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Proceedings
AT
Ordinary General Meeting, September 17, 1910.

The Ordinary General Meeting of the Society was held in the Lecture Theatre of the South African School of Mines, on Saturday evening, September 17th, Prof. G. H. Stanley (Vice-President) in the chair. There were also present:

43 Members: Messrs. E. F. Alexander, R. Allen, Tom Johnson, A. Richardson, H. A. White, Prof. J. A. Wilkinson, J. Littlejohn, A. F. Crosse, E. H. Johnson, H. R. Adams, E. Blume, B. V. Blundun, A. J. Bowness, F. W. Cindel, M. H. Coombe, J. Daniel, R. Gascoyne, J. Gray, A. B. Inglis, J. H. Johnson, G. A. Lawson, H. J. Lea, J. Lea, G. Melvill, J. T. Mitchell, S. Morison, W. Nicklin, F. D. Phillips, W. S. V. Price, J. F. Pyles, G. A. Robertson, H. Ross, G. H. Smith, T. F. Smith, B. C. T Solly, J. A. Taylor, A. Thomas, C. F. Thomas, O. Tonnesen, Chris. Toombs, A Wilkinson and L. J. Wilmoth.

17 Associates and Students: Messrs. A. R. Adams, C. A. Damant, F. E. Doble, B. Hasserüs, B. W. Holman, W. J. R. Hunter, A. King, G. W. Leach, R. Lindhorst, G. H. Little, H. Lomberg, H. J. v. d. Merwe, F. J. Pooler, R. E. Sawyer, G. L. Shearer, H. Stadler, and L. A. Womble.

12 Visitors, and Fred. Rowland, Secretary.

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

NEW MEMBERS.

Messrs. J. Gray and Andrew F. Crosse were appointed scrutineers, and after their scrutiny of the ballot papers, the Chairman announced that all the candidates for membership had been unanimously elected, as follows:—

ALLEN, VIVIAN STEVENS, P. O. Box 369, Salisbury, Rhodesia. Mining Engineer.

BRODIGAN, VINCENT FREDERIC, East Rand Proprietary Mines, Ltd., P. O. Box 24, East Rand. Mining Engineer.

JONES, WILLIAM, P. O. Box 42, Randfontein. Mine Captain.

ODGERS, WILLIAM HEARLE, Village Deep, Ltd., P. O. Box 1064, Johannesburg. Cyanide Foreman. *(Transfer from Associate Roll.)*

SCHWARZ, ADOLF, Meyer and Charlton G. M. Co., Ltd., 10, Janie Street, Jeppestown, Johannesburg. Cyanider.

STEVENS, THOMAS BARNBROOK, Oroya Links, Ltd., P. O. Box 57, Kalgoorlie, W. Australia. Metallurgist. *(Transfer from Associate Roll.)*

TRIGGS, JAMES THOMAS, Village Deep, Ltd., P. O. Box 1064, Johannesburg. Cyanider.

The Secretary: Since the last meeting of the Society the following have been admitted by the Council:—

As Associates.—

BRINSDEN, FREDERICK GEORGE, Kalgurli G. M., Ltd., Fimiston, West Australia. Metallurgist.

THORNE, THOMAS LLEWELLYN, Geldenhuis Deep, Ltd., P. O. Box 54, Cleveland. Cyanider. *(Transfer from Student Roll).*

TUCKER, PERCY ALEXANDER, Consolidated Gold Fields Laboratory, P. O. Box 108, Germiston. Metallurgical Chemist.

GENERAL BUSINESS.

PRESIDENT INDISPOSED.

The Chairman: I am sure you will be sorry to hear that Dr. Moir is not very well to-night, and is therefore unable to take the chair. Consequently it falls upon me, as the only Vice-President present, to occupy that position.

THE DISTRIBUTION OF PULP FOR TUBE MILLING.

By G. A. ROBERTSON (Member).

Mr. H. Stadler in reply to the discussion before another technical society on his paper, "The Computation of the Crushing Efficiency of Fine Grinding Machines," stated with reference to tube mills that there was still "ample room for further great improvements, especially with the regulating and maintaining of the recognised most suitable conditions of the feeding." I do

not know what the author had in his mind at that time, but no doubt during his investigations into tube mill efficiency on behalf of the Mines Trials Committee he found room for a more even distribution of the coarse products. If the tube mill is most efficient on particles of ore up to 0·27 in. diameter it is only reasonable to suppose that in order to get maximum efficiency from a battery of tube mills that each tube mill should have an equal share of these coarse particles or its *pro rata* share in accordance with its grinding capacity for the time being.

Whilst various methods might be employed for distributing the pulp between tube mills, including the method suggested by Mr. J. E. Thomas,* none of these can be considered satisfactory under all operative conditions, and the difficulty becomes more apparent when, say, from one point we may have to divide a pulp equally between from 10 to 15 tube mills. In order to overcome these difficulties the writer has prepared the accompanying sketches, which will suffice to show the practical utility of the scheme herewith advocated. In cone classification for tube milling we may not get an equal grading analysis from every foot of the cone perimeter, but such grading is much more equal when the underflow is shut off, and taking the necessary precautions with regard to the vertical inflow, etc., there is no reason why an equal grading of pulp at equal distances on a cone perimeter should not be an accomplished fact.

If the pulp for distribution were elevated by a tailings wheel, naturally such pulp would have to be given a vertical fall into the pulp distributor; on the other hand, however, where a pump acted

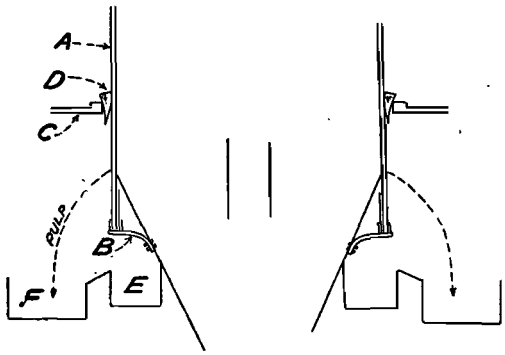


FIG. I.—Elevation of Cone Pulp Distributor.
A—Pulp dividers made of wood. B—Channel iron bracket (with guide plate on inside) fixed around cone. C—Circular platform. D—Wood wedge. E.—Spill launder. F.—Peripheral launder common to the overflows from cone. G.—Pulp inflow from launder.

* See discussion G. O. Smart, "The Tube Mill Circuit and Classification," June *Journal*, p. 453.

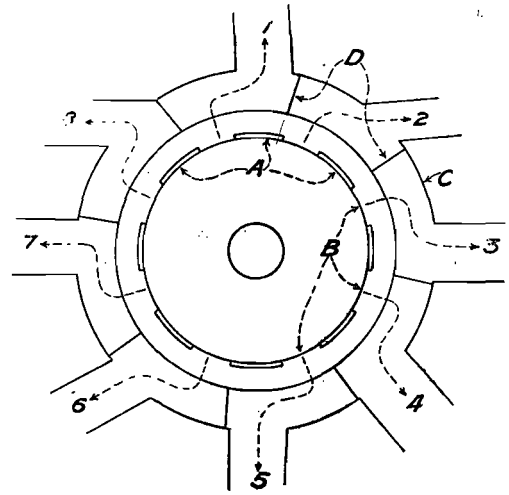


FIG. II.
A—Dividers in position. B—Divided overflows feeding into C—Common launder. D—Guide gates.

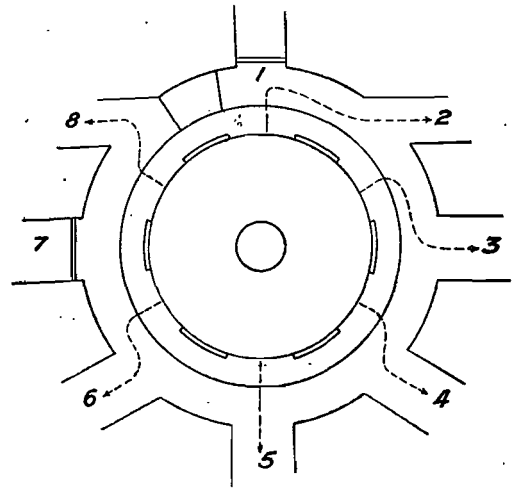


FIG. III.—Shows that only six tubes are in action, Nos. 1 and 7 being idle and overflows again divided.

as the elevator it might be convenient to run the pump column up through the cone, the column discharging itself above the cone overflow level. This latter method would be adaptable to diaphragm cone classification as advocated by Mr. G. O. Smart,* which, in new designs, effects a considerable saving in height of elevation of pulp.

Since the preparation of these sketches the writer has seen illustrations of another pulp distributor†, one objection to that device being that it cannot divide pulp equally if one or more tube mills were to lie idle, but with a few alterations in the design it could also be made suitable to all conditions.

* See this *Journal*, February, p. 282.
† See this *Journal*, July, p. 38.

By this method any number of tube mills can be in or out of action, and the pulp evenly divided amongst the tube mills running.

Mr. Fraser Alexander (*Member of Council*): I should like to propose a hearty vote of thanks to Mr. Robertson for his valuable though short paper. I am sure we all feel very much the want of an adequate distributor leading to the various tube mills. The author does not say whether this apparatus shown in the diagrams is actually at work and doing well, and where it can be inspected. I should like to ask him whether it is entirely theoretical or in actual operation? I think it will be agreed that it is a step in the right direction, as it is a point on which we often meet with a great deal of trouble.

Mr. G. A. Robertson (*Member*): The idea is entirely theoretical. At the same time it is just about guaranteed in practice.

The Chairman: I have pleasure in seconding this vote of thanks. It is one of those little papers which are very useful, and if it helps to solve this practical difficulty of the final division of pulp, the Soc ety and those members supervising the running of tube mills will be under some indebtedness to the author. I do not quite follow personally the course of the pulp through this distributor. I do not see how it gets into this particular device. Perhaps the author could illustrate it on the blackboard?

Mr. G. A. Robertson: The pulp has a clear fall into the common launder from whence it then finds its outlet. You know that with a conical classifier if a piece of wood be placed against the overflow the pulp is divided, so in this case the inflow to the distributor is down the vertical pipe as shown in Fig. I., then between the dividers. It follows, of course, that if several tube mills adjacent to one another are out of action that a few short portable launders would be necessary to receive the pulp and convey it to the outlets on the common launder. Broader dividers would have to be kept in stock in order to meet these requirements.

The Chairman: I think the matter is made clear now.

NOTES ON BATTERY PRACTICE.

By A. R. Stacpoole (Member).

Mr. H. A. White (*Member of Council*) read the following paper in the absence of the author:

The mill manager and amalgamator are rapidly having their power and responsibility usurped by the reduction officer and cyanide manager, but have only themselves to blame for it.

From the early days of the Rand the mill man looked on it as his first and almost only duty to crush, and rush a big tonnage through, paying little or no attention to the extraction of gold; the idea being, that what is not caught in the mill will be caught in the cyanide works. Had the amalgamator looked on the matter of extraction as of equal importance with that of crushing, he would be in a much stronger position to-day.

The first and principal duty of every amalgamator should be *to see that no free gold, amalgam or mercury goes over the end of the amalgam plates.* As a rule the free gold and amalgam which finds its way to the cyanide works requires a much stronger solution and a longer treatment to dissolve it than it gets, so that a large portion of it goes to the dump thereby enriching the residue.

Collection of Sand from Amalgam Plates.—This sand is commonly known as "black sand." The clumsy and imperfect method used in collecting the sand calls for a better and more careful way. In many mills the method used to-day to collect plate sand is as follows:—

The water is shut off, and stamps hung up; then a very imperfect dam is made across the foot of the plate with brushes; the sand is hosed down till stopped by the brushes, but it frequently happens that the flow of water is so great that a large quantity of sand is washed over them, as well as a considerable quantity escaping between and under the brushes, so that probably the quantity of sand which is washed off the plate is equal to the quantity collected. If 50% of the plate sand is worth collecting and treating in a special way, why not catch the other 50% also? To collect the plate sand is a simple operation, and can be done in the following manner:—First hang up the stamps, shutting off the water at the same time; then place a "collecting trough" under the lip and at the foot of the plate, and hose the sand down into the "collecting trough." Care and judgment should be exercised as to the quantity of water used, so that the sand will not be washed out of the "collecting trough." In dressing the plate a quantity of sand is loosened from it. This necessitates the plate being hosed down a second time before the "collecting trough" is removed. When steaming and scouring plates the trough should also be used. Sheet iron would be a good material to construct the "collecting trough" with, as the particles of amalgam which are hosed down from the plate will not stick to it. The collecting trough should run the full width of the plate, and be constructed with an overflow pipe. Fig. I. shows the trough. "A" marks the overflow pipe. The trough should rest on two

brackets, and when in position the inside wall of the trough should fit close against and under the lip of the plate, the overflow pipe protruding through a slot in the plate launder. This slot could be closed by a simple and effective shutter arrangement, so that the pulp leaving the plate would not splash through it. The overflow pipe should discharge into a bucket placed on the mercury trap. By this means any sand washed from the collecting launder would be retained in the bucket. Fig. II. shows the collecting launder in position. This arrangement would necessitate a slot being cut in the plate launder. The supporting brackets for the trough should be so arranged as to give strength to the side of the plate launder, where it was weakened by cutting the slot in it.

To remove the sand quickly and cleanly from the collecting trough, it will be necessary to have an iron cylindrical tank full of water, about 4 ft. × 1 ft. 3 in. would be sufficient. It should be suspended on a small carriage in the same manner as a truck is. It could be easily wheeled from plate to plate. When the "collecting trough" is removed from the foot of a plate, the sand would be washed easily and quickly from it by dipping it up and down in the tank two or three times. There should be two collecting troughs to prevent delay. In order that no sand shall escape at the corners of the plate and over the ends of collecting trough, two small angular blocks of wood 8 in. long running from 4 in. wide to half an inch at the point, and 1½ in. thick should be pressed tightly on the plate, and screwed to the side of the table. These blocks would be permanent fixtures.

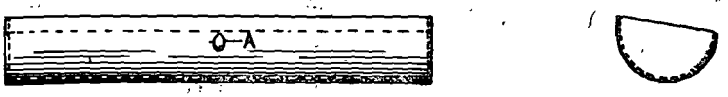
Carrying Amalgam Box, Brushes and Tools from Plate to Plate.—These should be carried on a small trolley table fitted with a sheet iron or copper plate top, and a launder at foot to catch the drainings from the table. If this trolley were supplied to the amalgamator when dressing and scraping plates, it would prevent all loss from dripping brushes, which are full of amalgam, and are frequently carried around in the hand. The trolley should be small and handy, and as high as the amalgam table.

