give a coloration in argentocyanide solutions containing 16.9 molecules of free cyanide to one of silver, is directly proportional to the free cyanide present. The equation on p. 266 requires (applying the mass action law):

$[S''] [AgCy'_2]^2 = K[Cy']^4$  or since  $[Cy'] = 16.9 [AgCy'_2]$  in the experiment,  $[S''] \propto [Cy']^2$ ; it follows that the equation governing this reaction is not the reverse of that on p. 266, but is  $KSH + KAgCy_2 = 2KCy + AgSH$ . The latter equation requires, as above, that  $[SH'] [AgCy'_2] = [Cy']^2$  which, in the author's experiment, reduces to  $[SH'] \propto [Cy']$  which is just what he finds by experiment. It follows that the brown coloration is due to AgHS rather than to Ag<sub>2</sub>S.

I have recalculated the author's figures, so as to allow for the effect of dilution due to addition of the reagent. It will be seen that the figures give quite a good constant at the end (see Table I. on opposite page).

The conclusion come to above is in agreement with earlier thermochemical work by Berthelot,\* who found that the heat of neutralisation of dilute solutions of H<sub>2</sub>S with one and two molecules of caustic potash were the same, which shows that the reaction  $KSH + KOH = K_2S + H_2O$  does not occur in dilute solutions, whence it follows that when K<sub>2</sub>S is dissolved it breaks up into KOH + KSH. He also found that no precipitate of Ag<sub>2</sub>S, only a coloration, was obtained on adding H<sub>2</sub>S to KAgCy<sub>2</sub> solutions, unless the solutions were made acid or boiled, which certainly suggests that the coloration is not Ag<sub>2</sub>S but AgSH.

I have made some experiments to test this view: (1) Two different argentocyanide solutions were titrated with sulphide solution (a) alone, (b) after adding an excess of caustic alkali to convert SH' into S''. In both cases the alkaline solution gave a precipitate with very little sulphide, the quantity required being the same for both: the quantity of sulphide was thus independent of the amount of silver present if there is an excess of alkali.

TABLE II.

No.	Cy' Con- centration.	AgCy' <sub>2</sub> Con- centration.	S'' or SH' concentration	
A	·077	·004	·00023	} without alkali.
B	·090	·009	·00010	
A'	·100	·005	·000011	} with alkali.
B'	·100	·010	·000015	

\* Comptes rend., 1899, pp. 630 and 706.

TABLE I.—Data of Table A of p. 267, recalculated.

Free Cy' % calculated as KCN in Original.	Silver in Original.	Na <sub>2</sub> S used to first color- tion. '01% Solution.	Corrected Volume of 100 c.c. Original	Cyanidion concentra- tion in Equilibrium.	AgCy <sub>2</sub> Con- centration = Cy' /169.	Na <sub>2</sub> S present in Corrected Volume.	S'' or SH' Concentration in Equilibrium.	Value of [SH'] [AgCy <sub>2</sub> ] [Cy'] <sup>2</sup> = K
%	gm./litre.	c.c.	c.c.			mgm.		
·50	·490	2·65	105·3	·0728	·00431	·53	·0000645	·000052
·40	·392	2·20	104·4	·0588	·00348	·44	·0000540	·000054
·30	·294	1·55	103·1	·0446	·00264	·31	·0000385	·000051
·20	·196	1·00	102·0	·0301	·00178	·20	·0000252	·000049
·10	·098	0·50	101·0	·0152	·00090	·10	·0000127	·000049
·05	·049	0·35	100·7	·0076	·00045	·07	·0000089	·000069

One would expect from this result that excessive alkalinity would hinder the solution of silver ore in cyanide, even though the amount of sulphide in solution be not increased correspondingly, the action being due to the conversion of hydrosulphide into sulphide.

On the other hand, in concentrated solutions, it is probable that most of the sulphide is present as such (*i.e.*, not hydrolysed to SH'), in which case the first equation given by me holds good, and this appears to account for the precipitation of Ag<sub>2</sub>S on diluting such a solution, as mentioned by the author on p. 268.

(2) *Experiments with Constant Cyanide and Varying Silver.*—These were made by weighing various quantities of solid KAgCy<sub>2</sub> into decinormal and centinormal cyanide solution and titrating with sulphide until a coloration appears after waiting for a minute. In the table the results have been corrected for change of volume by titration:—

TABLE III.

No.	Cyanide Concentra- tion.	AgCy <sub>2</sub> Concentra- tion.	SH' Concentration	Value of K= [SH'] [AgCy <sub>2</sub> ] [Cy'] <sup>2</sup>
1	·047	·0047	·00003	·000064
2	·095	·019	·000025	·000054
3	·090	·009	·000057	·000063
4	·010	·0005	·000015	·000075
5	·010	·001	·000011	·00011
6	·010	·0024	·000009	·00021
7	·010	·0050	·000005	·00024

It will be seen that there is considerable deviation in the constant when the ratio between cyanide and silver is diminished.

The meeting then closed.

### Contributions and Correspondence.

#### HONEYCOMB TUBE MILL LINERS.

With reference to my "Notes on Waihi Ore Treatment," read before the Society last year, I have received the following memoranda from

Mr. H. P. Barry, superintendent of the Waihi G. M. Co., New Zealand:—

"The honeycomb tube mill liners continue to give every satisfaction. With castings 3½ in. deep, we get over 12 months' wear. By making castings slightly deeper and allowing the rough rock to project well above the castings, a life of 18 months could be obtained. Surely, there must be hard local rock at Johannesburg that would be suitable and that would come out much cheaper than cast iron or steel liners? It is of advantage to leave the rock projections very rough, as the rougher the liner the better, as it prevents slip of the mass of pebbles. We found out in the early days of tube milling that smooth iron or steel liners were bad on this account, so had ribs cast on them which prevented the slip.

There is no standard design for these liners; each man can make them the size and thickness he thinks best. It is well not to make the openings too large. I put eight segments into a 5 ft. mill, each segment being divided into four compartments."

RALPH STOKES.

May 30th, 1908.

#### DESCRIPTION OF AN APPLIANCE FOR WASHING SLIME OR SANDS RESIDUES.

The appliance described below has been used at the Simmer and Jack G. M. Co. for washing sands and slimes residues for the last two months and has given quite satisfactory results. The device is made from a piece of 8 in. pipe, 9 in. long, with a flange screwed on each end. The bottom end has a ¼ in. iron plate bolted on, through which a number of ¼ in. holes are drilled. Between this plate and the flange a piece of ¼ in. mesh iron screening and a filter cloth are inserted to form a filter bed. The top cover consists of a round piece of ¼ in. plate similar to the bottom cover, but in this case the bolt holes in both flange and cover are slotted out, so that the bolts can be removed by loosening the nuts, and the cover removed and replaced

quickly. A  $\frac{1}{2}$  in. pipe is screwed into the cylinder immediately under the top flange and connected to an air main. The sample to be washed—sand or slime—is dumped into the cylinder and clear water added until nearly full; when the whole has been mixed up by hand the top cover is put on and the air valve opened slowly. The air entering at the top drives the liquid through the bottom filter, and after about three minutes the cylinder can be opened, when, if a slime sample has been operated upon, a cake containing about 15% moisture will be left on the filter cloth. This is again mixed up with clear water and the operation repeated, after which the sample is removed and sent to be assayed. Two washings have been found quite sufficient, and assays of the second wash filtrate only gave traces of gold. Cyanide managers will appreciate the advantage of getting a slime or sand residue washed in a few minutes instead of having to go through the tedious method of diluting with water, settling and decanting the liquid a number of times. If no air main is available an ordinary bicycle foot pump will supply the air pressure required.

GEO. O. SMART.

May, 1908.