Changing Screens, etc.—When changing screens, working at shoes and dies, etc., there is always far too much scratching and tearing up of the plate and amalgam by throwing tools, screen pins, pulling coarse gravel out of mortar box, etc., on to the plate. This should all be avoided, and can be with a little care. Before opening a mortar box a rubber apron made from insertion should be laid down first over the plate. There should be two aprons kept, one small and one large; the small apron to be used when changing screens, and the large one when working on heads, shoes, dies, etc. The apron should be laid down before the shoe and die board is put on the table. A roller should be attached to each apron, so that it could be rolled up after use.

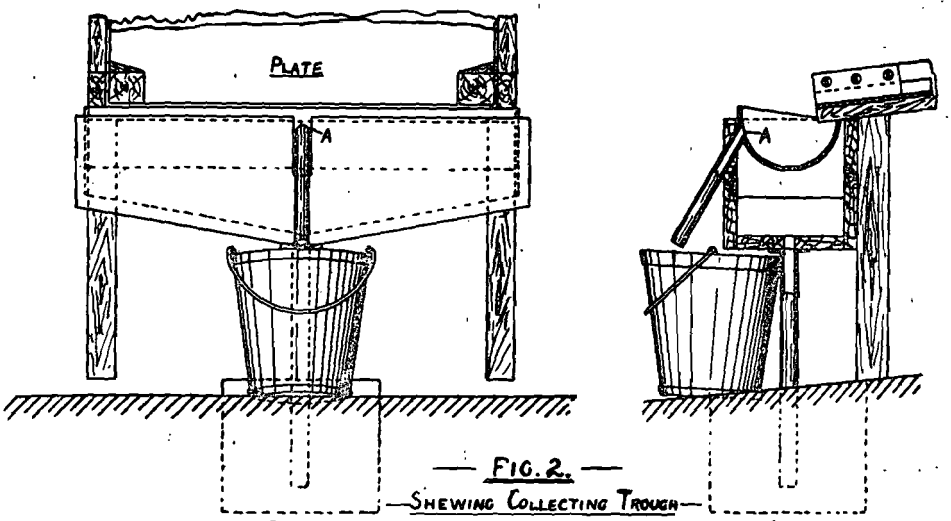
The gravel or sand which escapes from the mortar box, when the box is open, should not be permitted to come in contact with the plate. Two important points are achieved by preventing this: first, the plate and amalgam are not cut up; second, it will not be necessary to take the gravel to the amalgam barrel, as it has not come in contact with amalgam or mercury. The gravel and sand can be returned direct to the mortar box, thereby reducing by at least one half the quantity of gravel and sand, which the amalgam barrels are called on to deal with. The rubber aprons should be made to fit close up to the lips of the mortar box, and close to the sides of the table. There should be also a shallow iron tray, so made that it will fit close to and against the lip of the mortar box, and run along the full length of box. The tray should have several drainage holes in it.

To Change a Screen.—When the stamps are hung up, and water shut off, lay down the smaller of the two rubber aprons, then put the tray in position. The screen pins, splash board, screen and tools can all be laid on the apron without damaging the plate or amalgam. When hosing the gravel off the sill or lip of the mortar box, the tray should be held tightly in position, so that it will catch the gravel and coarse sand. The new screen having been adjusted, the apron should be swept, and the sweepings put on the tray. The contents of tray may be returned to mortar box at once.

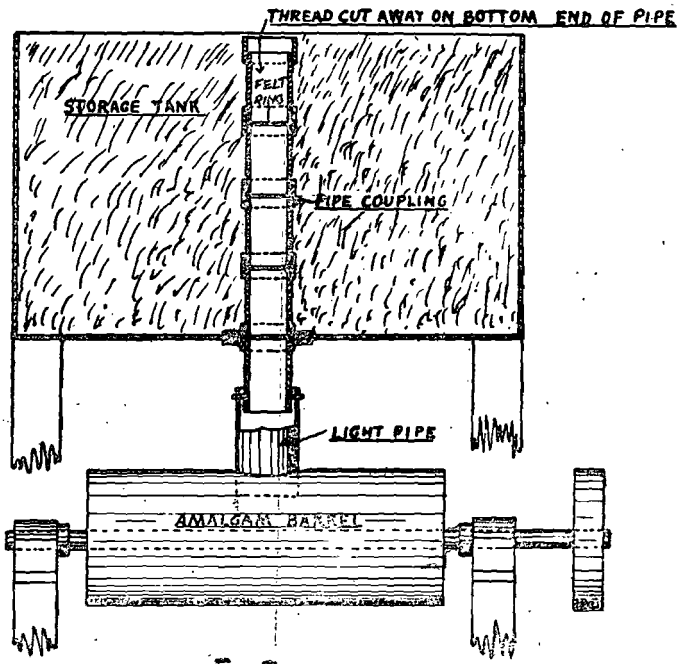
Bad Arrangement in "Wash Up" and Amalgam Rooms.—The tanks, bins, boxes, and tubs in which coarse gravel, sand washed from the amalgam, sand taken from mercury traps, and plate sand, are stored until they can be dealt with in the amalgam barrel, are in most cases badly arranged, which entails a very considerable amount of labour, as well as loss in handling. These storages should all be so placed that once the sand has been deposited in them, it could be fed automatically (or nearly so) and directly into the amalgam barrels. The amalgam barrels should be so situated that large storage tanks could be placed directly over them; into these storages the plate sand, and other by-products which are to go through the amalgam barrels should be put, also all by-products from the battery manager's wash-up table. They should be delivered direct into the storage tanks. This arrangement would obviate the necessity of carrying tons of sand in buckets from one place to the other. When this sand has passed through the amalgam barrels, it is again all carried in buckets to a battery specially set apart



— ELEVATION — — FIG 1 — — END ELEVATION —
— COLLECTING TROUGH —



— FIG. 2. —
— SHEWING COLLECTING TROUGH —
— ELEVATION — — IN POSITION — — SECTION —



— FIG. 3. —

for it. Instead of bucketting it back to the battery, it should be transferred by some mechanical arrangement, but a better arrangement would be a light battery of three or five stamps, so placed that it would receive the residue from the amalgam barrels direct. The pulp should pass from the plate of this battery into the main tailings launder going to the cyanide works. A simple and good arrangement for feeding the sand from the storage tanks into the amalgam barrels is: To fit each storage with an 8 in. pipe, passing through the centre of its bottom, say 12 in. long on the outside, and 6 in. high in the inside. The inside pipe should be built up of 12 in. lengths till the pipe is as high as the top of the storage, each length of pipe to slip down over the other a distance of 2 in., and to have a felt washer joint. From the end of the pipe should be attached a detachable pipe leading into the amalgam barrel. When it is required to fill the barrel the outside detachable pipe would be placed in position, and the sand shovelled down the pipe. The sand having been lowered a foot in the storage tank, the top length of pipe should be removed, and so on. This would make the discharging easier and quicker. Fig. III. shows storage tank.

There are often by-products passed through the amalgam barrel which are most detrimental to the extraction in the cyanide works. It is a common practice to burn old brushes which have been used in dressing plates, old screen frames, etc., and collect the ash and half charred remains, and put them through the amalgam barrels. Charcoal or ash of any kind should on no account be sent to the cyanide works, nor should old plumbago pots. This description of by-product should be handed to the cyanide manager who can make some return to the battery manager for it. It can be got rid of in the lead bullion furnace to good advantage.

Mill men should join more generally in the discussions on tube mills, and help to solve the knotty question as to their suitability and efficiency. Another point which gives very large scope for debate is the removal of the amalgam plates from immediately in front of the mortar box. Although the plates are being taken out of the charge of the mill man, yet he sits quiet without raising a single point, either in favour of the new scheme or otherwise. Personally, I should not be surprised if it were found to be a mistake to have the amalgam plates away from the mortar boxes, as the free gold will be sure to concentrate in the cement launders, and in every corner it meets on its way to the amalgam table.

Let me conclude with one word of advice to all mill men and that is, to study their extraction, and work as if there was not a cyanide works behind them.

Mr. E. H. Johnson (*Past-President*): In proposing a vote of thanks to the author for his interesting little paper on various stamp mill accessories, I think he has somewhat misunderstood the purpose of removing the plates from the front of the mortar box. Where very high stamp duties are being obtained by the use of coarse screens, the erosion of amalgam by the coarse grit passing over the plates precludes the possibility of successful amalgamation. Hence the relegation of the plates to the tube mill circuit, where the ore is sufficiently comminuted to obtain good amalgamation results.

Mr. Andrew F. Crosse (*Past-President*): I have much pleasure in seconding the vote of thanks, though I am afraid that some of our members will consider the loss of amalgam mentioned is much over-estimated.

Mr. F. W. Cindel (*Member*): I should say the best thing in this paper is the first paragraph, but as I have not studied the paper I shall reserve any remarks for a future meeting.

The Chairman: It is the first paragraph which strikes me. It is suggested that the mill manager and amalgamator are not looking after their extraction. I think they are, but we do not hear enough from them. Mill men seldom join in our discussions, whilst the cyanide men are chasing each other round and arguing the point all the time. I think if the mill men would make themselves heard a little more, they would get on a little more. It is generally the cyanide men who rise to be reduction officers, and I think that is generally due to the fact that they discuss these matters publicly, and people know what they are thinking about.

Mr. Fraser Alexander (*Member of Council*): I cannot agree that this paper is really representative of common battery practice on the Rand, and, to say the least, I think it is rather obsolete but there are a good many points dealt with in the illustrations, which may give men who are not mill men and conversant with the common practice here, some idea how to handle amalgam, black sand and mercury. I do not entirely agree that the cyanide manager or reduction officer usurps the rights of the mill men. There are a great number of very capable mill men but they are somewhat shy and diffident, and as milling often causes a slight deafness, they do not always hear interesting remarks applicable to their case, and to accuse them of not being alive to the position is hardly fair. As regards the conveying of brushes, amalgam, and tools, the author must have been on a

very large mill where so much amalgam was collected that it could not be conveniently carried, therefore necessitating a trolley. In fact he might have added a horse to pull it. I am sure mill men will be able to defend themselves quite ably in this matter.

Mr. J. H. Johnson (*Member*): With reference to the author's remark as to rushing a big tonnage through, he is evidently not speaking of batteries where tube mills are working. Now that tube mills have come in they do away with many things of which he speaks. But with batteries where no tube mills are working there is a great idea of getting through a big tonnage and trying to rush the products in the rest of the reduction plant, and the total extraction has to suffer. The mill manager is not always to blame for this as he is very often crushing according to instructions. As regards the handling of tools, etc., I have seen things happening in mills which were not altogether an advantage. Some amalgamators seem to think that once the amalgam is on the plates it requires a charge of dynamite to get it off, except when scraping.

Mr. J. F. Pyles (*Member*): The author, like many others who have dared to criticise Rand practice, seems to have stirred up a hornet's nest. While his statements are rather sweeping, I do not believe all amalgamators or mill men are above criticism. From personal experience I know there is a great difference in the results obtained by different mill men under identical conditions. Wherein does this difference lie if not in the dressing of plates and the handling of amalgam? Mill men tell of amalgam recovered from plate-floor sweepings. How does amalgam get on to the floor? Then one sees a modern plate house the floor of which would not hold shelled peas. Does amalgam only find its way on to tight floors? I was almost convinced at one time that amalgam lined tube mill launders were unavoidable. I have since found that this is not true; that there need not be any more amalgam in the tube mill circuit launders than there was formerly in launders leading to the cyanide works. It is simply a question of men.

Dr. W. A. Caldecott (*Past-President*): The policy of entrusting the whole ore treatment, both crushing and cyaniding, to a reduction works manager, was adopted by the Consolidated Gold Fields some years ago. I am glad to say that this system, now quite common on the Rand, offers more responsible and lucrative positions to metallurgical workers on mining companies than was possible with the previous separate battery and cyanide managers. It has in some measure rendered the prospective advantages of promotion and status for surface workers comparable with those obtaining underground.

I trust that Mr. Alexander will supplement his remarks upon this paper by some constructive criticism as well, and give us the benefit of his long experience by describing better devices, if possible, for the purpose of carrying out the various necessary operations brought forward for discussion by the author.

SOME EXPERIMENTS ON SMELTING TITANIFEROUS IRON ORE.

(*Read at November Meeting, 1909, and April Meeting, 1910.*)

By Prof. G. H. Stanley (Member of Council).

REPLY TO DISCUSSION.

Prof. G. H. Stanley (*Vice-President*): I was very glad to hear from Mr. Harbord that he does not share the prejudice with which the use of titaniferous ores, even in small proportions, is viewed in many quarters, and though cautious, and rightly so, with regard to the use of an all titaniferous charge, he evidently preserves an open mind.

So far as I know, no further results of blast furnace smelting of titaniferous ores since Rossi's have been published, but I am able to infer that something is being done in this direction in the U.S.A., as Dr. Waterhouse, of the Lackawanna Steel Co., in writing me with reference to my experiments, mentions that in working on titaniferous ore (no locality stated) the furnace linings give trouble through corrosion, and suggests that the cause may be the basic nature of the slag due to a lower oxide of titanium being formed according to the view I brought forward.

Mr. Crosse shows that when the local ores are worked, as I feel assured they will be at some future time, the local smelters will at any rate have a purer and more likely ore to work on than the buyers of the ores which he analysed.

Mr. Adair's contribution is of great interest, and I certainly agree with him that the physical condition of iron ores has a very important bearing on their ease of reduction. His observations show that ilmenite is much more difficultly reducible than hematite, and the second set of experiments at Pretoria also bear this out and point to the fact that the iron in ilmenite reaches the bosh unreduced, and needs the intense reducing conditions there to effect its reduction, but without enabling it, or giving it time, to take up carbon, or that the oxidising effect of the TiO_2—

itself being reduced—uses up carbon which might otherwise have been absorbed by the iron; so that his suggestion that titanium interferes chiefly by preventing carburisation appears likely to be near the truth, since if it prevents carburisation it would also tend to prevent reduction: later experiments show that the supposed infusibility of titaniferous slags does not of necessity exist.

I do not think that the silicious slag I proposed would need any special lining if the iron could be reduced out of the slag, as I suppose the lower oxide of titanium to act as base and thus satisfy the affinity of the silica; the corrosion experienced may easily be attributed to the ferrous oxide in the slag. Even if it were necessary to run a basic slag to secure fusibility (whether due to excess TiO_2 in the charge or other base) a more basic lining would be called for rather than clay with added quartz.

In reply to Mr. Greig, I admit at once that he has put his finger on a weak point with regard to the fusibility of most of the slags, *i.e.*, the presence of ferrous oxide as a probable cause; but it was largely because of this that I made the platinum crucible experiments with pure TiO_2 as a component, and showed that the slags could still be fusible and fluid with scarcely any iron oxide at all present.

The question of fuel was not dealt with more fully as being somewhat outside the scope of the paper, which was only intended to deal with the experiments carried out; but I may say that both before and since I have made a good many examinations of both Natal and Transvaal coals with a view to determine their coking possibilities, and have obtained some quite good results. I will not go so far as to say that it would be possible to produce coke here of an equal average grade to Durham coke, but coke not greatly inferior with ash not much exceeding 10% and sulphur in the neighbourhood of ·7% or ·8% can, I think, be made from a few coals. At a works in South Wales with which I was connected, the figures for ash in the chemist's daily report sometimes went over 14%, and were certainly underlined as a mark of warning when over 10%, but most of it was used in the blast furnaces of the works and the rest sold to other consumers, and at some foreign iron smelting centres the coke obtained is often inferior to this. Middlesbrough with coke containing under 8% is exceptionally favourably situated.

A limey slag is of course the best way of keeping sulphur low in the iron, but there are other ways in which it can be largely removed, notably by use of manganese in mixers or by Saniter's process, which can remove over 60%, though obviously at increased cost. White iron is not necessarily high in sulphur unless produced from materials rich in sulphur, and when I said that such iron was well adapted for open hearth steel making I referred to the fact that the low silicon content precluded the Bessemer process and necessitated the employment of the open hearth furnace. Obviously it would be a basic furnace to facilitate removal of sulphur and by using fluorspar or calcium chloride in the final slags, they can be made to carry over 55% lime and still be fluid, and thus 90% of the sulphur may be removed, and in this way pig containing ·37% sulphur is being treated in the U.S.A. Sulphur, therefore, although sufficiently objectionable, is not quite the bugbear it used to be, and by a final refining in an electric furnace the sulphur may be practically all removed, and without excessive cost if this be fed with the molten steel from the open hearth furnace.

Coal as a blast furnace fuel is used successfully in Scotland, and I merely suggested in this connection that smelting by coal might be a matter for experiment.

With regard to the silicious character of the dolomite, I looked upon that rather as an advantage, because my ideas required silica to be added as a flux, since the ore was deficient in this respect, and the experiments showed that TiO_2 and CaO would not flux. My conclusion was that the slag should contain about as much SiO_2 as Ti_2O_3, and the silicious dolomite provides silica as well as the other bases required, thus lessening the amount to be provided by other means.

In reply to Mr. Pooler.—I am afraid I cannot yet state definitely what compounds are formed, but that titanium dioxide is reduced to a lower oxide, at least partially, I consider to be proved by the change of colour and loss of weight mentioned when the slags examined were heated in a reducing atmosphere, though the exact loss of weight I have not been able to find time to determine. Further than this, the very marked lowering of the melting point brought about in the same way, indicates the occurrence of a chemical change. Ti_2O_3 seems to be most probably the lower oxide formed and, as shown by this lowering of the melting point, undoubtedly acts as a base with silica as the acid, as, I think, I pointed out; but it is quite possible and probable that in the complex slags formed, TiO_2 is also present forming titanates. The results of analyses were expressed as TiO_2 because of this uncertainty as to the degree of reduction and the curious coincidence between molecular weights noted by Mr. Pooler was encountered directly I endeavoured to devise a method for determining this with some certainty. The oxides of iron would indeed have to be removed first, and I know of no method of doing so without altering the state of oxidation of the titanium present at

the same time. Lastly, may I point out that a reducing atmosphere *is* essential, not with the object of reducing TiO_2 but to reduce *iron*!

I think Mr. Hamilton was a good deal too severe on the Government. Naturally I should have preferred a proper blast furnace to experiment with instead of a foundry cupola; but as nothing of the kind was available I was glad, though not too optimistic as to the result, to make the best of what could be had, and in the limited time available, rather than do nothing at all.

The installation of a small blast furnace on anything like a working scale, and running it for a long enough time to do really satisfactory work would obviously have cost a good many thousands of pounds and occupied more time than I could have given. As there existed also considerable doubt as to whether (even if it were shown that such ore could be smelted) there was any likelihood of iron ore of any kind being treated profitably, because of the limited market available, the wisdom of making such an outlay, from the Transvaal point of view, was questionable.

As I stated in the paper, I had other parts of the world in mind as well, where the question of titaniferous iron smelting is of more pressing importance than here. Had it been otherwise, and the local consumption of the product likely to be sufficient to keep the proposed industry occupied, I have no doubt the Government would have felt justified in so doing. However, the country is going ahead, and with increase of population and industrial activity there must come a time when South Africa will produce its own iron.

I agree that a preliminary calcination of the ore would probably be advantageous, as it is extremely dense in character, and calcination might render it more porous, and perhaps, through oxidation of FeO, less likely to flux and go into the slag, and a blast heating stove would undoubtedly have been a most useful adjunct.

In conclusion, I must express my thanks to the gentlemen who joined in the discussion, and to the members generally for the very appreciative way in which the papers were received.

THE TREATMENT OF ACCUMULATED SLIME, AND THE USE OF FILTER PRESSES FOR CLARIFYING SLIME SOLUTION AND BY-PRODUCTS.

(*Read at November Meeting, 1909.*)

By JOHN D. O'HARA (*Member*).

REPLY TO DISCUSSION.

Mr. J. D. O'Hara (*Member*): There having been practically no discussion on my paper, no reply is really necessary from me. The paper was written really for the purpose of giving the experience gained in filter press work and the treatment of accumulated slimes, and to place on record the fact that the clarifying of slimes solution by filter presses here was first carried out by me at the Nigel mine, which has since then been generally adopted on the Rand.

NOTES ON PRECIPITATION.

(*Read at July Meeting, 1910.*)

By F. D. PHILLIPS (Member).

DISCUSSION.

Mr. H. F. Lofts (*Member*): The main advantages set forth in this paper appear to be that less time is taken in "cleaning up," a saving of acetate of lead is effected, which is combined with a complete precipitation, and, more important still, there is less gold carried forward. Though there is no actual proof of precipitation, it is still possible for it to happen on a minute scale, in compartments beyond No. 4, yet sufficient on a big works to affect the figures dealing with actual and theoretical extractions. It is questionable whether the scheme of re-pumping through the extractors two or three times direct from the sumps would not enable one to work with even less than four compartments. My experience is exactly the same as the author's, namely, that all precipitation takes place in the first four compartments; I therefore work with five only, and for forms sake clean up the fifth every three months. To put it briefly, after No. 4 compartment every space filled means more labour, expense and no better efficiency, but waste of time, labour and material. It appears to be open to question whether there is a consumption of zinc where no precipitation takes place. It is well known of course that zinc is soluble in cyanide. Some scientists maintain that the zinc is first attacked and then regenerated, and to prove this, state that in practice working solutions do not carry much zinc.

Mr. A. F. Crosse (*Past-President*): I would make one suggestion. It would be interesting if the author in his experiments on precipitation with solution passing through zinc boxes, made some experiments as to the beneficial effect, or otherwise, of oxygen in his solution. It might in some way assist in the precipitation of the gold by counteracting polarization. In the first box there is oxygen, in the subsequent boxes the solution goes through there is a total absence of free oxygen. Some months ago there was a

paper read on a revolving cylinder containing zinc, showing how efficient it was in the precipitation of gold. Perhaps that was chiefly caused by the removal of the bubbles of hydrogen.

Mr. J. F. Pyles (*Member*): Because little zinc is found in the solutions I hardly think it follows that a large quantity of zinc is not dissolved. Mr. Brazier showed in his diagrams a great loss of free cyanide in the boxes, a loss, I should say, equal to the total consumption of cyanide on the plant. Now, does not the explanation of so little zinc being held in solution lie in the fact that zinc is precipitated out in the sand and free cyanide regenerated?

Mr. Fraser Alexander (*Member of Council*): It seems to be the general opinion that to have a long box say of 10 or 12 compartments entails an immense amount of labour and expense. Personally I do not find that is so. It is undoubtedly a fact that to dress 10 compartments of an extractor box with zinc where 5 is quite ample to do the work appears wasteful and foolish. Nevertheless, in taking out the 10th compartment traces of gold will be found with practically no action on the zinc taking place. The zinc is therefore serving a useful purpose if only as a filter. It is not always that the extractor boxes are run at an even flow, and cases crop up where the flow is disturbed by one means or another, and precipitated slime is liable to escape from one compartment to the other. Although it may be that precipitation has actually taken place in two or three compartments, it is no detriment whatever to have more than is absolutely necessary to precipitate gold immediately even if the zinc only acts as a filter. In last month's *Journal* Mr. Brazier published some interesting diagrams on the requisite amount of zinc. I do not find that any loss of zinc is sustained by having a little excess in the boxes.

Mr. H. A. White (*Member of Council*): There are some interesting points in connection with the excellent paper read by Mr. Phillips and the discussion by Mr. Brazier thereon, upon which I would like to make some remarks.

In the first place I would refer to the small costs of precipitation which, no doubt, correctly, omit mention of cyanide and lime consumption. It seems clear from the general experience here that the total cyanide contents of the solution leaving the extractors are (within limit of testing error) identical with those of the solution at the head of the boxes. At the same time it is also well known that the "free cyanide" is, especially in the "strong" boxes, much reduced by conversion to the double zinc cyanide which, however, is quite a good gold solvent especially in presence of alkali in excess. The cost, 0·75d., for total handling required is equivalent to a gold value of 0·015 dwt., and it is obvious that solution of this value is the maximum allowable for the sump value in good work. But the real point of interest is the economic limit of reduction, which must be below 0·015 dwt., as that figure would pay for the whole process to be gone through again. Now, Mr. Brazier has clearly proved that zinc consumption is very small when the bulk of the gold has come down, and this is particularly the case in the boxes dealing with solutions, which in the course of treatment will partly remain in the residue either of sand or slime, and whose gold value is thus lost.

It seems clear that if the whole of this gold could be retained in the boxes by mere increase of capacity the chief extra items of cost involved would be extra labour of dressing and loss of interest on zinc shaving, and extra capital for boxes. These items are small, and if it is considered that some insurance for safety against occasional large increases of flow, and other known sources of decreased precipitation is required, it seems that the economic limit is not so much in danger of being exceeded as of not being reached.

In ordinary routine assays an accuracy of 0·01 dwt. on gold solutions shows good and careful work, though this amounts to two-thirds of the cost of precipitation. In experimental work of this kind it is not correct, however, to base deductions on such results, and it is possible by special arrangements, suitable to experimental work upon which important conclusions are to be based, to get an accuracy of ·001 dwt. where large volumes of solution are assayed and the results weighed on the finest balance procurable. Of course such accuracy is only necessary in the case of the last compartments of the "weak" boxes. In such a series of experiments with "slime" boxes I have found that a "trace" with 20 assay tons of solution from the fifth compartment was successively reduced about 50% in each of three more compartments and something less than $\frac{1}{1000}$ of a dwt. could still be detected leaving the extractor, when especial precautions were used with a much greater amount of solution for assay.

It is usual, as the author has done, to quote cubic feet of zinc per ton of solution per 24 hours but as the real criterion is area of contact it is necessary to quote the thickness of the shaving used. In this connection 19·5 lb. to 23 lb. per cubic foot would apparently indicate very coarse material. I find that new zinc cut with one tooth on the "Betty" lathe is about ·0013 in. thick,

and weighs dry about 6 lb. per cub. ft., and (taken from middle of "medium" box) 12 lb. to 14 lb. wet. I would suggest that the use of such fine shavings beside giving a greater area of contact per cubic foot of box space, decreases the consumption of zinc in the acid vats and reduces the "carry over" of gold to the minimum while maintaining sufficient strength to enable it to be moved successively right up the series to the top of the "strong" boxes.

I think it would be of great interest to the members of this Society if someone would undertake the tracing out of cyanide consumption. We know it is rare to lose much in the boxes, and that the residue carries only about one-third of the total cyanide used to the dam or dump. It is possible that a careful and accurate investigation of the many other known sources of loss might reveal some gaps that could be stopped with economic advantage, and in any case the scientific interest of knowing exactly where every pound of available cyanogen goes is considerable.

With reference to Mr. Brazier's question as to advantage of heating for purposes of precipitation of gold I think the solution of the problem will be found in the strength of free cyanide maintained. I find that solutions below 20° C. require a strength of 0·010% KCy to render precipitation complete and certain for any length of time and that ·008% or even as low as ·004% "free cyanide" is sufficient with hotter solution up to 28° C. Before using heat for the slime solutions I found in the winter that it was necessary to use as high as ·017% KCy to ensure certainty in precipitation with the same flow as the hotter solutions with maximum strength of ·010% even in the coldest weather.

Dr. W. A. Caldecott (*Past-President*): As regards the influence of thickness of zinc shaving upon weight I believe that the Knights Deep shavings are between ·002 in. and ·003 in. in thickness. Whilst freshly cut filiform zinc may only weigh 6 lb. per cubic foot, yet this weight may be trebled or more for old zinc, such as the author presumably refers to, by compression and breaking up during use. I can hardly agree with Mr. White's view that the cyanide consumption in the extractor boxes is entirely due to formation of potassium zinc cyanide, inasmuch as with weak solution undissolved zinc cyanide may be found and possibly other nitrogen compounds.

Mr. F. D. Phillips (*Member*): I have to substantiate Dr. Caldecott's remark in reply to the point raised by Mr. White, as to the weight of zinc returned to the boxes. The figures referred to zinc that had already been in use—some of it for as long as ten weeks. As regards the assay values mentioned by Mr. White, I may say that we have numerous traces in our reports, but they are always entered up as 0·010 dwt.

Mr. Mather Smith (*Member*) (contributed): I was very interested in this paper, as I think it marks a further step in the right direction. It took years to discover that it was possible to reduce the number of compartments in a precipitation box; many plants are still using weak and slime solution boxes of seven and eight compartments, and I think that if the author were to try again he would have better results. From his own figures, in his third table, the solution leaving the fourth compartment of the lightly packed weak box contained only ·02 dwt. gold per ton, and the last two compartments were useless. Did his tightly packed weak boxes give lower results after the fourth compartments? If not, then the fault did not lie in the lighter packing of the first four compartments, but in the lack of fresh zinc in the fifth, and had the 660 lb. of old zinc been taken out of the 5th and 6th compartments and the fifth dressed with new zinc, he would still (allowing for a 50% extraction in the 5th compartment) have had a ·01 dwt. solution leaving the box, and a saving of the 6th compartment with its 330 lb. of zinc.

Lightly packed boxes require to have the zinc well packed into the corners of the compartments and round the frame handles, and with lighter packing the boxes require more frequent dressing but as there is less zinc to handle the dressing takes less time. With lighter packing and more frequent dressing the zinc will always be fresher in the first compartments; this is substantiated by the author's fourth table, which shows that the first compartment of the slime box gave a considerably greater percentage of extraction than the first compartments of the strong and weak, and this is evidently due to the greater freshness of the zinc in the first compartment of the slime box.

Mr. Brazier in the August number of our *Journal* ends his discussion by saying that "where any appreciable quantity of this white precipitate is formed, the author will I fancy, agree that more zinc is required in the boxes than would be the case in clean solution such as at the Knights Deep."