#### NOTE ON BARBERTON "BLACK SANDS."

Looking through an old note-book recently, I came upon the following record of tests made upon samples of this product, which may be worth while placing upon record:—

In July, 1905, a sample submitted by Mr. Waites was examined, and it was found that from 83.3% to 84.5% could be separated by magnetizing. The original sample assayed 22.5 dwt. gold per ton, the magnetic portion only 1.5 dwt., and the non-magnetic residue went 131 dwt., showing that 94% of the gold became concentrated in about 16% of the weight of the parcel. In March, 1906, another sample assaying 5.5 dwt. was received from Mr. Harrison and tested with similar results, the magnetic portion removed representing 80.2% and assaying 1.0 dwt., and the non-magnetic residue 19.8% assaying 24 dwt., thus nearly 86.0% of the value was concentrated in 20% of the weight. Different parcels of this material, as is well known, vary greatly in value, but if the above samples represent a typical well concentrated product, the further concentration of roughly 90% of the values into 20% by weight, simply by the very cheap process of magnetic separation, obviously has a most important bearing on the extraction of the gold, lessening freight charges by 80% if, as usual, sent up to Johannesburg for treatment, and reducing by 80%

also the tonnage to be submitted to the more expensive metallurgical treatment.

Personally, however, I think the best way of dealing with the problem would be the establishment of a custom works at some suitable spot in the De Kaap valley, in which case, if various estimates I have heard with regard to quantities available are reliable, a great revival of activity in the district should take place, and profits could be made in many cases which at present are quite unpayable.

G. H. STANLEY.

Transvaal University College,  
Johannesburg.

#### Obituary.

The following deaths are recorded with much regret:—

Mr. CLEMENT LECKY JOHNSON, B.A. (Associate), who died on the 31st May, had been in very indifferent health for some years. He was connected with the General Mining and Finance Corporation for many years, having been for a considerable period the cyanide manager at the New Goch Gold Mines, Ltd., and for the last two or three years the caretaker at the Aurora West Gold Mine, at Maraisburg, where he passed away. Mr. Johnson was a B.A. of Dublin University, and in addition to his scientific attainments, he was an excellent cricketer, being a member of the first S.A. team, which visited England in 1893-4. Mr. Johnson was admitted an Associate of the Society in July, 1898.

Mr. FRANCIS GORE KNOX LITTLE, who was elected a Member in Sept., 1906, and was for some time an assayer on the Crown Deep, Ltd., died some months ago in West Africa, whence he had gone to engage in mining pursuits.

Mr. HARRY THOMAS PITT, who was elected a Member of the Society in Feb., 1898, died suddenly on the 2nd March from the result of an operation which had become necessary, owing to the after effects of a wound received during the war. Mr. Pitt was one of the oldest residents on the Rand, and was for many years cyanide and mill manager of the Rose Deep, Ltd., and was greatly esteemed by all with whom he came in contact. He served during the war with the Railway Pioneer Regiment, and was dangerously wounded at Sand River in 1900, from which he eventually succumbed. The funeral ceremony, at Braamfontein, was largely attended, and the Transvaal Town Police band and a firing party of the Witwatersrand Rifles, under Capt. C. B. Saner, being in attendance. The Society sent a wreath, and was represented at the funeral.

## Notices and Abstracts of Articles and Papers.

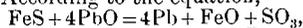
### CHEMISTRY.

**PRODUCTION OF ALUMINIUM CARBIDE.**—"The researches of the late Prof. Moissan in pyro-chemistry led to the discovery and isolation of many inorganic compounds, the production of which is only possible at the high temperatures attained by means of the electric furnace. Among the combinations thus effected for the first time was the interaction of carbon and aluminium to form aluminium carbide  $Al_4C_3$ . This carbide is distinguished from the carbides of other metals by the fact that it yields methane on treatment with water. Although earlier investigators had failed to obtain this substance in ordinary gas furnaces, Matignon has recently been led to repeat the attempt, owing to the circumstance that the heat of formation of aluminium carbide has been determined by Berthelot as having the high value of 245 major calories. The combination is accordingly a reaction accompanied by a great generation of heat, and should proceed to completion, if once initiated in a sufficiently intimate mixture of the two reagents. This theoretical view has now been confirmed by experiment. Finely divided aluminium and dry lamp black are intimately mixed with the aid of some turpentine, and the mixture, which should contain some excess of the metal, is heated for 20 minutes in a Perrot furnace, when the combination is complete. The excess of aluminium is extracted by rapidly washing in the cold with hydrochloric acid, or preferably aqueous caustic potash. The central portions of the fused mass consist of small hexagonal leaflets having a clear yellow tint, quite similar to the product obtained by Moissan. After removing aluminium, the carbide, when decomposed by water, gave rise to nearly 97% of the calculated amount of pure methane. The combination may also be effected by the thermite process without the application of external heat, and the various chlorides of carbon when heated with aluminium at comparatively low temperatures give rise to the carbide. With aluminium and hexachlorobenzene in sealed tubes the reaction takes place completely to form aluminium chloride and carbide, even at 225° C. Other chlorides of carbon interact when their vapours are lead over gently heated aluminium (*Comptes rendus de l'Académie des Sciences*, October 21, 1907). As a method for the commercial production of methane this reaction would not be so economical as the patented processes, in which carbon monoxide and hydrogen are caused to interact in the presence of finely divided nickel at about 230° C. Nevertheless, Matignon's method of obtaining carbides is capable of wide application."—*Times Engineering Supplement*, Jan. 8, 1908. (J. A. W.)

**A NEW REDUCER IN FIRE ASSAYING.**—"It is quite well known that much annoyance is experienced in assaying highly silicious ores by the contents of the crucible boiling over when argol or other carbonaceous reducing agents are used, and to avoid this it is necessary to start the fusion at a low temperature and heat gradually. (It was brought out in the discussion of this paper that a crucible is not so liable to boil over if the fusion is started in a very hot muffle.) Sometimes a cover of salt is used to prevent this boiling, but in that case the fumes are troublesome in pouring, if the slag is decanted from the lead.

This boiling is caused by the disengagement of carbon dioxide from the carbon in the reducing agent. (If a fusion is made without a reducing agent it will not boil over.) It occurred to me that if a sulphide were used as a reducing agent no gas whatever would be produced from this source, since the metal would combine with the silica and the sulphur be converted to sodium sulphate. This would have the double advantage of producing more quiet fusion and introducing a basic element into the slag. Probably the most convenient sulphide for this purpose is iron sulphide, which can be obtained free from gold and silver. This was tried with very satisfactory results.

Silicious ores that boiled badly, when a 0.5 assay ton was fused in a 20 gm. crucible, using argol as a reducing agent, gave no trouble whatever when the assay was made in a 15 gm. crucible, with the same quantity of ore, and iron sulphide used in the place of argol. According to the equation,

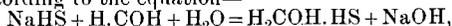


1 gm. of iron sulphide ought to reduce 9.4 gm. of lead. In practice about 9 gm. are obtained, so that the reaction is nearly quantitative.

The following charge was used: 0.5 assay ton of ore; 80 gm. litharge; 20 gm. sodium bicarbonate; 10 gm. potassium carbonate; 3 gm. iron sulphide.

I have never seen the use of iron sulphide as a reducing agent suggested in any literature on assaying, but it ought to appeal especially to those who assay sulphide ores by the nitre method. Its use instead of argol saves about 15 to 20 minutes in time of fusion."—C. A. ROSE.—*The Mining World*, Jan. 11, 1908, p. 61. (W. A. C.)

**VOLUMETRIC DETERMINATION OF SODIUM MONOSULPHIDE.**—"The volumetric method of determining sodium monosulphide which is based on its decomposition by the action of water into sodium hydrosulphide and sodium hydroxide, has the drawback that commercial sodium monosulphide almost invariably contains free sodium hydroxide. In the author's method this is obviated by first titrating the solution with  $N/10$  sulphuric acid, with phenolphthalein as indicator, then adding formaldehyde solution in excess which acts upon the sodium hydrosulphide according to the equation—



and continuing the titration with the standard acid until the liquid again becomes colourless. The first titration gives the amount of free alkali in the salt together with that liberated by the action of the water, whilst the second gives the amount of sodium hydroxide formed by the action of the formaldehyde on the sodium hydrosulphide. Thus one molecular of sodium monosulphide yields two molecular of sodium hydroxide. Results thus obtained are stated to agree closely with those given by the iodine and zinc methods."—E. PODRESCHETNIKOFF, *Z. Farben-Ind.*, 1907, 6, 338.—*Journal of the Society of Chemical Industry*, Jan. 15, 1908, p. 21. (A. W.)