In my paper on precipitation (this *Journal*, March, 1909) I endeavoured to show from practical working tests, that the reverse is the case, and that the greater the quantity of zinc used beyond that actually required for good precipitation, the greater would be the quantity of white precipitate formed.

In Rose's "Metallurgy of Gold" (fifth edition) a table is given on p. 296, which shows that the zinc continues to dissolve after all the gold has been precipitated, and I feel certain that the "appreciable quantity" of white precipitate is caused by excess of zinc in solution.

The Chairman: This has been a very interesting and valuable discussion, particularly to those engaged in zinc box work. Heating the solution certainly does improve precipitation where weak solutions are concerned.

THE ASSAY OF TIN ORES.

(*Read at March Meeting, 1910.*)

By Jas. Gray, F.I.C. (Member).

REPLY TO DISCUSSION.

Mr. Jas. Gray (*Member*): I fully anticipated when bringing my paper on the above subject before the Society that the discussion would be full and instructive, and I am therefore disappointed to find that only two of our chemical members have come forward with criticisms, and to these gentlemen I have to express my thanks.

Mr. Croghan's opening remarks may be passed over without comment. The statement that perfect fusion, even of a concentrate, can be obtained in ten minutes, using NaOH, is open to serious question and notwithstanding Mr. Croghan's experience, I maintain that complete solution, except in a few isolated cases, is impossible in this time, even with the aid of the blowpipe. I admit that when reduction of the tin solution is effected with iron nails about two drops of iodine solution are necessary to produce the end point, and in ores where the tin content is low this is a decided disadvantage. Since the publication of my paper I have carried out experiments on the nickel reduction and have found that it is much superior to iron. When a mixture of 0·25 gm. of silica and ferric oxide was fused with sodium peroxide and the solution obtained reduced with nickel, it was found that half a drop of iodine solution was sufficient to produce the starch blue reaction. With ores containing arsenic it is advisable to remove it by acid treatment before fusion when the nickel reduction is used.

The first point of Mr. Whitby's criticism is with regard to what he terms my "industry in the bibliographic line," and I would suggest that he might probably have arrived at some reliable method for assaying tin ores by this time, had he been willing to wade, even superficially, into the "mud of history." It will be a source of satisfaction to me if I have been able to assist him in this matter.

I would ask my critic by what authority he takes it upon himself to speak for his professional brethren, when he says: "what we chem sts are looking for is a method which will estimate, with reasonable accuracy, the tin in an ore ranging, say, from ½% to 1%." To me, since the adoption of the process outlined in my original paper, tin ores of all kinds present no terrors. Mr. Whitby's assumptions regarding the details of my experiments are incorrect and therefore need not be considered.

To come now to what my critic terms "more serious matters;" I always have a fair excess of HCl present before titration, and indeed during the whole reduction (about 60 cc. of concentrated HCl being used for each estimation), and so far I have never found that tin and arsenic are co-precipitated, even as an alloy. I hope our metallurgical friends will note his suggestion for the preparation of an alloy by precipitation. It is unfortunate that he has failed to bring forward experimental proof regarding this statement. I regret to have to suggest that he should repeat his experiments on the masking of the starch blue colour by the presence of tungsten. I will admit that to the inexperienced eye it is difficult in some cases to note the change in colour when iron nails are used for the reduction but with nickel it is comparatively easy. I would further refer him to Thibault's "Metallurgy of Tin," where it is stated that Ibbotson and Brearley found tungsten to have no interference. A wade into "the mud of history" would have repaid Mr. Whitby in this instance.

I should like to know how Mr. Whitby has "frequently proved" that the peroxide fusion is open to the same objection as the hydrogen reduction method? Has he ever found when using sodium peroxide that some cassiterite was unattacked? I fear he must have been thinking of caustic soda when penning his contribution. From the manner with which he concludes his remarks, it would appear that only lack of time has prevented him from proving, to his own satisfaction at least, that the impurities I have mentioned as having no deterrent action, have a serious effect on the correct assay of tin ores. I sympathise with my critic on his inability to separate "the grain from the chaff," and regret that my efforts in this direction have unfortunately in his case been unsuccessful.

In conclusion, I hope that my endeavours to throw some light on the dark paths of tin assaying have met with some success notwithstanding my regrettable failure in Mr. Whitby's case.

SAND-FILLING ON THE WITWATERSRAND.

(Read at June Meeting, 1910.)

By EDGAR PAM, A.R.S.M. (Member).

DISCUSSION.

Mr. Tom Johnson (*Member of Council*): Having recently been doing a little sand filling, I would like to say a little more on the subject.

We want to ascertain what are the best sizes of timber to put in the barriers. The weight of sand saturated with water, and water standing on it will be, say, 90 lb. per cubic foot, this would amount to something like 10,800 tons on a barrier 400 ft. long, 6 ft. high, 200 ft. backs, and a dip of 30°. I know our barriers are not carrying this weight therefore the sand itself must be to some extent self-supporting, probably in the same way as grain does in hoppers, where I think it is found that, after a height of three diameters of the bin is reached, there is no increase of weight on the bottom.

A workable rule for strength of round parts or sticks is: diameter in inches

$$= \sqrt[3]{\frac{\text{load in lb.} \times \text{factor of safety} \times \text{length in feet}}{\text{factor for timber (320)} \times \text{constant for loading (2)}}}$$

This rule calls for much larger sticks than are used in many cases, and I do not think it is the fault of the rule, but the condition of the sand behind the barrier, of which we know little as yet. The cause of leakage of a barrier, which often happens after 100 ft. or 150 ft. of sand has been filled in, is the breathing of the barrier, caused by the greater part of the weight being taken off when the place is idle and the water drained off. When the water is drained off, the sand shrinks and parts from the roof, and on commencing to fill again, the water makes its way over the sand and down to the barriers, and there is a very great weight put on to the barriers rather suddenly. This breathing of the barriers cannot be seen, but it is quite reasonable to suppose it occurs. This movement of the barriers opens the joints at the roof and floor; therefore, to remedy this trouble of leaks, some coarse stuff should be mixed with the sand. It would also make the filling leach quicker, and it would reduce the cost of filling, as the barriers would not need so much particular work on them. To crush this coarse filling would not increase the cost, as the above saving would more than pay for the crushing. Personally, I think we should do away with built barriers as much as possible. For old workings, some drive where the lower rib is solid or mostly so, should be selected, either between the shafts or on the outer side of the shafts, the drive blocked off and the whole filled, blocking up the drives in rotation until the area is finished. After being allowed to drain, any of these drives could be re-opened to get out the drive pillars, or any other ore that was worth the trouble. In new work, the getting out of the ore and filling could be done as mentioned in the June *Journal*, p. 452, in this way only boxholes would need barriers.

Mr. H. A White (*Member of Council*): I should like to ask exactly how the coarser rock is going to help you?

Mr. Tom Johnson: If there is a leak in the barrier the coarser particles will go with the stream towards the leak; these coarser particles will get stopped at the leak, other particles back up against the coarser ones, and so on until slime will not pass, *i.e.* something in the manner in which we would form a filter. Once the filter was formed it would not matter how much breathing of the barrier took place, it would not leak sand, only water which does not matter. Another thing I may mention is that I do not think it is necessary to send sand down with water in our deep vertical shafts. I think it could be sent down dry, in the same condition it is received from the dumps. The question of sending sand down with water or dry, is more a chemical than a mechanical one, due to the nature of the water after use.

Speaking of dumps, on the night Mr. Pam's paper was read, Mr. W. T. Anderson mentioned that he looked forward to the time when all our dumps will have disappeared. I do not know that I should like to live so long, if we have to wait for the sand filling to cause the disappearance. I think it impossible to put the whole of the sand back, and those who expect it, expect too much. The crushed rock is in the ratio of 10 to 6 with the solid rock, and if we allow 40% slime, then the sand remaining is enough to fill up all the excavations made, and we cannot do that. The sorted waste will not affect the question much, as the excavations will be reduced by the weighting of the roof to that amount. I see by one of our technical papers, that some people are of the opinion that sand-filling is more or less a failure. I do not think so, I think it is going to turn out a great success.

Mr. G. H. Smith (*Member*): The costs on the Ferreira Deep are 1s. 8d. per ton. That is as expensive as stonewalling. Another thing is the difficulty of making the bratticing watertight. Why not put in a stone wall right away?

Mr. R. E. Sawyer (*Associate*): I should like to ask Mr. Johnson if reinforced concrete could not be used instead of timber.

Mr. Tom Johnson: That is a matter of expense; also the movement of the roof or floor would crack and crush the concrete, or any solid barrier, where timber gives way an appreciable amount before breaking. Using concrete at the roof or floor is liable to fail for the same reason. I think the best way is to take advantage of any natural barriers such as faults, dykes, etc., and use as little artificial barrier as possible.

Mr. B. C. Travers Solly (*Member*): I should like to know how Mr. Johnson would propose to carry the dry sands down the shaft?

Mr. Tom Johnson: The sand would fall like rain. It would not drop in a lump. You can stand in a shaft and allow the fine sand to come from the top and drop on you. It must be understood that the sand would be passed through a grating.

Mr. F. W. Cindel (*Member*): In regard to falling sand, I notice that in the discharge of cyanide tanks the hole in the bottom of the tank gradually becomes choked up. Now as Mr. Johnson proposes to take the sand from the dumps, there is of course a certain amount of moisture in the dump, and it seems to me that if this sand is taken directly from the dump, and not being dry enough it is liable to choke up the pipes.

Mr. Tom Johnson: There might be a difficulty at the entrance of the pipe, if using sand containing a large amount of moisture for so called dry sand, but there would be no difficulty below, as the sand would begin to lose its moisture in rushing through the air. The choking up of the tank outlet is generally caused by the bottom layers of sand in the tank, these bottom layers usually being much wetter than the average. For passing dry sands down a shaft large pipes, or a whole compartment would be used.

Mr. F. W. Cindel: I should like to ask if this method has been tried?

Mr. Tom Johnson: I do not think so.

Mr. M. H. Coombe (*Member*): The point that troubles me is as to what happens to the sand in a stope that carries a watery footwall or hanging? The whole thing would become a huge quicksand which could not possibly support the hanging and would in its turn become a danger and standing menace to the levels below when the pillar or stull crushed or decayed above the level.

Mr. Tom Johnson (*Member of Council*): That is certainly one of the dangers we have to guard against; it is recognised and will be provided for at the Ferreira Deep. The water must be led away to the outside of the barrier, and not allowed to stay in the sand. This can be done by pipes of iron, wood, clay, etc., or by laying a track of rocks covered by matting, etc., to keep out the sand, or the stope could be left unfilled if necessary.

Mr. M. H. Coombe: The whole thing hangs on our mining methods. We are losing the art and have ceased to mine. In other countries where mining methods are adopted one does not hear of the ventilation question. The stopes are filled with rock and the air currents can be controlled and taken where desired. Here, where such huge mills have to be supplied with rock, everything is done at such a rush underground that we have not time to apply common-sense methods in our mining practice, therefore everything is gutted out and left open—"If the hanging stays, all right; if it does not, let it come." That is our mining on the Rand to-day, and sand filling in our flat stopes is not going to be a cure but rather add greatly to the troubles of our underground men.

Mr. Fraser Alexander (*Member of Council*): I think the average miner in dealing with sand filling is rather getting out of his element. I do not want to disparage Mr. Johnson's remarks, for he is experienced in both practices, but we know from the experience of handling sand dumps, that it is not every man who can handle large quantities of sand economically. I think we have probably reached that stage in sand filling in the mines where the miner finds that handling sand is somewhat different to handling rock, and I believe in a combination of the two. I agree with Mr. Johnson's remarks that he requires a filter bed that will assist leaching in the mine, and a man thoroughly acquainted with the handling of sand could assist the miner in this.

Mr. Tom Johnson: The point seems to be forgotten that if you put in a pack wall, you still have to put in a wood barrier making a double expense. A pack wall newly put in, would not stand the weight of sand and water, which on the Ferreira Deep has several times amounted to as much as 1,400 tons of sand in four hours or so, plus the water (calculate the weight when the stope is about full.) The trouble is the movement of the barriers: coarser stuff to form filters is the help we need. If we had packs in that had been compressed with the weight of the roof, it would be feasible to use them.

The coarse crushing I speak of is not by the battery, but by a crusher. I think I mentioned in my last contribution to this paper, that at one colliery they passed their filling through 3½ in. apertures. At the colliery I visited in Belgium, 30% of the filling was from ⅜ in. up to 2 in., this

latter is what I should like to see going in with the sand. I believe there is no place in the world where they have tried to water-fill stuff of such average fineness as we are doing here.

These remarks to-night apply to the deep levels. Outcrop mines have a pull over the deeps, as they do not need to use so much water as the deeps do, for they can go more direct to the stopes.

The Chairman: It seldom happens that a new process for doing anything can be introduced without unexpected difficulties developing, and I hope that our mining, metallurgical and mechanical engineers will all be able to do something towards solving this problem.

Mr. Robt. Allen (*Member of Council*): In connection with the author's interesting paper, a short description of the rill system of stoping and filling, a method largely used in Western Australia, may be of interest to some members.

When working under ideal conditions, the ore body, if continuous, is divided into blocks by equidistant levels, about 200 ft. apart, and by equidistant winzes on the hanging wall side of the lode at about from 150 ft. to 200 ft. apart. The reason for having the winzes on the hanging wall side is that the stopes are thus filled with the least amount of handling. The accompanying diagram shows how a mine is blocked out in the above manner. It is assumed in it that only one winze is connected with the surface. This serves as a pass for filling, but the number of passes from the surface is purely a matter of convenience.

The filling usually consists of fresh residue, which may be of sand, or of roasted "slimed" ore or of raw "slimed" ore, and which may contain up to 25% of moisture. If the residue contains too much moisture there is a danger of it clogging the passes, so that sometimes it is necessary to stack it on the surface for a short time previous to delivery to the mine, and by this means also much of its residual cyanide content is destroyed. No chemical treatment whatever of the residue to destroy its cyanide contents is practised in Western Australia. The residue received from the surface can be distributed to the various winzes by means of a belt conveyor system along a disused level above the topmost workings.

The methods of stoping and filling on the rill system are as follows:—On any particular level a leading stope is taken out below the ore to be mined, when the drive is timbered usually, either by single stulls, or when the lode is over 14 ft. wide, by saddlebacks, at intervals of 5 ft. The latter consist of pairs of stulls sloping towards one another like the rafters of a roof, and bearing upon a longitudinal ridging of sawn timber 2 in. thick. The stulls are lagged with poles about 4 in. in diameter of a local wood called gimlet wood, or with old iron pipes. The lagging is in turn covered with old filter cloths, or the sides and linings of cyanide cases or any other inexpensive material which serves to prevent the residues from falling through.

Two alternative methods of stoping the ore are shown at A and B on the diagram. In the former all the holes are "down" holes and can be drilled wet, which is an important consideration in view of the necessity of reducing dust production. In this method the benches are taken out at an inclination slightly flatter than that of the natural slope of the filling, which in the case of residues is about 45°. This is the more commonly used method.

In the method shown at B the ore is mined by a series of horizontal cuts, and some of the holes drilled must be "uppers" and drilled dry.

Usually during the timbering of a drive ore chutes are put in at distances of about 50 ft., one (P) midway between the winzes (W), the others (Q) being intermediate. As stoping proceeds passes about 4 ft. × 4 ft. in the clear are built above these chutes, usually of 7 in. logs, but sometimes of 9 in. × 3 in. sawn timber. Each winze is also "cribbed" up except when it will not be required later on for passing "filling" to lower workings or for ventilation. In such a case the timbering of the level below the winze can be closed up and the winze filled up.

The breaking of the ore and the filling of the stopes with residues succeed one another alternately. Before the benches of ore are blasted eucalyptus saplings or slabs are laid on the sloping surface of the residue filling. These serve to keep separate to a great extent the broken ore from the residue, and assist in its "rilling" into the passes, very little labour being then required. The passes are then built up close to the working faces and covered over to prevent residues from entering them, and the poles or slabs are removed. More residue is then dumped down the winzes into the stopes, filling them up to a convenient distance from the faces. When a stope has assumed the appearance shown at C, when all ore can be rilled to the passes P, the intermediate passes Q are no longer entirely necessary. As stoping proceeds the appearance of the stopes becomes similar to that shown at D. It is usual to stope a series of blocks on the same level simultaneously, so that the filling of the stopes with residue on both sides of the winzes and the building up of the passes can be carried on symmetrically. When the stopes are nearly beaten out, as shown at E, it is usual to sink subsidiary winzes R in the triangular blocks of ore left

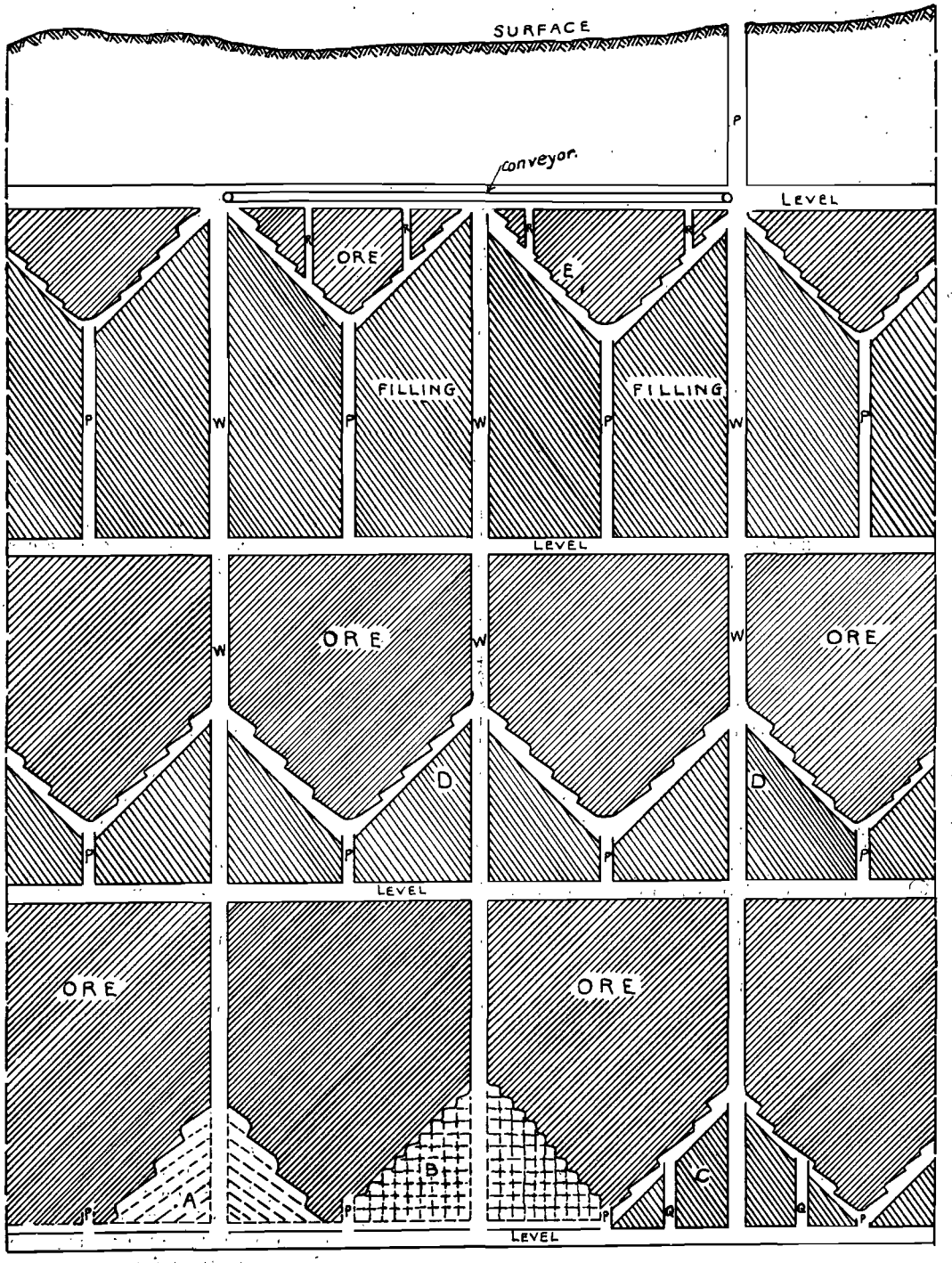


Diagram showing the method of Stoping and Filling on the Rill System.

below the level above, through which the residues for filling can be dropped. The lodes in Western Australia are usually very steep, and the rill system of stoping and filling is generally particularly applicable to them. Generally, as the dip of the lodes decreases below 45°, more and more shovelling is necessary to assist the rilling, and the method becomes inapplicable when the dip is less than about 35°.

Filling with residue in Western Australia has been in use for about 13 years, and its present cost is about 10d. per short ton of ore mined.

The accompanying photograph is of an underground belt conveyor installed on the Great Boulder Perseverance mine for distributing residue, the drive it was installed in being specially driven for it.

A Member : What is the average dip of their stopes?

Mr. R. Allen : They are very steep, quite different to what they are here.

Mr. G. H. Smith (*Member*) : What is the average dip of the face from the winzes?

Mr. R. Allen : About 45°.

THE EFFICIENCY OF LABOUR UNDERGROUND.

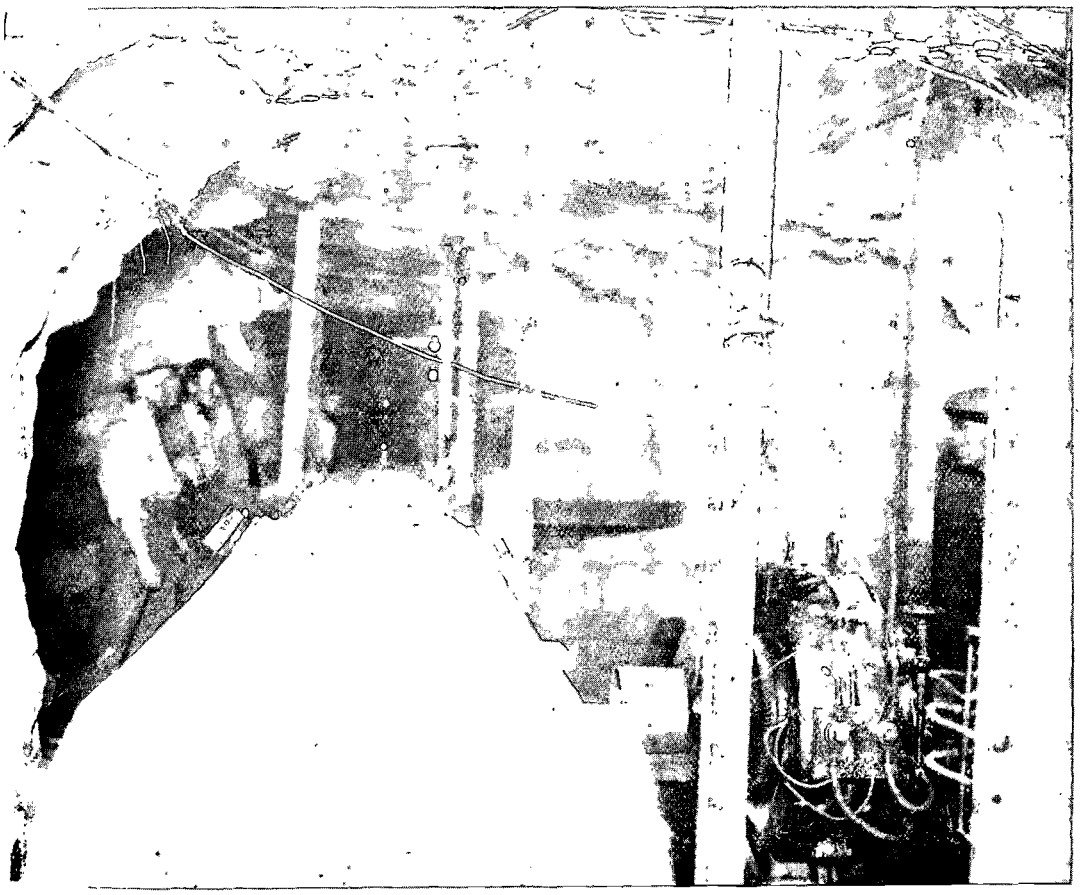
(*Read at August Meeting, 1910.*)

By Tom Johnson (Member of Council).

DISCUSSION.

Mr. W. S. V. Price (*Member*) : I was more than pleased to find that a phase of mine economics (I had almost said mine economy as practised on the Rand) was to be dealt with at our last meeting, and introduced by so able an exponent as the author.

It may be taken as an axiom that economy without efficiency is only another way of spelling extravagance, yet to the average mining-man it



Underground Conveyor on the Great Boulder Perseverance Gold Mine, Kalgoorlie, W.A., for Distributing Residue for Filling.

would appear as if most of the mine managers studied economy instead of efficiency with regard to their white workers. Possibly the reason for this is the same as that alluded to by the author with regard to a certain class of shift-bosses, viz., lack of experience in themselves. There are some wide-awake managers who do try to gauge a man's capabilities and who pay him accordingly, and, as was pointed out, he will and does get better value for every pound he pays. He gets and keeps a good, efficient and trustworthy lot of men who are proud to work under a boss who knows his own work and knows when his subordinate does a good day's work.

On the mine where I am working there is a system of calculating all the expenses connected with the stopers. Every day returns are made out showing number of boys, number of holes, inches, and the amount of gelatine used. After measuring-up day, it is ascertained exactly how many boys were required to break a fathom of ground, or a ton of rock; and the number of pounds of gelatine to break a fathom of ground; the men being paid bonuses on these figures. Making all allowance for different kinds of rock, hard or soft, etc., a very good idea of a man's efficiency can be formed. For instance, in one case it had taken a great number of boys to break a fathom of ground, but the amount of gelatine per fathom was very low. What did that show? That the man had poor boys, but being a good miner, knowing his boys could not drill long holes, he only put the burden on them that a short hole could carry and then gave it just sufficient to break it. So I think this is quite a step in the right direction.

It may not be necessary for the supreme head of a concern (mine or store) to have worked in each and every department under him, although it would doubtless be of great advantage, but I certainly think that those immediately in charge of any workmen should be men who have done the actual work themselves, so that whatever may go wrong they could tell what to do to right it. Some may think this a pretty tall order in mining, but when I was at sea I used to think that the mate of a sailing vessel who could not do anything that was required on board of his own vessel, was not fit for the berth, whether it was to make a sail or spar, caulk a deck, or put a few rivets in the hull, and the same principle applies to a boss in a mine. A few years ago I heard a shift-boss say that he "was not in the mine to teach miners their work, they were supposed to know their work before they started on the mine." That may have been all right then, but even with my limited experience I have found that a good many men need quite a lot of coaching, and if by doing that a shift-boss increases the miners' efficiency, I claim he is only doing his duty, for I take it that that is why a man is put in such a position—to get the best work done. There is one very big mistake often made by otherwise zealous shift-bosses, they seem to forget that the white miner cannot do (is not supposed to do) the work himself, but to make his boys do it. He will, as the author has pointed out, get in a temper and bully a man before his boys, which can only spoil discipline, which is already too lax. Or if a man complains of laziness or insolence on the part of the boys, he only laughs at him, whereas if a native complains that he has to *work* for his pay, he appears to side with him. The *native* is the unit, *not* the white man. Train the white man how to get the last ounce out of those under him, and the only way to do that is for the white man himself to have learned by actually doing the work, so that he shall not expect an impossibility. The same thing holds in this case as I mentioned before, for after all, the white man is not the *miner* as we mean the word at home, but ganger or boss, and I hope he always will be in this country.

The author spoke of the way men are paid a fixed or standard wage, no matter how much or how little work is done. We know that is foolish, as the good man gets discouraged and the lazy or otherwise inefficient is satisfied to do as he is doing, seeing that he is as well paid as a first-rate man. There is another way whereby good men are discouraged. The same rate will be paid for stoping or development whether the ground is hard or soft, tight or easily broken! One man may have the luck to get a big cheque for very little work, and another, perhaps better, man will slave for barely day's pay. Perhaps I am digressing from the point at issue, but a little common-sense would prevent such a piece of injustice.