**ASSOCIATION OF ALUNITE WITH GOLD.**—"In the Goldfield district, Nevada, alunite is so abundant and so generally associated with the intense alteration accompanying ore deposition that it must be looked upon as one of the most characteristic minerals of the gold deposits. This close association of the gold with a mineral which apparently had not been recognised in Nevada prior to the present investigation and which has been reported in comparatively few localities elsewhere, seems worthy of some description, especially as the relation promises to throw light upon the origin of the ores.

Attention was first directed to the occurrence of alunitic in the district by the finding of small nests, or crystalline aggregates, up to 5 mm. in diameter, of a pale pink mineral in a greatly altered nearly white rock exposed on the east shoulder of Vindicator mountain, about 2 miles north-east of the town of Goldfield. The rock contains quartz phenocrysts and was probably originally a rhyolite. The microscope shows that the feldspar phenocrysts, as well as the ground-mass, have been altered to an aggregate of quartz and alunite.

With the alunite is associated a subordinate quantity of an anhedral, colourless mineral of high refractive index, strong birefringence and possessing a single well marked cleavage. The mineral is biaxial and optically negative, the plane of the optic axes coinciding with the cleavage. It resembles diaspore but differs from that mineral in being optically negative. It has not yet been identified. Undoubted diaspore has been found intergrown with alunite, in the altered dacite of the Florence mine.

Microscopic study of the bleached, altered rocks of the district has revealed alunite in nearly all of them. The rhyolite forming the summit of Columbia mountain, just north of the towns of Goldfield and Columbia, has been changed to a quartz alunite rock, the alunite, with some quartz, forming pseudomorphs after the feldspar phenocrysts which probably were originally orthoclase.

Alunite is very abundant in the altered andesite and dacite of Preble mountain and vicinity, and is present in practically all of the altered dacite which forms the country rock of the Combination. Florence, January, Red Top, Jumbo and other mines near Goldfield. It also occurs in abundance directly associated with the sulphide ores in these mines, in some places, particularly in the January mine, as a soft, easily pulverised material, of snow white or faint pink colour, which is not always distinguishable on mere inspection from certain pure forms of kaolinite which occurs in the mines of this and other districts.

At Goldfield, the delicate pink tint of the alunite is rather characteristic and serves often as a rough-and-ready means for distinguishing it from the kaolinite with which it is often associated. Close inspection of the altered rocks in which alunite is abundant usually reveals this faint pink colour in the aggregates which have replaced the original feldspar.

As a constituent of altered rocks in the Goldfield district, the alunite does not have complete crystal form. It is tabular in habit, the basal pinacoid being well developed and the rhombohedral faces being absent. In thin sections it shows distinct cleavage parallel to the basal planes and parallel extinction with reference to the cleavage. The interference colours in good thin sections are mostly yellow but rise in places to red or blue of the first order. The mineral gives a positive uni-axial interference figure, in sections showing no cleavage. The index of refraction is distinctly higher than the balsam of the slide (1.54).

The most decisive optical means of distinguishing the mineral from the optically negative sericite which it resembles in habit and refraction, is by means of the quartz wedge or gypsum plate. Sericite shows the higher interference colours when the section is turned with the cleavage at right angles to the direction of greatest elasticity ( $\lambda$ ) of the plate or wedge, while alunite shows the brighter colours when the basal cleavage is parallel to this direction.

The soft massive alunite, such as occurs in the January mine, when gently crushed on a micro-

scopical slide and stirred in a drop of water, shows under a high power of the microscope thin, colourless scales of hexagonal outline. The average diameter of those examined is less than 0.05 mm., and such is their tenuity that no crystal faces can be recognised on the edges of the basal pinacoidal scales."—F. L. RANSOME.—*Mining World*, Jan., 1908, p. 66. (H. A. W.)

UTILISATION OF CERIA.—"The production of pure thorium for incandescence mantles from monazite sands leads to an accumulation of large quantities of crude cerium salts as a bye-product for which at present there is no outlet. Attempts are being made to employ these compounds in the arts. Cerium salts have been tried as mordants in dyeing, and recently F. Gareli has experimented on the utilisation of cerium and its allies in the tanning industry. Hide powder rapidly decomposes the salts of cerium, lanthanum, and didymium, fixing these elements in the form of their hydrated oxides. These salts appear, therefore, to behave like those of aluminium or chromium. Ceric sulphate and the double nitrate of cerium and ammonium give rise to leather which has a pale yellow colour and resists the action of water. The leather obtained with thorium or zirconium nitrate is flexible and has a white tint. The hide partially reduces ceric salts and absorbs oxygen, which gives rise to a good quality of leather. The employment of ceric salts as oxidising agents in the manufacture of certain important chemical substances, such as benzaldehyde and related compounds, has been patented, and a series of cerous salts of the organic acids employed in pharmacy has been prepared with a view of ascertaining the therapeutic effect of cerium."—*Times Engineering Supplement*, Jan. 29, 1908. (J. A. W.)

DETECTION OF RUTHENIUM IN PLATINUM ORES.—"A portion of the alloy is melted with lead, the regulus treated with nitric acid, and the residue ignited in the air in order to expel the osmium. The residue, containing platinum, iridium, rhodium, and ruthenium is heated with a mixture of potassium nitrate and potassium hydroxide, and after cooling, the mass is extracted with water, and excess of nitric acid added to the solution, when the following reaction takes place:

$$2K_2RuO_4 + 4HNO_3 = RuO_4 + Ru(OH)_4 + 4KNO_3$$

The brown liquid is poured into a beaker or test-tube, and the latter covered with a piece of filter-paper. If the alloy contained ruthenium, the paper will be blackened in the course of 12–24 hours, owing to the liberation of vapours of ruthenium tetroxide. It is stated that 0.01 gm. of ruthenium can be easily detected in this way. Care must be taken that the whole of the osmium is removed, as this gives a similar reaction."—N. A. ORLOW, *Chem.-Zeit.*, 1908, 32, 77.—*Journal of the Society of Chemical Industry*, Feb. 15, 1908, p. 125. (A. W.)

DETECTION OF FORMIC ACID.—This depends on its reduction to formaldehyde by metallic magnesium. The solution to be examined is digested with the metal, in the form of ribbon or powder, at the ordinary temperature for some minutes, and the resulting product is tested for formaldehyde by any of the usual colour tests, or if the quantity is sufficient by means of phenyl hydrazin. Soluble metallic formates give a similar result but the reaction is much more delicate when the free acid is employed. H. J. H. FENTON and H. A. SISSON.—*Proceedings Cambridge Philosophical Society*, xiv., iv., 385. (J. A. W.)

**GOLD LOST BY VOLATILISATION IN ROASTING.**—“As the author's general experience had not shown any marked difference between assays done on the raw or on the roasted ore, although the advantage, if any, lay with the latter, further experiments were tried to ascertain whether gold is actually lost by volatilisation during careful roasting.

In one experiment, a known weight of a rich mercuric telluride ore was placed in a combustion tube, the ends of which were loosely plugged with glass wool and asbestos to prevent any escape of particles of ore. One end of the tube was left open to the air, but the other end was attached by a rubber tube to an aspirator, there being a Woolf's bottle between the glass tube and the aspirator. The gaseous products from the tube were all drawn through water in this Woolf's bottle. The tube was placed in a combustion furnace and aspiration started so as to draw a current of air through the ore, and, after gradual heating to a cherry red, the ore and sublimate were assayed separately.