When I spoke at our last meeting I said that the author's paper was practically bound up in that of Mr. Penlerick's. I mean this. Improve the conditions underground and you will increase the efficiency of all, of coloured first, for a native cannot drill as well in a gas-laden, foul atmosphere as he can if the air approaches something akin to purity; the boss too will feel more like hustling round a bit if his head is clear of smoke, etc., instead of being half "dopy." We know that mine gas is not the only thing that makes men "dopy" underground, they *have* been known to get it in other ways, even sometimes in a canteen; but what has the average miner got to keep him out of the canteen? Go to the average mine single quarters and see how snug and homelike they are! I say nothing about boarding houses, for some are very well kept, and all things considered,

such as the erratic hours of mining men, etc., they keep very fair tables. There are exceptions of course, where the boarding-house keeper is first cousin to a shark and anything is good enough for a common miner; but in most cases it is easy for a man to board elsewhere if he is not satisfied. But a man's life is something more than work, eat and sleep, and what has the miner got to keep him from the canteen, which spoils the efficiency of many of our very best men? We heard about having better rooms, efficient change-houses, etc., at our last meeting. The inefficiency of the latter makes many men keep their digging clothes in their rooms in spite of the law. I remember that Dr. Macaulay, in his paper on mine gases, a few years ago said it was up to the miners to use the means at their disposal to prevent dust and gas, etc., and I quite agree with him, although in this case many men keep their clothes in their rooms because they are not safe in the change-house, or may be owing to the stupid or defective system of heating. Their clothes are damp when they go to put them on, and that is a very good way to get a touch of the pneumonia we heard so much about last meeting. I have known change-houses where the steam pipes were up in the air over the tops of the lockers. You would think they wanted to warm and ventilate the roof of the change-house. The engineer that invented and installed such a system evidently knew very little of the action of heat and evidently thought it struck down. In other ways I admit the miner does neglect the means at his disposal to maintain his health, and thereby increase his efficiency. I am sure that I have inhaled far more dust, gas, smoke and foul air generally in six months' shift-bossing than I did in six years' mining. I need to keep telling men to use the water hoses, etc., at their working places. How very rarely we find men keeping the face of the rock wet when collaring a hole, and if spoken to about the dust from it, they will tell you it is to be a wet hole when it is in far enough. I have repeatedly told men, and insisted on it, that water be applied to the face as soon as the drill strikes the rock, *whether it is for a wet hole or a dry one.* It is necessary from the miner's point of view to prevent dust, and it is equally important from the owner's point of view, for it helps to keep the temper of the drill and so increases the efficiency of the drill, and thereby the miner. The drills need all the nursing we can give them to keep their temper and incidentally the cutting edge, for goodness knows the tempering of the drills is seldom troubled about in the shops, they are not paid to do that, and so long as they look pretty and the number is turned out, all is well. In this way the efficiency of all connected is seriously impaired. To my mind, the whole system is entirely wrong. Instead of the drill-sharpener being paid so much per score or hundred, he should be paid so much per machine; then far more attention would be paid to the tempering and steel would be tempered to cut the particular kind of rock for which it is to be used. The steel bill per month would be decreased by quite 50%, for drills that now get stamped up with a few taps of the machine would do duty for two or three holes, and even then would not come out of the hole like so many we see—either flat or else the corners all broken off. The amount of steel wasted each month must be enormous, to say nothing of the smith's time in cutting off damaged ends, etc., and the miners' time changing drills in the machines. The same thing applies to hand-drills, a white man does the mechanical part in the drill-sharpening machine and a nigger looks after the tempering, if any. How does this affect the dust question? A well-tempered drill, drilling a dry hole *cuts* the rock and the chips fall to the ground, but the badly-tempered drill has to pound and pulverise the rock and then the fine particles float about in the air and get inhaled by the miner and his boys. I like the author's idea of the shift-boss, or what would be his equivalent, getting paid a bonus for the efficiency of those under him, and would like to see it put in force now, but—and it is a very big *but*—only one man would have to have the handling of one squad of men, they would always have to be on the same man's shift, and no one else must be allowed in their working place, or we should find exactly what is happening all along the Rand where men are working double shifts, *i.e.*, the loafer would get the same as the hustler's share of the cheque, while the hustler's average was reduced to the ordinary dead level. One man one job is a pretty good principle anywhere, but especially in mining, where we so very rarely find two men who do the same work in exactly the same way. There are so many ways whereby the general efficiency of the mine workers could be improved that it would perhaps be wearisome to continue, but one thing stands out very strongly in my mind as a general efficiency destroyer, and that is that curse of the Rand, Keep the mill going! It does not matter (at least it looks that way) what goes through the mill, but the directors must not on any account hear that the mill has been stopped. Seven days a week it has to be fed with something. Would it not be better to run it only five days a week and put only very finely sorted ore through, leaving out everything except pay-dirt? Of course we cannot sort fines, but if closer attention was given to mining only the pay-ore, more resueing done, greater care in developing so that while the mine

is being opened up the thought of handling the broken rock from the stopes will be kept in mind, efficiency would be improved, dividends would rise, and the wages need not come down at all, but if anything increase to those who put their hearts and their backbone into their work.

Mr. E. J. Wiseman (*Associate*): It is with some hesitation that I, as a young man, venture to add any remarks to this excellent paper; but every practical miner who gives this subject a little serious thought will agree with the author that there still remains ample scope for improving our efficiency underground.

In one part of his paper he deals with methods whereby better results would be obtained in the breaking of ground, and I will endeavour, if possible, to emphasise his remarks in this direction by bringing forward some ideas I have heard expressed on the matter by old experienced miners, in the way of comparisons between hand and machine stoping. One cannot but be very forcibly impressed by the great difference existing between the efficiency we obtain from hand and machine stoping. A good hand stoper on contract would not expect a good cheque if he did not break five fathoms of ground per case of explosives used, while his fellow workman, the machine stoper, is well satisfied if he manages to break only three fathoms per case. Both men may be equally energetic and capable miners, and have similar working conditions as regards stoping width, dip of reef, and breaking qualities of ground. Naturally one asks why this disparity should exist, and why we have this expenditure of energy in one case, without a compensating efficiency. Is it due to the two different ways the men stope, and does the system of handstoping possess any advantage over the methods prevailing in machine stoping? The machine stoper drills his four, five and six holes per bench, as the case may be, each shift. It is not so long ago since he used to fire *all* these holes at one blast, with the result that long sockets were often left; the whole bench would sometimes be hung up, or one hole would cut out another, and so leave a misfire. As this did not pay him he resorted to the scheme of blasting only the front holes on one shift, leaving the back holes for the next, and found he was then better able to judge the burden on the holes and the formation of the ground, and so obtained better results for the amount of explosives he used. This is the system of blasting now in vogue. But still the hand stoper can show greater fathomage per case of explosive used, and why is it so? If the machine stoper pondered the matter over he, would probably conclude that the hand stoper's method must be better than his own, and set himself to find out where it was so.

The hand stoper drills only one hole a shift on each bench, and blasts it separately. Sometimes he does drill two holes on the bench, not more, and even then, after blasting them together, he will likely find that he has not obtained as good results, as if one hole at a time had been drilled and blasted on that bench. Would it therefore not pay the machine man to take a leaf out of the hand stoper's book, and instead of trying to drill as many as six holes to a bench, put in *only two*, and blast *one* each shift? By giving him extra bars, he could still drill the same number of holes per machine each shift, and I think the results would amply repay him for the extra trouble he is put to, in rigging up his machines twice, or even three times, a shift. He would gain further efficiency if he could have his working face to himself, single shift.

Another comparison can be drawn in regard to the sockets of holes left after each blast. In hand stoping we have the holes of almost uniform gauge right through, so that the explosive can be evenly packed, right from the bottom of the hole, thus allowing a well-placed hole to break clean out. On the other hand, in proportion to the depth and burden of machine holes, what a large number of sockets, 12 in. to 18 in. deep, are left standing, especially if the ground does not happen to be exceptionally good breaking. These sockets represent the depth drilled by the long chisel bits, so here again we see a waste of energy. Even if the chisels are well sharpened and tempered, they lose their proper gauge quickly, and the bottoms of the holes drilled by them are then too small to receive the sticks of machine gelatine. If the stoper fails to notice this, he gets a cushion of air left between his charge and the bottom of the hole, and a deep socket is left standing after blasting. On the other hand, should he procure hand gelatine or else split his machine gelatine, so as to get it right into the bottom of the hole, it simply means that the portion of the hole drilled by the chisel is of such narrow gauge as really to represent only a hand hole, but it is given the burden of a machine hole. So he gets another deep socket left after blasting it. Under these circumstances, would it not be much better to dispense altogether with the long chisel bit, and use in its place a long star bit? I believe that the holes would then break better, and deep sockets would then be conspicuous by their absence.

I put forward these ideas for what they are worth, and am only sorry I cannot express them more concisely. I believe that, given an impartial trial, they would lead to improved efficiency. If, on the other hand, a test should prove them impracticable, then we should have the satisfaction of knowing that the present method of drill-

ing over our benches must be as good as we can have, as far as machine stoping is concerned.

The meeting then closed.

Visit to the City Deep, Limited.

Some two hundred members of the Chemical, Metallurgical and Mining Society paid a visit to the City Deep on Saturday afternoon, the 10th September, to inspect the new surface plant, which is now in course of construction. The visitors were conducted over the works by various officials of the company, Mr. Whitford (Mine Manager), Mr. A. M. Robeson (Consulting Mechanical Engineer), Mr. E. J. Laschinger, Mr. Simons (Mine Secretary) and others, and visited the mill, the tube mills, the gold recovery house, sand plant, slime plant, and power arrangements. A few of the more venturesome climbed the big cantilever boom, which reaches to a height of 200 ft. and which will be used for forming the dump.

In welcoming the members of the Society on behalf of the directors of the City Deep, Mr. R. W. Schumacher said he thought all would agree that these visits of engineers and scientists to the various mines were a very excellent thing for the industry. He would not dwell at length on the special points of the new plant, but he thought his hearers would agree that those responsible for its erection might well be proud of their work. He referred in the first instance to Mr Robeson, who was back at work again, after a short but serious illness, also to Mr. Laschinger, Mr. Bowen, Mr. Henderson, and Mr. Ross, all of whom shared the responsibility of the construction work. He thought the City Deep had the finest mill in the world. They would have noticed the steel work and the use made of reinforced concrete, and Mr. Laschinger's cam shaft support which was an entirely new feature. However proud they might feel of the work that had been accomplished, he thought if they asked Mr. Robeson that gentleman would tell them he was confident of being able to do even better work next time. Mr. Robeson had several new ideas up his sleeve and he (the speaker) would not give him away, but at their next plant, the Modder B, there would be a number of novel features of great importance. One of these would be the installation of the Butters vacuum filter process. This had been tried successfully in the States and in the mines of Mexico, and they had the pleasure of welcoming the inventor (Mr. Paterson) among them that afternoon.

It was a process for separating the gold solutions from the slimes, and, speaking as a layman, he had no hesitation in saying that it struck him as being one of the most important innovations introduced since the tube mills. It was installed at a Barberton mine (the French Bobs) and at the Crown Mines almost simultaneously. They were sometimes accused of being too conservative, but he did not think the reproach was deserved, for during the last few years the industry had made very great strides. A few years ago, for example, they knew very little about tube mills, but these were now the most important feature of a reduction plant. The plant of the City Deep was laid out with the original intention of having as the first unit 200 stamps and nine tube mills. He thought, with the experience they had, it might be asserted with fair confidence that they would change their programme to a very great extent, in enlarging existing plants. He did not think it likely that the City Deep would ever add another stamp. They would add tube mills, but no more stamps, and he thought the same policy would be applied in very many other mines. Apart from the surface construction, in which he was glad they took such an interest, there was another consideration, and one he thought not unimportant, that was the mine. Mr. Whitford and Mr. H. Stuart Martin, gentlemen in whom they had the fullest confidence, said that the City Deep was going to be one of the great mines of the world. He wished all success to the Chemical, Metallurgical and Mining Society, and coupled with the toast the name of its president, Dr. Moir.

Dr. Moir, in the course of a brief reply, said the works of the City Deep had apparently been built with a view to an existence of several thousand years. He feared that archaeologists of the future would be sorely puzzled by the ecclesiastical appearance of the concrete buildings, and be led astray in their speculations by taking the vats for baptismal fonts. He wished all success to the City Deep.

Mr. Whitford and Mr. Robeson also responded.

A Short Description of the Ore-Reduction Plant.

General.—This plant is designed to handle and treat 65,000 tons of ore per month, and in general arrangement and details of construction presents some novel features which members of the Society will doubtless recognise when looking over the plant.

Sorting and Crushing Station.—The ore from the mine at the western shaft is dumped by 5 ton skips over grizzlies into fine and coarse ore bins built up against the steel headgear. Thence the fines are taken by a 20 in. belt direct to the main

ore bin. The coarse ore is taken by *four* 36 in. inclined sorting belts of an average length of 150 ft. to the crushers, which are set on the top of the main ore bin. Each belt feeds *three* 12 in. × 24 in. jaw crushers, one of which is used as a spare. The return portion of each sorting belt acts as a conveyor of the sorted-out waste and delivers the waste to a 20 in. waste belt (common to the four belts) delivering to the dump. The main ore bin is 113 ft. × 23 ft. × 16 ft. deep in the middle, capacity 1,000 tons, and is massively constructed in reinforced concrete so as to give ample strength and weight to minimize vibrations due to the crushers. The bin has hopper bottoms fitted with heavy doors operated by compressed air. Through these doors the crushed ore and fines are discharged into the trucks for transport to the mill. The eastern shaft is to be equipped with crusher station when the scale of operations is increased.

Ore Transport to Mill.—The railway line to the mill is 6,500 ft. long and of 3 ft. 6 in. gauge, formed mostly in embankment. Maximum grade 2·65%. A train will consist of *four* or more 40-ton all steel Klussman trucks, with with automatic bottom—side discharge doors. The electric locomotive consists of similar 25-ton 6-wheeled halves coupled as a unit and fitted with vacuum brake gear for the control of the whole train. Each half loco. has a 150 H.P. induction motor operating the driving wheels through gearing and side rods. The two overhead wires will be from 16 ft. to 17 ft. above the rails. The novelty of the system as far as South Africa is concerned is that heavy electric locomotives will operate on 2,000 volt 50-cycle three-phase current as furnished by the power company. As one leg of the line must be earthed, the current for the railway is taken through buffer transformers. The loco. will not run under the crusher station ore bins, thus obviating any danger of contact with live wires round the station. The ore transport line will also connect with the mill stores, general store yard and workshops, S.A.R. siding, and shafts.

Stamp Mill.—The mill consists of 200 stamps, arranged in units of *ten*, each unit being driven by a 50 H.P. motor. Weight of stamps when new, 2,000 lb. The stamps have long heads and short stems. The mortars are short with special housing round the heads. A layer of half-inch felt is placed between the mortar bases and the concrete foundations. There are no king-posts, as the concrete foundations are carried up 14 in. wide, with indented steel bar reinforcing to above the level of the mortar-box tops. On the tops of the foundations is bolted a heavy cast-steel frame which carries the cam shaft and stem guides. Each cam-shaft rests on 11 bearings, as beside the 3 main bearings there are bearings intermediate between the cams. This is to minimize, if not entirely obviate, cam-shaft breakages by reducing vibrations. Cam-shafts are riffled and hollow. Stems are 4 in. × 13 in. long running in cast-iron guide blocks bolted to the steel guide girts, but with a wood cushion between.

As these stamps are designed for very heavy duty, each set of five stamps is provided with two challenge feeders to feed behind the second and fourth stamps of each mortar-box.

The whole skeleton of the mill-bin structure and battery house and roof is of steel, wood being only used for bin-lining, floors and girts for fastening on the corrugated iron. The ground floor of the mill is graded and will be concreted throughout. This makes for cleanliness, and the whole building is well lighted and space allowed for easy access to facilitate inspection and proper attention. All launders have 9% grade and are formed in concrete. Lines of crawls are provided for moving all heavy parts in the battery.

Pulp Elevating Plant.—As the site of the plant is very flat ground, it was found advisable to give two elevations to the pulp, the first between the battery and tube-mills and the second between the shaking tables and sand collectors. In each stage the elevation is done by one 12 in. Robeson-Davidson sand pump, with another pump as stand-by.