The portion of the tube containing the oxidised ore from which all tellurium and mercury had been distilled away, was cut off, ground up and assayed, tube and all, the portion of the tube containing the sublimed mercury and tellurium being also ground up and assayed, tube and all. The result was that no gold was found in the latter, while the former showed a gold content which agreed with the roasting assays done in the ordinary roasting dish with subsequent potting, etc., and also with the direct crucible assay.

The water through which the gaseous products containing the last traces of tellurium, etc., were passed, was tested for gold, but yielded none.

The result of this experiment led the authors to suspect that the losses often attributed to mercury in the assay of telluride and other ores were either much smaller than is often believed, or were non-existent.

To test this, 50 gm. of the same rich ore was taken and cinnabar ground up with it, so as to give it a 4% mercury content. The tube experiment was then repeated, with the result that a bead of mercury was obtained from the distillation weighing nearly 2 gm. The mercury, which was coated and much contaminated with tellurium, was removed and dissolved in dilute nitric acid, but no gold was obtained. The rest of the sublimate, which contained mercury, but was composed chiefly of tellurium and sulphur, was carefully scraped out, dissolved in aqua regia and tested for gold, but, as in the case of the mercury, no gold was found.

In addition to these special experiments, a great many assays by roasting in the ordinary open dish and subsequent potting in the usual way were made, precautions being taken in every case to prevent mechanical loss through excessive draft, or during stirring. In the case of the rich ore before mentioned, the results obtained were always comparable one with another, and, although slightly higher assay results have been obtained from this ore in one or two cases by other methods, the repetition of the particular method by which the higher result has been obtained has always failed to give again this high value, and the great majority of other methods, which the authors have tried, have given lower results. These few exceptionally high results, such as occasionally occur with all specially rich ores, can only be attributed to sampling, although the ore actually employed appeared to be perfectly uniform, and every precaution was taken in sampling.

The value of the ore by the first tube experiment worked out at gold, 0·8473%, that is, 247·130 oz. =

247 oz. 3 dwt. per ton, and the value of the same ore, taking the mean result of six very concordant open dish roasting assays, was 0·8463%, and again on four more concordant roasting assays, 0·8472%. This shows, in the first case, a difference of only about 6 dwt. per ton as compared with the 'tube' assay, and in the second practically no difference.

In the assay of five poorer telluride ores ranging in value from 1 to 3 oz. per ton, the results obtained from the roasting method have commonly been slightly higher for gold than by potting the ore direct, although generally somewhat lower for silver.

From these results, it may be taken as proved that the loss by direct volatilisation is very slight under proper conditions.”—G. T. HOLLOWAY and L. E. B. PEARSE.—*Mining World*, Feb. 8, 1908, p. 252. (H. A. W.)

## METALLURGY.

**AGITATION OF SLIMED CONCENTRATES.**—“The solution in the agitation vats is made up to 0·1% KCN and the pulp is agitated, on an average, for 36 hours, the stirrers revolving at a speed of 12 r.p.m.; after agitation the percentage of KCN falls to 0·064% and the average consumption of cyanide amounts to 1·25 lb. per ton. Considerable trouble was experienced at the outset in dissolving the gold. The average time of agitation was 90 hours, and, although the resulting residues were good, the consumption of cyanide was high. This difficulty was overcome by the installation of a small air lift in each vat near the side to aerate the pulp. This method proved very effective, and brought the time of agitation down to an average of 36 hours, and decreased the consumption of cyanide from 2 lb. to 1·25 lb. per ton. The tonnages are arrived at by the specific gravity method, and check closely with the filter press tonnage. The extraction by cyanidation averages 96% in the agitators, but, taken on the raw concentrates, the extraction is equal to 81% of the original gold contents. The 16% extraction by amalgamation makes a total recovery of 97%. The residues, based on the raw concentrates treated, average for three months, 1·563 dwt. per long ton, or 1·395 dwt. per short ton.”—*Journal of Chamber of Mines, Western Australia*, Sept. 30, 1907, p. 853. (W. A. C.)

**DUST FROM ZINC GOLD SLIMES.**—“It is universally recognised that there is a loss of values, by dusting, during the preparation of zinc-box sludges by smelting treatment. A hood, suction fan, and mechanical dust-collector system was installed in the clean-up room at the Great Boulder Perseverance mine at a cost of £60, and subsequently operated as required during the period from October 1, 1906, to June 8, 1907. All screening and mixing of sludges was done under the pyramidal-shaped hood, fitted with side curtains reaching to within a few inches of the floor level. By means of the fan the dust was drawn upwards, sucked away through an opening in the top of the hood, and delivered to the dust-collector, from which the recovered material was removed at the end of the period indicated. The quantity of dust recovered was 16·3 lb., containing 29·756 oz. of fine gold valued at £126·46. The total weight of sludge handled during the eight months period was 20,784 lb. and from this 46,790 oz. of fine gold were recovered by smelting. Thus the recovery of gold from the dust, though insignificant when compared with the total values handled, justified the cost of installation of the system.”—*Journal of Chamber of Mines, Western Australia*, June 29, 1907, p. 727. (W. A. C.)

**USES OF TITANIUM.**—"Experiments with titanium in cast iron show that it adds greatly to the strength, and ferro-titanium is now being manufactured at a number of places in this country and in Europe for this purpose. It is said to add also greatly to the tensile strength and elastic limit of steel, and though much secrecy is maintained, it is known that titanium is used by various firms; and it seems probable that some steels, imported as containing vanadium, are really titanium steels.

If titanium will add to the good-qualities of steel and cast iron, as its advocates claim, there would seem to be a large future for it, owing to its plentifulness and cheapness, in sharp contrast with the comparative rarity and high price of other metals used for the same purpose, such as vanadium, molybdenum, and tungsten.

Experiments are now being conducted by several firms in an effort to make incandescent electric lamps with titanium filaments. The problems connected with the reduction of titanium to a metal are so difficult, however, that the lamps have not yet been extensively placed on the market. The melting point of titanium is very high; private experiments are said to have shown it to be about 200° C. higher than tungsten, with a higher electrical resistance than that metal, so that its efficiency in an incandescent electric lamp is considerably higher.

Titanium as a component of rutile ( $TiO_2$ ) and titaniferous magnetite is used in the electrodes of arc lamps, and titanium carbide is now coming into use for the same purpose. When one electrode is made of these substances, a block of carbon is used for the other.

Titanous chloride ( $TiCl_3$ ) has been used to some extent in dyeing, as a mordant, but is said to have little, if any, advantage over stannous chloride for the purpose. Titanous sulphate  $Ti_2(SO_4)_3$  is now being used as a stripper and mordant, and titanous potassium oxalate is used as a yellow dye and mordant in the dyeing of leather.

A new use for titanium as a detector of minute quantities of fluorine has been worked out by Mr. George Steiger, of the United States geological survey chemical laboratory. A solution of titanium sulphate  $Ti(SO_4)_2$  is oxidised by hydrogen peroxide, and the degree of decolorisation of the resulting straw-coloured liquid by fluorine is used to determine the quantity of fluorine present. Very small quantities of fluorine can be determined in this way.

In porcelain tile, rutile is used to give a soft, beautiful yellow colour. It is also used to colour artificial teeth. For this last use, however, only the purer grades of rutile can be employed. The entire demand for titanium ore is still small.—FRANK L. HESS, *Mineral Resources of the United States*.—*London Mining Journal*, Jan. 4, 1908, p. 25. (A. R.)

**USES OF ARSENIC.**—"Arsenic trioxide  $As_2O_3$ , 'white arsenic,' or 'arsenious acid,' is what is ordinarily meant by the general term of 'arsenic.' It is used as an insecticide in the preservation of hides and stuffed skins of animals and birds, in sheep and cattle dips and arsenic soap, and as a wood preservative, where it also prevents fungoid growth. It is used to prevent the growth of weeds and grass, in the making of some enamels, and many tons are used in the manufacture of glass. It is used as a fixer, or conveyor, of aniline colours in calico printing, and also enters into the manufacture of the dyes themselves. The use of arsenic as a poison for undesirable animals is familiar.

The greatest demand for arsenic is for making Paris green, used as a pigment, and as an insecticide on plants. Its toxic effect limits the former use. If pure, it is comparatively harmless to plant life, owing to its insolubility, but the trioxide is often present in some quantity, and if Paris green is then used largely it is fatal to plants, especially in dry climates, where it is not soon washed off by rains. Paris green is a copper aceto-arsenide, and should contain:—

	Per cent.
Arsenic trioxide ( $As_2O_3$ )	... 58·65
Copper oxide (CuO) ...	... 31·29
( $C_2H_3O$ ) ...	... 10·06

It is made by boiling verdigris, a mixture of basic acetates of copper, with arsenic trioxide. Scheele's green, a copper-hydrogen-arsenite, containing:—

	Per cent.
Arsenic trioxide ( $As_2O_3$ )	... 52·32
Copper oxide (CuO) ...	... 42·37
Water ( $H_2O$ ) ...	... 4·81

is also used as an insecticide.

London purple, also much used as an insecticide, is a bye-product in the manufacture of aniline dyes, and is thought to be composed mainly of calcium arsenite with some colouring impurity mostly rosaniline arsenite. Besides London purple, there are two other preparations—Paris and English purple, which have practically the same composition. London purple analysed by the New Jersey experiment station gave this analysis:—

	Per cent.
Water ( $H_2O$ ) ...	... 3·27
Arsenic trioxide ( $As_2O_3$ ) ...	... 41·44
Lime (CaO) ...	... 24·32
Alumina $Al_2O_3$ }	... 3·37
Iron oxide ( $Fe_2O_3$ ) }	
Sulphuric anhydride ( $SO_3$ ) ...	... 0·31
Dye, by difference ...	... 27·97

It is made by boiling a waste product consisting of dye and arsenious oxide with lime to make the insoluble calcium arsenite.

Arsenic also enters into red, yellow, and grey colours. In medicine arsenic is used as a tonic, and it performs a similar office in various stock foods and medicines. Realgar burns with a fine white light, and is much used in pyrotechnics, either as the natural sulphide or as the artificial product made by fusing together arsenic and sulphur.

In metallurgy arsenic has but little use. It makes lead hard, and arsenical work from the silver-lead smelters is sometimes used for shot. The presence of arsenic, unless in minute quantity, in brass or bronze, makes it brittle, though small quantities of less than 0·5% give cleaner castings and a greater ductility to Muntz metal.—FRANK L. HESS, *Mineral Resources of the United States*.—*London Mining Journal*, Jan. 4, 1908, p. 25. (A. R.)

**AIRLIFT AGITATION IN CYANIDING.**—"That fine grinding of most of the ores to be treated by the cyanide process is beneficial is generally admitted. The quartzose envelope of the metallic particles has to be broken in order to allow contact with the dissolving medium and so, departing from former practice, it has more and more become the endeavor of metallurgists to 'slime' all the ore and, abandoning the separation of the real slimes from the fine sands, agitating both together. Mechanical stirrers assisted by centrifugal pumps were used with some success, but it was found that the presence of even fine sands made the operation uncertain, and costly.

An improved method of agitating mixtures of slimes and sands has been invented by M. F. C.

Brown, manager of the Komata Reef Gold Mines in New Zealand and introduced by him on several important works in that country with remarkable success. Compressed air is used for this purpose, applied in a peculiar manner in tanks. The tanks are cylindrical in appearance but consist internally of a cylindrical part terminating in a cone with a 60° slope. The weight of the tanks and contents is transmitted by the continuation of the cylindrical shell, stiffened by vertical angle bars, to a double angle iron resting upon a solid foundation. The lower end of the cone is closed by a cast iron bottom. A doorway in the lower part of the shell gives access to the space around the cone and a man-hole in the latter facilitates inspection of the valves etc., when the tank is empty. The discharge of the contents, after agitation, takes place through a pipe near the bottom, provided with a valve, cock or plug.

The internal fittings of the tank consist of:—

1. A central tube called the airlift, of a diameter of about one-twelfth of that of the tank. It is open at both ends and extends from about 18 in. above the charging level to the same distance from the lowest point of the cone. It is supported by means of brackets of which there are three in each set.

2. An airpipe in the center of the lift, closed below and resting on the bottom of the tank, has at the level of the inlet of the airlift a 'sleeve' valve which prevents the pulp entering and allows the air to escape in an upward direction when the air-pressure exceeds that due to head of pulp in the tank.

3. A second airpipe outside the lift which serves to keep the pulp in gentle motion while filling or emptying the tank when the submergence is insufficient to allow the airlift to work.

4. An apparatus consisting of an annular casting surrounding the airlift, supported on the sides of the cone and provided with a number of pipes having sleeve valves discharging against the cone, the apparatus being connected to the air, and water or solution pipes. This arrangement is sometimes useful, when discharging, to wash down the sands which may have settled on the cone after the agitation is stopped.

The *modus operandi*, like the whole apparatus is very simple. The tank being filled with ground ore and solution, air is admitted through the pipe, which mixing with the contents of the lift, lightens the column inside and causes it to overflow, while fresh pulp runs in at the bottom and is in turn brought to the top, thus producing a perfect circulation which is kept up as long as the supply of air lasts. The initial air-pressure has to exceed that of the column in the airlift, but when once circulation has been established the pressure can be considerably reduced. At first also a greater quantity of air is required in order to obtain a sufficient velocity of circulation to remove all the pulp which may have packed on the cone during the period of filling. In commencing operations 50 lb. of air-pressure are used, but when the cone is clear of sand, the half of this pressure is sufficient. The quantity of free air consumed depends upon the proportion of slimes and sands, the degree of grinding and the viscosity of the pulp. In most cases 100 cu. ft. of free air per minute will keep the mass in lively motion and prevent the settling of sands on the cone. Large tanks, loaded with slimes only are agitated in New Zealand with an expenditure of less than 1 h.p. For a plant treating 100 tons per day of mixed sands and slimes a 10 h.p. compressor is sufficient.

The agitation by these means is perfect and very efficient aeration of the mass is obtained at the same

time. Even after weeks of rest, when the contents of the tank have packed to a hard mass in the cone, agitation can be readily started, and after about one hour the mass is in perfect circulation. Sands can be treated as readily as slimes, and as they fall through the solution more rapidly than the slimes they receive a more energetic treatment. It is clear that the dissolving action of the cyanide solution is much more rapid than when less perfect modes of agitation and aeration are used. In fact, 24 hours' treatment are generally sufficient, and extractions of 92% of the silver and nearly all the gold have been obtained in that time.

The first cost of a plant of this description is much less than when mechanical agitation of the ordinary kind is used, and the treatment charges are also considerably reduced. One man can do all the work connected with the agitation of a 100-ton plant. There is no danger of anything getting out of order, and no chance of the contents settling down."—*Mexican Mining Journal*, Jan., 1908, p. 20. (A. R.)

PROGRESS IN GOLD-ORE TREATMENT DURING 1907.—"It will not be possible for me to review the progress of gold-ore treatment for 1907 in as detailed a manner as is usual, owing to these notes being written while travelling some thousands of miles away from my base.

Fine sliming and the treatment of slimes is again the most interesting feature of the year's progress—even former exponents of coarse crushing are now converts to, and profit-takers of, modern slimes practice. From Eastern Asia to Western Australia and from South Africa to North America sliming methods are pre-eminent and have modified the other methods or processes employed in the industry, such as crushing to tube mills and roasting ores ground to very fine particles.