Tube-Mills.—There are *nine* 5 ft. 6 in. × 22 ft. Eckstein standard tube-mills. These are each driven by 100 H.P. slow- speed motor through a Citroen gear reduction. The driving gear is located at the outlet-end of the mills, thus avoiding the crowding together of classifying cone, pebble feed and driving arrangements at the inlet-end. It is at the head of the tube-mill where attention is most required and where things mostly go wrong, and the placing of the driving gears also at this end is both inconvenient and dangerous to the workers.

The tube-mill pebble-bin is built into the mill bin framing at one end, and a monorail arrangement provides for the transport of feed ore to the hoppers of 1 ton capacity at the head of each tube-mill.

Gold Recovery House.—Immediately behind the tube-mills is located the house where all the gold is recovered and handled. In this house are placed, under one roof, the amalgamating shaking tables, extractor boxes, clean-up machinery, strong room and refinery; also conveniently and incidentally the sand solution pumps, second pulp elevating plant, zinc lathes, and office of the ore-reduction officer.

There are *seven* shaking tables, 5 ft. × 12 ft., installed for each of the 9 tube-mills, and room left for the addition of an eighth table to each

one, and the whole of the mill pulp runs over these tables as no amalgamation is done in the battery. Easy and clear access to each table for the dressing and scraping of plates has been provided for, and the whole floor of the gold-recovery house has been concreted on solid ground.

There are *eighteen* extractor boxes, 5 ft. × 30 ft., each having 10 compartments for zinc shavings, but each box is partitioned off in the middle, thus virtually making two boxes set end to end. The bottoms of the boxes have a side slope one way only, and a cock is fitted to each compartment on its deeper side for discharging into a lock-up gold slimes launder running the full length of each box.

The main part of the gold-recovery house is built in wood and iron, but the refinery, which is partitioned off by heavy wire netting, is constructed of concrete and steel, the only woodwork being the purlins of the roof. Special attention has been paid to the ventilation of the refinery. An enclosed slag-yard has been provided outside the refinery door.

Solution Sumps.—The solution sumps behind the gold-recovery house are in excavation, lined out with reinforced concrete, and the bottoms have also bitumen sheeting embedded in the concrete. The sump dimensions are: *Three* sand solution sumps, each 48 ft. × 47 ft., and *one* slime solution sump, 122 ft. × 47 ft., all sumps 12 ft. 9 in. deep. The ample capacity thus allowed is to meet future requirements as regards increase in the scale of operations without adding to the sump capacity.

Sand Plant.—The collecting plant consists of one row of *six* vats, 50 ft. dia. × 10 ft. deep, built together with their supports and the trestling for the Blaisdell excavators, in reinforced concrete. A 24 in. conveyor belt running under the centre line of these vats takes the sand, elevates it, and delivers it to another belt, running at right angles, to the top of the sand-leaching vats. This leaching plant comprises *two* rows of *six* vats each. Vats are 56 ft. dia. × 12 ft. deep, the sands being filled in by a Blaisdell distributor. The vats are of steel, set on a floor and supports of reinforced concrete. The trestles and runway for the sand distributor and sand excavator are also of reinforced concrete tied into the vat supports. The solutions leached from the vats all go to a small solution classifying house where the classification of solutions going to the extractor boxes is made. The steady head tanks are installed in this house.

Sand Residue Disposal Plant.—The sand is discharged from the leaching vats by means of a Blaisdell excavator, the central discharge chutes delivering to a 24 in. conveyor belt, one under each row of vats. A short cross belt at the end brings the delivery to a common point; thence this sand is conveyed and elevated to the tailings dump by a 24 in. inclined belt carried on a steel cantilever frame instead of trestling. The delivery of this belt will, when the plant begins operations, be over 100 ft. above the ground, and the cantilever support will not be buried in the sand.

Slime Plant.—This consists of *four* conical-bottom collectors, 60 ft. dia. × 10 ft. and 16 ft. deep, built on the ground of reinforced concrete. These have inside peripheral overflow launders of steel. There are *two* steel conical-bottom air-agitation vats, 32 ft. dia. × 30 ft. and 38 ft. deep, for the aeration and agitation of charges, and *eight* steel conical-bottom 1st and 2nd wash vats, 70 ft. dia. × 16 ft. 6 in. and 23 ft. 6 in. deep. The pumping plant consists of *two* 12 in. Robeson-Davidson sludge pumps, *two* 9 in. Roturbo solution pumps, and *one* 6 in. highlift Roturbo sluicing pump. The air for slime agitation is supplied by a small Bellis & Morcom compressor belt driven. An emergency service is also provided by a pipe laid from the mine air service.

Mill and Return Water Service.—The mill service tank is 50 ft. dia. × 12 ft. deep, of steel, and supported on steel trestling. The two return water tanks are of steel set in excavations, and are 50 ft. dia. × 15 ft. deep. The pumps are *two* 11 in. Roturbos. The water-piping in the mill is formed as *two* ring mains, and is made up of exact duplicate lengths to facilitate any repairs or cleaning out.

Power Arrangements, etc.—All power for the operation of this plant is purchased electric power. All machinery is therefore driven by electric motors. In all cases except the tube-mills, the motors drive through belts. The power is distributed from the transformer station and switch house by underground cables.

It will be noted that room is left in all parts of the plant for extensions, and so that these extensions can be conveniently carried out.

Visit to the Simmer and Jack Proprietary Mines, Ltd.

On Saturday afternoon, the 24th September, about a hundred members of the Society spent a couple of instructive hours in an inspection of the reduction works at the Simmer and Jack Gold

Mine. They were received by Mr. O. P. Powell (Acting Manager), Mr. Wilson (Resident Engineer), Mr. Greathead (Mine Captain), Mr. Lea (Cyanide Foreman), Mr. Dixon (Mill Foreman), Mr. McArthur Johnston (of the Goldfields Laboratory) and Dr. Caldecott (Consulting Metallurgist to the Consolidated Goldfields). The visitors went about the property in groups, each of which was attended by one or other of the gentlemen named, and all were able to gather a comprehensive idea of the reduction part of the surface works. There was no time to go underground. One of the most interesting features inspected was the contrivance by which the disused workings below are being filled with sand in which all traces of cyanide have been destroyed.

After the inspection, the visitors assembled on the stoep of the recreation hall where refreshments were served. Amongst the members of the Society present were: Prof. G. H. Stanley (Vice-President), Messrs. F. F. Alexander, E. H. Croghan, Tom Johnson, J. E. Thomas, A. Whitby, H. A. White, A. F. Crosse, J. S. Cellier and Fred. Rowland (Secretary).

In a brief speech, Prof. Stanley expressed the thanks of the Society to Mr. Powell, Dr. Caldecott, Mr. Wilson, and Mr. Smart (Reduction Works Manager), the last-named of whom he was sorry to say was ill, and whom he hoped would soon be about quite well again. They had had an instructive and enjoyable afternoon. One of the things that had forcibly occurred to him (Prof. Stanley) was the free and open way in which information was given of the work done on the different plants. There were no secret processes at all, and there was a spirit of progress which gave a denial to the charge that on the Rand they did not take advantage of new practices. They had been thoroughly abused for not taking the lead in new ideas, of which, of course, they could not have a monopoly. In his opinion a large part of the progress since the early days was due to Dr. Caldecott and his able lieutenants, some of the innovations in the plant which marked a distinct advance in the metallurgy of gold-mining being directly due to their initiative.

Mr. Owen Powell (Acting Manager) heartily agreed with Prof. Stanley's remarks. When he (Mr. Powell) came to the mine the plant was already under construction, and his own work had only been to carry out what his predecessor had arranged. At all reasonable times, the Simmer and Jack, above or below, was open to members of the Society.

Dr. Caldecott said that the satisfactory working of the plant was largely due to those who were associated with him, especially Messrs. Smart, Wilson, and McArthur Johnston. He was pleased to recognise the value of pure chemical research in regard to the problems of the gold industry of the Rand. In regard to sand-filling the cyanicides they had found satisfactory and employed in daily practice on the Simmer and Jack were those discovered after long research by their friend, Mr. H. A. White.

Mr. Powell thanked Mr. Wilson and others who had assisted to show the visitors about the property, and Mr. Crosse maintained that by means of the successful chemical research employed on the Rand they were able to treat, at a profit, thousands of tons of low-grade ore.

Mr. H. A. White added a few words of thanks to the officials and directors, the latter of whom he likewise congratulated upon their enterprise and confidence in their technical advisers. After this a photograph was taken of the gathering, and the visitors returned to town at dusk.

Technical Details.—The following particulars of portions of the reduction works will be of interest:—

Stamp Mill.—Twenty-four batteries are equipped with short wooden observation tables only, without amalgamated plates, the pulp being elevated to classifiers for amalgamation of the metallic gold on the plates in the tube-mill circuit.

Tube-mill Classification.—One diaphragm cone classifier (6 ft. x 9 ft.) is employed for each tube-mill, as described on p. 282 of this *Journal*, February, 1910.

The classifier underflow is about 400 tons of sand per 24 hours as a pulp containing about 27% moisture.

Sand-filling.—The method employed for current sand residues was fully described in this *Journal*, August, 1910. The 60% of clean sand yielded by the crushed ore occupies about the same space in the mine as the ore before it was mined a fortnight earlier. This sand drains readily underground, yielding a clear effluent free from cyanide. The 4 in. centrifugal pump in use is capable of pumping 32 tons of sand per hour as a pulp containing 60% of moisture.

Continuous Sand-Collecting Plant.—Three sand filter tables, 25 ft. in diameter, with 3 ft. filtering breadth, are installed, of which one is a spare; the two others readily handle the total production of 1,300 tons of sand daily. The sand from the filter tables is transferred with cyanide solution as a pulp to the cyanide treatment vats. No washing cones are employed in

addition to the four 8 ft. × 10 ft. diaphragm cones above each table. Six similar return sand cones following the table cones are in use. By means of this installation the renewal of a large number of old wooden vats has been avoided and better extraction obtained, apart from securing other advantages. The method was fully described in this *Journal*, August, 1909.

Battery Sand Clean-Up Plant.—This constitutes a miniature reduction plant for the rich mill by-products, and includes a 3 ft. 6 in. × 6 ft. tube-mill, followed by a small shaking amalgamated plate. The very fine tailing pulp, after this crushing and amalgamation, is pumped to 2 air-lift cyanide vats, 10 ft. in diameter by 18 ft. deep, one of which is used for collecting and the other for treatment. These vats have 60° bottom grades, and are fitted with Adam's cut-off gates. A considerable amount of mercury and amalgam is recovered in these vats or dissolved in the cyanide solution, and subsequently on the zinc shavings, from which it is obtained later by condensation during calcination of the zinc gold slime. The cyanide treatment in the air-lift vats yield about 98% extraction and comprises repeated agitation by air and the circulating pump, with alternate settlement and decantation of the gold-bearing solution.

Obituary.

The deaths of the following members are recorded with much regret :—

Mr. John Radford Streeter, the mine captain of the Village Main Reef G. M. Co., Ltd., died on the 9th of September of miners' phthisis.

Mr. Streeter was elected a member of the Society in January, 1909.

Mr. Alexander Holmes Hartley, the assayer of the New Goch Gold Mines, Ltd., died in Durban on the 27th. September, after an operation for some internal trouble. The deceased was one of the older members of the Society, having been elected a member in June, 1896. He was a contributor to the discussions on several metallurgical papers read before the Society, and in May, 1899, submitted an original paper, "Notes on the Precipitation of Gold from Cyanide Solutions." Mr. Hartley, who was 51 years of age, was the son of the late William Hartley, J.P., of Annadale, Co. Down, Ireland, and leaves a widow and two young sons to whom the Society tenders its sincere sympathies.

Notices and Abstracts of Articles and Papers.

METALLURGY.

Reduction Works of the Randfontein Central.—"*Metallurgical Treatment.*—One departure has been made in the existing methods of recovery by plate amalgamation. It was felt by your technical advisers that the general system of plate amalgamation in coarse crushing and tube milling plants was susceptible of improvements, and they therefore made extensive investigations, beginning in Feb., 1909, at the North Randfontein Gold Mining Co., with a view to carrying this into effect. The experiments conclusively demonstrated that no provision for amalgamation should be made until after the pulp has passed the classifiers, and that owing to the fineness of the sand in the overflow from the classifiers the area of the copper required is greatly lessened, while in the tube mill circuit all that is necessary is sufficient plate area to prevent enrichment of this circuit. This provides a method of treatment that is altogether superior to existing practice, and it has had its commercial value proved by months of daily usage in all the four mills on the Randfontein South G. M. Co. Incidentally, it is also found that there is no necessity to give a shaking motion to the tube mi l plates. The net result is, that while the extraction is slightly improved, the capital cost is greatly reduced, the aggregate crushing time of the mill is increased, there being no stoppages required for dressing plates, the whole of the amalgamation is done in one building, and the maintenance and power costs are reduced through the disappearance of the machinery required for shaking the tables, and it becomes possible for the plant as a whole to be kept in a far better state of repair, owing to the gold recovery and the maintenance of the plant being each attended to by differently trained men. In view of the importance of this subject, many metallurgists and engineers have visited the mills of the Randfontein South, with a view to studying the method and results. The new system has now been running satisfactorily at each of the four mills for several months past and is now freely adopted on other groups. The railway system converges into a terminus at the reduction works, all rock from the shafts being dumped at this point into a large receiving bin, from which it is taken by conveyor belts into the classifying, sorting and crushing station, whe e there are six lines of sorting belts, delivering into the six jaw crushers, which are the largest that have yet been used on the Witwatersrand. Each line acts independently of the other, and under normal conditions, the whole of the plant here, as elsewhere, will be kept at work, but the capacity has been so arranged that in the event of any section having to be shut down for repairs the remainder can deal temporarily with the resulting overload. The waste rock is ta' en by a mechanical haulage to a separate dump to the east, while the product for the mill will be taken to its destination by conveyor belts.

The Mill and Tubes.—The mill itself consists of 600 stamps, each weighing 1,650 lb., arranged in batteries of 10 stamps in two lines, the mortar boxes being set on massive concrete blocks. The main building is a composite structure of steel and timber, covering an acre of ground, the length of the building being 630 ft., but the mill bins are entirely of timber framing, lined with steel, and these will hold 34 hours' supply of ore. In front of the mill, and in the centre of its length, is the tailings pump-house, where

there will be six pumps for the coarse sand and four pumps for handling the fine sand, all of the pumps being of the centrifugal type. In front of this again is the tube mill building, which is a steel structure, also covering an acre of ground; in it are 16 tube mills placed in two parallel rows of eight, and arranged between the tubes are all the amalgamating plates. The reduction officer's headquarters are in this building, enabling him to supervise simultaneously all the work going on in the plate and tube mill house, at the sand and slime plant, at the residue dump, while the pump-houses, extractor-house, mill and crusher station are easily accessible, being but a short distance from his office.