*Crushing*.—We have heard less this year of 'Pans v. Tube Mills.' Experiments in the Transvaal appear to have resulted in favour of the latter, and many more tube mills have been installed. With the heavy stamps previously referred to in these notes the Luipaards Vlei appears to have been able to output 8.5 tons per head per diem, but in Rhodesia I am advised that 10 tons per head is being obtained in more than one instance—crushing, however, to pans instead of to tube mills. In Rhodesia the practice differs from that on the Rand; with their much smaller installations pans are found to be more suitable, as being cheaper to efficiently install more convenient units and making less demand on power; consequently we hear of but few tube mills operating in Rhodesia. The pans thus used are of the West Australian type.

But tube mills have unquestionably been improved in their *modus operandi*. Linings now last much longer, and instead of having to lay off a mill every three weeks or so for a week for a new lining, a mill can be run for three to six months with a lining placed in position in from two to three days. 'Automatic' linings such as those of El Oro, where pebbles are supposed to fix themselves in rebates cast in iron liners, do not yet appear to have proved themselves entirely satisfactory, and changes have recently been made in the shape of the recesses, which now have their sides chamfered to each other.

The use of quartz or other reef matter to be crushed in place of imported pebbles has much increased, and this and the utilisation of reef matter and local chert or other hard rock in linings of the Barry or Brown type have very materially reduced the cost of operating tube mills. In Africa a mill is started with a

charge of pebbles containing, say, 10% of reef matter (stones), then in regular work it is found possible to reduce the daily feed of pebbles by 5%, adding reef matter (stones) to balance until the daily feed contains only, say, 10% of pebbles, the whole of the remainder being reef matter.

*Slime Treatment.*—It is in the treatment of the slimes produced by tube mills and otherwise that the progress of 1907 was most manifest. It may be regarded that the decantation process, which originated in Africa, and has so long survived there, has now had its day. Designed to treat material of low value, at that time not amenable to any method of treatment known to the industry, it resulted in profits being realised from material which otherwise must have run to waste. But the huge and expensive plant and low extraction had to give place to more modern automatic methods capable of cheaper equipment and higher saving, and already such well-known workers as J. R. Williams, W. A. Caldecott, E. J. Way, and H. S. Denny have been looking into methods employed elsewhere, notably the Ridgway, Butters, Burt and Merrill.

In my notes of last year reference was made to the new Denny plants at the Meyer and Charlton and the New Goch. The Denny brothers severed their connection with these two mines shortly afterwards, but in spite of their absence filter-pressing appears to be regarded as successful by the heads of the group, although much doubt has been expressed as to the wisdom of running cyanide solution through the mortar boxes. George Albu has criticised this as making it difficult to obtain the assay value of the mortar-box product, but I assume this point would not be regarded as serious if Mr. Albu were convinced he were obtaining a higher recovery of his gold. This discussion has been fully dealt with in another place.

On the Rand, however, the Denny methods have not been followed. Rhodesia on the contrary, having investigated the results, has had quite a Delne filter-press boom, plant after plant having been installed during 1907.

*Vacuum Filters.*—But undoubtedly filter-pressing even with Delne presses must give way before the suction filters of which so much has been heard in 1907. Cheap and efficient, the daily tonnage handled by vacuum filters is increasing by leaps and bounds.

Of the various vacuum filters on the market I must unhesitatingly refer first to the Ridgway. This differs from the filters of the basket type in being more rapid, working with thinner cakes, giving better washing and emptying itself (automatically) in more cleanly fashion, than the basket filters. Working as it does with the whole cycle of operation complete in 60 seconds on normal slime pulp, it will be seen that difficulties of keeping material on cloths during transit of frames or during emptying and filling of tanks, which arise in other methods, are entirely avoided, while the washing of so thin a layer ( $\frac{1}{4}$  to  $\frac{3}{8}$  in. thick) is much more rapid and complete than the washing of a cake of double or treble the thickness. The Ridgway is thus able to work with a much smaller area of filter cloth for any given capacity.

During 1907 a 560 ton-a-day plant was put into successful operation at the Great Boulder in West Australia, and I understand another plant of similar capacity has already been started. Plants of similar and smaller capacities have been erected (or are in course of erection) in Mexico, India, South Africa, Eastern Asia, etc., and altogether we hear much of the Ridgway filter at the present moment.

The official figures of the West Australian Chamber of Mines show Ridgway to be working under expensive local conditions (purchased power, dear labour and supplies) far under 4d. per ton treated; in Africa the cost should be little more than half this.

*Butters-Cassel v. Moore.*—The Butters-Cassel filter has been installed very largely in Mexico as well as in the United States, not perhaps because it was the best vacuum type of filter—the recently published correspondence shows the pioneer Moore to be at least the equal and probably the superior—but because it was pushed by a man of repute and of great energy who was recommending the best thing he could get hold of.

But in the recent correspondence in the technical papers it appeared to me that the only persons writing in favour of the Butters-Cassel filter were those who were, or had formerly been, associated with Butters, while the Moore was recommended time and again by persons in no way associated with Moore, who had installed it after investigation and who were apparently in no way connected—not even by paying royalties—to Moore or his associates. Certainly if the Moore process had been run with half the energy, experience, knowledge and skill of the Butters I cannot imagine the latter type of filter to be employed at all.

For it is obvious that the hoisting of a basket of frames is a much neater and better and quicker expedient than the pumping in and out of pulp and solution or wash, each successive charge being mixed with the residue of the previous charge whether of pulp or of solution—if used—or of water, and it is within the knowledge of men experienced in slimes filtration that the maintenance of a vacuum in a basket half immersed—or partially immersed for, say, four periods of 10 minutes each during each cycle of operations—does not make for good results. The lower portion of the cake is being thickened by pulp deposit while the upper is cracking from the rush of air, an unequal cake is formed and the washing must be unequal and there seems to be a very great waste of water in discharging the cake after filtration. From the published correspondence it seems to be laid down without serious contradiction that the Moore is more accessible and open to inspection, and that it does more work for a plant of given cost or in a given time.

Further modifications of the basket filter have appeared in the form of enclosed filters worked by pressure or vacuum, such as the Blaisdell, Burt, and Kelly. This form of filter is scarcely new, and when tried some years ago proved itself unhandy, liable to freeze or choke, difficult to wash thoroughly and difficult to discharge—in a word, it seems to be an attempt to work in the dark. Of course, modifications of the standard immovable enclosed type are made; one runs the frame into a cylinder, or runs the cylinder away from the frame; this makes the operation more accessible, but so far does not appear to be as successful as the pioneer basket type.

*Merrill's Plant.*—Reverting, however, to filter presses, Merrill has at last got his big plant at work after three years' labour. It appears now to be working to nearly its full capacity—according to my recent information—and to be treating slimes at a remarkably low rate. Advantage has been taken of every natural condition to make a successful working plant, and using static pressure to fill the presses and wash their charges, the Merrill costs are probably as low as any slimes treatment method yet put into practice. But my information is that with such a

method the satisfactory treatment and hydraulic discharge of thick cakes in huge filter-presses is possible only with crystalline or granular slime pulp and with a great waste of water. In a word, it appears that successful results may not be anticipated by the process on ordinary slimes pulp in ordinary gold fields under ordinary economic conditions; the thorough washing of 4-in. cakes is feasible only with crystalline, granular or readily permeable slimes pulp.

The successful handling of slimes pulp has also caused more attention to the solution of the gold contained therein. Last year I referred to the system of treatment based on feeding solution or wash water at the bottom of a tank, the slime contents of which were in a state of gentle agitation, the idea being an overflow of clear solution containing the gold content of the charge. W. L. Holms of Mexico City (late of Consolidated Gold Fields of South Africa) brought this to my notice some years ago, but it did not appear to make headway. Bewick, Moreing & Co., were working it in 1906 in West Australia, but evidently without commercial success; in 1907 Adair Usher boomed an apparently similar process in South Africa. Possibly some metallurgist at the Geldenhuis Deep or elsewhere may make something tangible out of the idea, but it looks as though the only result would be to call attention to the extra extraction obtained by the increased agitation of the pulp, and in this connection the work of the Brown agitator ought to be investigated carefully.

*The Brown Agitator.*—This apparatus, introduced into New Zealand at the Komata Reefs, depends on the principle of the lessened specific gravity of a centre column into which air has been introduced at just such a pressure as will overcome the weight of the column of water at the point of introduction. There is no question of a jet of air circulating solution on the principle of the injector, but merely the physical lightening of the weight of a column by the displacement of a small proportion of the water by air.

Brown uses long narrow vertical (cylindrical) tanks of, say, 40 × 10 ft. or 55 × 15 ft. with a centre column of 1 in. diameter per 1 ft. of tank diameter. Into this central tube or column the air is introduced and agitation is so effective as to lift stones at, perhaps, the smallest horse-power per ton agitated of any efficient mechanical agitator. The power taken appears to be about 2½ h.p. per 50 tons (slime) pulp charge, which is the slime content of a 40 × 10 ft. tank. By this method the advantage of the accelerating action of air agitation in certain ores is obtained at a small cost.

The use of these tanks has been offered to the Waihi—the manager of which speaks very highly of these agitators—and Waihi Grand Junction in New Zealand, and a number have been installed in Mexico under the name of the Pacluca agitator.

*Cyanidation of Silver-Gold Ores.*—The progress of cyanide treatment for the silver-gold ores of Mexico has been one of the features of the slime boom. Chihuahua has been adopting cyanide with avidity, and Pacluca appears to be now coming into line. Of course El Oro has done pioneer work and Guanajuato has also been in the forefront.

In almost all instances the practice is similar, tube-mill sliming and treatment of the slime pulp at first by decantation, but now by basket or Ridgway filter. Very high extractions are claimed, but having regard to the natural refractoriness of silver sulphide ores—in some instances, as at Chincates associated

with pyrites, chalcopyrite, galena and blende—one is impressed by the necessity of having recourse to the most modern practice, whether for getting the silver and gold into solution or for recovering the solution from the pulp for precipitation of the precious metal.

There is but little new either in roasting or concentration on gold ores this year. Merton and Edwards still hold the field for roasting, and the many flotation processes more or less (unfortunately with more of the less) successfully applied to the zinc-lead ores at Broken Hill, such as the Potter, Delprat, De Bavay, Cattermole and Elmore, do not yet appear to be successfully running at any gold mines known to me; although it is thought that the mechanical and metallurgical skill at the back of the Elmore process will bring it to the forefront during the coming year (1908).—ALFRED JAMES.—*Engineering and Mining Journal*, Jan. 4, 1908, p. 17. (K. L. G.)

## Mining.

**Mining and Metallurgy in 1907.**—“The study of the origin of ore deposits has a practical bearing on prospecting and on the valuation of mining properties. Theoretical studies should therefore be welcomed by practical men. It is agreed that the metals found in the earth's crust come in various ways from a nucleus, with a specific gravity exceeding 5.5, in the centre of the earth. This central nucleus may be compared to a bath of molten metals, the slags of which constitute the eruptive rocks. The magmatic segregation or igneous crystallisation which occurs within these eruptive rocks during the process of cooling is the cause of the formation of many ore deposits of value, including iron, manganese, chromium and nickel. The most faithful reproductions of ore deposits yet produced have been published as coloured photographs by Mr. B. Baumgartel. They were obtained in a most ingenious way. Directly a suitable fresh lode surface in the Harz mines was opened up, the author took a photograph of it with magnesium flashlight. The photograph was developed the next day, and on the day after the author again went underground and indicated with coloured pencils the distribution of the ores and lodestuff in the photograph, the face being illuminated for the purpose by a miner with an acetylene lamp. The final colouring was executed subsequently.

Observations on underground temperature have been made in the Pas-de-Calais coalfield. The temperature increases 1° C., not with every 31 metres, as has hitherto been thought, but only with every 56 metres. At a depth of 1,200 metres the temperature in a borehole was 35° to 40°. Various improvements in deep-boring appliances are recorded, and an electromagnet has been successfully used in recovering a broken drill from the bottom of a borehole. Serious attention has been devoted to the surveying of boreholes, and several ingenious instruments have been devised for the purpose. In one of them the magnetic needle is clamped by the air pressure caused by the detonation of a cartridge. Mr. L. H. Cooke recently questioned the general reliability of borehole surveying based upon the use of the magnetic compass, on the ground of the polarisation of the boring rods by constant vibration.

Shaft sinking has received an unusually large amount of attention. Discussing special methods of shaft sinking, Prof. H. Louis stated that the largest amount of water successfully sunk through was at Horden colliery, where 9,250 gallons of water at a depth of 540 ft. were dealt with.

In explosives and blasting continued improvements are to be noted. On the occasion of the visit of the Iron and Steel Institute to the Styrian Iron Ore Mountain, which was ably described in a paper by Prof. H. Bauerman, a remarkable display of blasting took place. No less than 1,602 boreholes, of a total length of 6,050 ft., were loaded with 3,208 lb. of dynamite and fired simultaneously, bringing down 58,200 tons of ore.

The preservation of timber for use as mine supports continues to receive attention. Careful experiments made in the United States have shown that the most successful treatment, economically considered, is with creosote, the timber being immersed in an open tank, without pressure, in successive baths of hot and cold liquid. Reinforced concrete has been used to replace timber for roof supports, and success has attended the introduction of the Sommer pit props, made of weldless steel tubes, in Austria. On the Rand a strong preference for steel for constructional work is being manifested. Opinion in this country is less favourable to steel for underground requirements, owing to its shorter life underground as compared with iron.

Practical steps in the direction of winding by electricity have been taken at collieries in South Wales and at the mine of the Tarbrax Oil Co., Ltd., in Scotland. Experience tends to show that, compared with electricity, winding with the steam engine is cheaper in installation and in working costs, and consequently that when steam raising plant is available the steam winding engine is to be preferred. The use of turbo-generator sets is rapidly increasing, and Parsons' steam turbines have been installed at various collieries.

In ore dressing the improvements introduced during the year have been in details rather than in principle. The steam stamps have been discontinued everywhere except at the Lake Superior copper mines. The Harz jig continues to hold its own, and for fine concentration various modifications of the Wilfley table have been introduced. Magnetic concentration is making steady advances, considerable attention having been given to it for the separation of tin wolfram ores. Excellent results are being obtained in Scandinavia with the briquetting of iron ore. In gold extraction the use of tube mills has extended, leading to better returns from producing mines, though a careful computation of added capital, charges, and working costs in every case is necessary before their installation can be justified.

In metallurgical matters the principal changes seem to have been mainly in the direction of modifying accessory details in existing processes. The application of electricity to driving reversing rolling mills in Austria, and Mr. Sandberg's method of manufacturing silicon steel rails are the most striking novelties to record. Considerable progress has been made in the electric smelting of iron, and notices published in these columns from time to time during the year have recorded the results of successful experiments in various parts of the world. Aluminium is now manufactured by electric methods at a number of works controlling over 84,000 h.p., and there are some 34 electrolytic copper refineries at work in Europe and America. The process of pyritic smelting, where the sulphur in the ore is utilised as fuel in the blast furnace in the treatment of ores containing precious metals but poor in copper, continues to make way in districts where fuel is expensive. Hot blast has been abandoned at Mount Lyell, Tasmania, after four years' trial. It has also been given up at Ducktown, Tennessee, and at the

Copper Queen mine, Arizona. The smelting of antimony ore has received less attention; and in lead smelting there has been a tendency to revert to older methods. The reinforced concrete dust chambers have fallen into disfavour, and the lime roasting process does not appear to have been an unqualified success.

The greatest honour conferred on the mining industry during the year was, however, the institution by the King of a new medal, for bravery in mines, to be known as the Edward medal.—*London Mining Journal*, Dec. 28, 1907, p. 793. (A. R.)

#### MISCELLANEOUS.

THE INTERACTION BETWEEN MINERALS AND WATER SOLUTIONS.—"The natural silicates precipitate the metals from solutions of salts, while at the same time the bases of the silicates are dissolved in quantities nearly equivalent to the precipitated metals. The bases most commonly replacing the metals in these processes are potassium, sodium, magnesium, and calcium. Where exact equivalence is wanting, it is attributable either to solubility of the mineral in pure water or to the precipitation of basic salts.

The metals are precipitated as hydroxides or basic salts (in the case of cupric sulphate, for instance, as a basic sulphate similar to brochantite or langite) with more or less metal silicate.

The specific materials on which work was done are albite, amphibole, augite, biotite, enstatite, garnet, clay gouge, kaolin, microcline, muscovite, olivine, orthoclase (3), prehnite, shale, talc, tourmaline, and vesuvianite, with cupric sulphate solution; and orthoclase with salts of sodium, potassium, magnesium, calcium, strontium, barium, manganese, iron, nickel, copper, zinc, silver, gold, and lead. Experiments were also made on the action of kaolin on solutions of salts of zinc and iron, and of glass, fluorite, and pyrite on cupric sulphate, and of carbonic and sulphuric acids on orthoclase.

Absorption, a mechanical surface attraction, plays a comparatively insignificant part, if any, in the retention of copper by kaolin. If absorption is lacking in the case of kaolin, it seems reasonable to assume that it is lacking in the case of other silicates also.

Under the conditions of experiments described the precipitation of copper caused by orthoclase does not differ materially in quantity from that caused by pyrites.

A dilute salt solution decomposes orthoclase and dissolves its constituents to about the same extent as does a saturated solution of carbonic acid. Both are much more efficient in this respect than water.

Orthoclase (and presumably other alkali silicates also) accelerates the oxidation of ferrous sulphate by the oxygen of the air. A similar influence may be expected in other oxidation reactions that take place more readily in the presence of alkali than in the presence of acid.

It is worthy of note that the iron of ferrous silicates is not found in solution replacing copper after the precipitation of the latter. This was also true of the iron in a clay gouge containing considerable ferrous carbonate.—EUGENE C. SELLIVAN, U.S. Geological Survey, Bulletin No. 312. *London Mining Journal*, Dec. 28, 1907, p. 783. (A. R.)

THERAPEUTIC APPLICATION OF ARSENIC COMPOUNDS.—"Numberless experiments have been made with the view of obtaining arsenic compounds, which,

while giving the beneficial effects of this element, would be more readily tolerated by the system than either arsenious oxide or Fowler's solution (potassium arsenite). Sodium dimethylarsinate has been suggested in this connection, but its use is attended with one grave disadvantage—namely, the tendency which this salt possesses of giving rise to the very malodorous cacodyl compounds. Moreover, this substance is nearly as poisonous as arsenious acid itself. A much more satisfactory compound has recently been employed under the name of atoxyl. This substance, which is prepared by heating together aniline and arsenic acid at 200° C., was formally supposed to be the anilide of arsenic acid. Its extensive use in therapeutics has, however, led to the revision of its chemical constitution, and Ehrlich and Berthelm have recently shown that the compound is in reality a new organic acid—arsanilic acid,  $\text{NH}_2 \cdot \text{C}_6\text{H}_4 \cdot \text{AsO}(\text{OH})_2$ . The drug is usually exhibited in the form of its sodium salt

$\text{NH}_2 \cdot \text{C}_6\text{H}_4 \cdot \text{AsO}(\text{OH}) \cdot \text{ONa} \cdot 4\text{H}_2\text{O}$ , which is readily soluble in water, giving a neutral solution (*Berichte der deutschen chemischen Gesellschaft*, July 20, 1907). Forty times as much arsenic can be administered in the form of atoxyl as can be tolerated in the condition of Fowler's solution. The drug is now being used for a variety of maladies, and has given very promising results in cases of sleeping sickness.—*Times Engineering Supplement*, Dec. 18, 1907. (J. A. W.)

## Reviews and New Books.

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

### MINES AND MINERALS OF THE BRITISH EMPIRE.

By RALPH STOKES, late Mining Editor, *Rand Daily Mail*, Central News Agency, Johannesburg, 2s. net. (London: Edward Arnold.)

This book is a description of the historical, physical and industrial features of the principal centres of mineral production in the British Dominions beyond the seas. It is illustrated by a large number of excellent photographs, and fortified by several well digested and clearly presented groups of statistics. Undoubtedly, the keynote of this extremely interesting account of the mining activities of the Empire is revealed in the care, with which its author has investigated the labour conditions in all the districts covered by his tour of 35,000 miles. In this country, and at this time, no work on mining will better repay a careful study than this collection of the facts of mining conditions and their action on the problems of social economies. It is fitting, therefore, that the first chapter is devoted entirely to mine labour, and the various elements employed from Sudbury, Ontario, to Waihi, in New Zealand. Contrasts of great economic importance are revealed in the study of so wide a field, and the difficulty and complexity of the questions involved are treated with a frank thoroughness which renders this chapter as valuable as any of the following twenty-five.

Very many of the mines visited and clearly brought to our notice deal with other than the precious metals, and mention may be made of the vivid descriptions of tin working in the Malay States, nickel in Ontario, copper in Tasmania, and graphite in Ceylon. Lead, zinc, mica, asbestos and precious

stones are all represented in these pages, and nearly every part of the Empire has been laid under contribution. The graphic accounts of the districts visited bring their features distinctly before the reader, and it is a further illustration of the modern shrinkage of world space, that so much has been seen and such lengthy journeys have been undertaken in so short a time. As a consequence, the result is a work completely up to date and, therefore, of exceptional value to the mining engineer, the investor and the student. It is worthy of a place on any bookshelf, and has so enhanced the reputation of its author as to engender a desire for many successors. (H. A. W.)

THE METALLURGY OF IRON AND STEEL. By BRADLEY STOUGHTON. 9 × 6, viii. and 509 pp. \$3, post paid. (New York: Hill Publishing Co.)

As stated in the preface, this book covers every branch of the metallurgy of iron and steel, but the attempt to deal fittingly with such a vast subject in a limited space does not appear to have resulted altogether satisfactorily.

The great fault appears to be a want of proper balance in the treatment of the various sections; certain portions are so curtailed as to be quite elementary in character, while others are of real use to students and technical men; for instance, the construction of the blast furnace itself receives about nine lines! This and the various blast furnace accessories take up two and a half pages of type, while rolls are dealt with in detail in about twenty pages, and iron and steel founding in fifty-five pages.

Further, want of care in compilation is seemingly shown in several erroneous statements, notably with regard to Sheffield practice in crucible cast steel manufacture, the explanation of the heating of metal in the Kjellin furnace, and the nomenclature of silicates.

In order to make the rest of the book intelligible to those without the necessary knowledge of chemistry and physics, a brief section of about 30 pp. is introduced to supply the defects, in these directions, in the education of readers, and if thoroughly mastered would, no doubt, go a long way towards this end; whether this is likely to be the case, however, is open to serious doubt, and to properly understand anything of modern metallurgy a really good grounding in these sciences is essential. This section is marred by one or two unfortunate examples also.

On the other hand, the principles of the manufacture of pig iron and its purification, and the various methods of manufacturing steel are very well treated, as also is the mechanical treatment of steel and iron and steel founding; while the chapters on the solution theory of iron and steel, the constitution of steel and cast iron, and the heat treatment of steel are really admirable; if only for these alone the book is well worth reading, bringing, as it does, this constantly widening part of the subject right up to date.

The value of the book is largely increased by useful bibliographies at the end of the chapters, and it is copiously and excellently illustrated and well printed, as indeed are all those produced by this firm.

It is disappointing, however, that what is, in the main, an excellent book, should be marred in the directions noted, and scarcely, therefore, does justice to the reputation of its author. (G.H.S.)