Sand and Slime Plants.—The sand plant is arranged in two rows, each row being almost at right angles to the battery, and this plant consists of 35 steel tanks, 60 ft. in diameter, of which 12 tanks are for collecting the sands, and these are arranged at a higher level than the treatment tanks; the system of transferring the sand will be by means of trucks, and the residues will be taken to the tailings dump by mechanical haulage. As the sand must be all got rid of in the hours of daylight, the quantity reaching the large total of 400 tons per hour, two dumping faces will be provided, which will gradually converge into one as the dump grows larger. Between the two lines of sand tanks is arranged the slime plant, consisting of 23 tanks each 70 ft in diameter, of which six are collectors, and situated in the middle of the slime plant is the slime pump-house, where the pumping operations connected with circulation, transferring and emptying the charges is concentrated. Immediately below the slime plant are the extractor-house and solution sumps; the extractor-house, besides being equipped with the most modern plant, also contains the furnaces for the smelting of the cyanide gold, and the whole is amply safeguarded against theft. In connection with all the above machinery and plant, the necessary appliances for rapid handling and replacing of parts have been amply provided for."—DAVID GILMOUR.—*South African Mining Journal*, Feb. 12, 1910, p. 609. (A. R.)

FILTER FRAME AND CLOTHS FOR LEACHING TANKS.—"In the designing of filter frames for leaching tanks every inch or less of vertical height is of considerable moment when it is considered that all absolutely unnecessary space occupied by the filter frame represents so much available space lost just as many times per month as the tank or tanks in question are filled. As in many leaching plants (cyanide or other kind) there is a considerable tendency to use shallow tanks, the space occupied by the filter frame represents quite a considerable proportion of the total available space in the tank. The writer has found that in practical work one easy way at least of always having the total possible filtering area, and always having the filtering cloths free from slime or hard packed sand, and never after any length of time having slimes or sand passed through the cloths is as follows:—

The wooden filter frames are put in as usual in sections of convenient size, and the complete circle or rectangle, as the case may be, is centred by wooden wedges between the side of the tank and the caulking edge of the frame, thus leaving a caulking space about ⅞-in. wide. Cocoanut matting to exact size of the filter frame is laid down; this matting need not be sewn into one piece, but can be in pieces of convenient size lightly held together, and with a few stitches here and there. On top and all over this matting is laid jute cloth or duck in sections overlapping each other by a few inches, and, if preferred, loosely held by stitches here and there so as to form a lap joint. This jute cloth or duck is caulked tight into the space between the caulking edge of the frame and the side of the tank with ¾-in. manilla rope. The jute cloth or duck should be cut full and large enough, and should lie quite slack above the cocoanut matting in the tank so as to allow for shrinking, also several inches of spare end of cloth should appear above the caulking rope, thus insuring a good caulking joint. This jute cloth or duck forms the fixed filter cloth. On top of this in similar sections is laid a similar cloth with plenty of slack in it everywhere. This latter cloth need merely be tucked in by hand into the caulking space. This constitutes the loose or easily replaceable cloth. No shovelling laths (which diminish filtering area and get hard unleachable sand packed between them) are put on top of of the cloths, nor is any means used to protect the cloths, except that a constant small quantity of sand is always left in the tank on top of this upper loose cloth.

If at any time leaching is not as it should be, all sand is removed from upper cloth and the cloth is removed, cleaned, and replaced again, or a similar ready spare cloth is put in. This can be done very quickly seeing that there are no shovelling laths to move and replace, and because the cloth is in sections that can be handled by one man, and is not a large unwieldly sewn cloth of, say, 40 ft. or more diameter.

It is not found in cases where shovels are used for discharging that the upper cloths get cut; this is partly because the cloth is laid slack and not stitched tight, and partly because an inch or less rather of sand is always left.

The cloths can be kept in perfect filtering condition, and there is no lost filtering area, due to shovelling laths and packed sand between them, and as a result it is found in the case of sand being leached by cyanide that on discharging the residue the assay value is practically the same, regardless of whether the sample is taken from the upper or lower portion of the tank; this cannot be the case when the filter arrangements are imperfect, or of unequal resistance in different portions of the filtering area.

In order to be sure of keeping cloths and the space under them perfectly clean, it is essential in the case of a tank freshly filled with sand that before pumping on solution sufficient solution should be put in from the bottom through the leaching pipe so as to fully cover the upper filter cloth. It is also important that in case of a dry tank solution should not be pumped on when the leaching cock is open. With these latter precautions, the writer has found it impossible to get solutions leaching cloudy, or to get solid matter through the filters. If the upper loose cloth is cleansed occasionally, the fixed jute cloth (or duck) and the cocoanut matting will not require cleaning, or attention, or lifting till such time as, through rotting, they are too weak to carry the charge weight.

Some operators prefer to do without cocoanut matting; in this case another jute cloth (or duck) may take its place, but this is not necessary except with very deep tanks.

These precautions, etc., may sound totally unnecessary, but there is no doubt that the actual leaching and filtering arrangements are, through stress of work, not always what they should be.

If all the lower portion of the charge in the tank does not, after treatment, assay practically the same as the central or upper portions (that is assuming the

charge to be divided for sampling purposes into three equal horizontal layers) then improvement may be possible.

In the case of a new charge starting treatment (other things being equal), the metal value of the leaching solution should rapidly rise to its highest value, and then regularly fall to its lowest value at the time of discharge, when treatment is finished.

If the leaching solution values are plotted they should show rapid attainment to the highest metal contents, and then the gradual and regular diminution of leaching solution values. Granted that there is no other cause for irregularity, bad filters alone are sufficient to cause irregular leaching values."—*The Mining Journal*, Feb. 12, 1910, p. 197. (W. A. C.)

FLOTATION PROCESS FOR THE CONCENTRATION OF TIN ORES.—"The Elmore oil flotation process has recently found a new application to the concentration of tin ore. There is an old tin mine in Dolcoath, Cornwall, where for many years past those parts of the workings have been abandoned which contain mixtures of the ordinary tin oxide (cassiterite) with sulphides of various metals, such as copper, zinc, iron and arsenic. Such mixed ores were valueless, because if concentrated on water-dressing appliances, the sulphides, or, at any rate, a large proportion of them, were collected with the tin oxide, and the resulting product was too impure for the tin smelter. For the past two years an Elmore vacuum plant has been in operation there with complete success. The complex ore, after crushing, is treated in the vacuum machine, with the result that practically all the sulphides are removed as a concentrate which is almost free from tin, and substantially all the tin is left in the tailings from the Elmore machine, these tailings being passed on to usual water-concentrating appliances for the recovery of the tin, which is found in practice to be practically free from sulphides. Some of the Dolcoath tin ores are free from impurities except copper sulphide. From this ore the vacuum process produces a high-grade copper concentrate, in some cases as high as 19% of copper, which is readily saleable to the copper smelter.

Following the success at Dolcoath the process is about to be applied to two other widely separated parts of the world, viz., the Straits Settlements and South Africa. With the tin concentrates from South Africa a working trial of the Elmore process on about 35 tons of this material has recently been carried out, the results being given below.

	S %	Cu %	As %	Sn %
The impure tin concentrate treated assayed	1·69	0·26	2·91	72·38
The sulphide product assayed	20·77	2·78	33·0	3·64
The cleaned tin product assayed	0·12	0·03	0·14	74·9

The loss of tin in the sulphur product amounts to only 6 lb. per ton of material treated, while 95% of the total impurities have been removed."—E. S. ELMORE.—*Mining Journal*, Aug. 28, 1909; *Metallurgical and Chemical Engineering*, April, 1910, viii. 4, p. 204. (J. A. W.)

CYANIDING SILVER ORES IN MEXICO. — "The ores from the San Rafaël mines are derived from a system of veinlets which form the Vizcaina vein, or mother lode, of the district. They contain from 70 to 75% silica and from 10 to 20% calcite as gangue. The silver exists mostly as sulphide Ag_2S associated with varying proportions of iron sulphide FeS_2, lead sulphide PbS, and zinc sulphide ZnS. The gold occurs in nearly constant proportions of 4 or 5 gm. per ton in high and of 3 or 4 gm. in the low-grade ores. All ores carrying more than 300 gm. of silver to the ton are cyanided.

The ore is sorted by hand; crushed in Blake crushers, and sampled in the mine yard, after which the treatment is as follows: (1) Pulverised with stamps in a solution containing 2.5 kgm. of potassium cyanide to the ton of water; (2) concentrated on Wilfley tables; (3) the tailings are classified; (4) the sands from the classifiers are ground fine in Krupp tube mills; (5) the material from the Krupp mills is settled in pulp thickeners; (6) the thickened pulp is agitated in Pachuca pneumatic tanks; (7) the slime is separated from the solution by Moore vacuum filters; (8) the gold and silver in the filtered solution is precipitated by zinc filaments. The mill is driven by electric power, the motors for the most part being of the C.C.L. type of the Westinghouse company. Each 20 stamps has a 75 h.p. motor, and each Krupp mill a 100 h.p. motor. The water supply comes from the mine and is slightly alkaline. The surplus water from the filter plant is returned to the water tank, which has a capacity of 400 cub. meters.

The pumps used are of the vertical triplex power type with solid water end. The product of the tube mills and classifiers is handled with 5 pumps 10 in. × 54 in. The filter plant is served by one centrifugal pump, and the filters are worked by two Gould vacuum pumps, 14 in. × 14 in., although a third vacuum pump is held in reserve.

There are 80 stamps in use, 40 weighing 850 lb. set on timber foundations; and 40 weighing 1,250 lb. bolted to concrete foundations. The lighter stamps drop 7¾ inches, the heavier 6¾ inches, and both drop 104 times per minute. The El Oro solid guides with back plates are in use. The shoes and dies, which are of forged steel, last from 90 to 100 days.

The 850 lb. stamps crush from 3 to 3½ tons of ore daily through 10-mesh, 18-wire, 1·32-millimeter aperture screen.

The 1,250 lb. stamps crush from 6 to 6½ tons daily through the same mesh screen, and from 5½ to 6 tons daily through 12-mesh, 20-wire, 1·067-millimeter aperture screen. The total capacity of the stamps through 10-mesh screen is from 300 to 400 tons per day, but at present crushing is limited to 350 tons to prevent overloading the tube mills. From 7 to 8 tons of solution is used in the mortars per ton of ore crushed. One pair of rolls 36 in. × 16in. and two additional tube mills have been ordered, and when these are installed 8-mesh screens will be placed in the mortars, and it is expected that the capacity of the stamps will be raised to 500 tons of ore daily.

The following is a sizing test of the battery pulp when using 10-mesh screens of 1·32-millimeter aperture: +40, 37%; +60, 9%, +100, 7%; +150, 13%; +200, 4%; −200, 30%.

The total consumption of power in the mill is 1·68 horse-power per ton of ore treated, of which 1·05 horse-power is consumed in crushing. The mill solution is contained in two tanks 24 ft. × 20 ft. having a capacity of 200 cubic meters each. These tanks are used alternately to supply the stamps, Wilfley tables, classifiers and tube mills, and to receive the barren solution, being worked in cycles in order to avoid enrichment of the solution. With this system the working solution contains not more than 50 grams of silver per cubic metre. According to the class of ore treated, from 6 to 12 kilos of lime is added in the bins per ton of ore. The alkalinity of the solutions is kept at 1 kilo of calcium oxide *CaO* per ton of ore.

The mill solution averages ·25% potassium cyanide *KCN* to the ton; and there is a cyanide loss of about 600 grams of *KCN* per ton of ore treated in the stamp battery. The extraction of silver and gold in the crushing operations is high, averaging one month 38·3% of the silver and 70% of the gold contained in the ore.

One Wilfley table takes the pulp from 5 stamps. The 16 tables employed each receive from 17 to 30 tons of ore per day without being overloaded. The tables are also used as auxiliary classifiers, the slime overflow being sent to the pulp thickeners.

When the tables receive material at the rate of 30 tons daily, about 22% of the mineral is recovered as concentrate which contained 1 month: Silver, 25·586 kilograms per ton; gold, 126 grams per ton; iron, 34·45%; silica, 12·25%; lime, 3·40%; sulphur, 40·50%.

The concentrate is sold to the smelter, and, as previously stated, the tailing goes to the classifiers, five of which discharge their product into one tube mill. The proportion of solid to solution is at this point of the system abont 1 to 2·3, but as it leaves the classifiers the proportion is about 1 of solution and 2 of solid. From 50 to 70 tons of pulp and all the return from the tube mill are fed to each classifier. The classifiers discharged the following material during 1 month: 2·5% remained on 100 screen, 15·5% remained on 150 screen, 7·5% remained on 200 screen, 74·5% passed a 200 screen.

The five Krupp tube mills, which are 4 ft. in dia. and 20 ft. long, have Neal's baffle at both ends and El Oro liners. Each of the present mills is supplied with 100 horse-power motor, but this being unnecessary the new mills will be supplied with 75 horse-power motors. The original steel lining of the Krupp mill lasted 90 days, while the El Oro lining lasted 8 months.

Abont 350 tons of material pass through the 5 mills daily, but their capacity for regrinding battery sand is rated at 50 tons daily, the heads averaging: 48% on 100 screen, 39·6% on 150 screen, 4·2% on 200 screen, 8·2% passing 200 screen.

The sand entering the mill contains about 60% of moisture, and but 8·2% is finer than 200 mesh. The slime discharged from the classifiers had 74·5% that wonld pass a 200 mesh.

The pulp leaving the tube mill and entering the pulp thickeners has a consistency of 1 dry slime and 10 solution, but leaves the thickeners at 1 dry slime and 1·2 solution. The pulp is unwatered previous to agitation in the Pachuca tanks. The pulp thickening or unwatering tanks are five in number; three 20 ft. in dia. and 10 ft. high receive 70 tons dry pulp each, and two 24 ft. in dia. and 10 ft. high receive 100 tons each per day. The solution from the tube mills is raised to the unwatering tanks by 3 pumps, two 8½ in. × 10 in. and one 7 in. × 9 in. The capacity of the two first pumps is 350 gallons per minute, that of the second pump 250 gallons per minute.

Ten 15 ft. × 45 ft. Pachuca tanks are used to agitate the pulp at present, and two more have been ordered. All the pulp deposited in the Pachuca tanks is intended after agitation to go to the filters. The Pachuca tanks have a capacity of 183 cubic meters and are given an average charge of 100 tons of dry slime and 120 tons of solution. The proportion of 120 tons of solution and 120 tons of slime was tried, but it was found difficult to discharge the unwatering tanks, and the proportion of 1 part slime to 1·2 parts of solution was found more satisfactory. The charges mixed in the proportions of 1 dry slime to 2 and 3 of solution were also tried but gave more or less trouble in filtration.

For agitating the slime in the Pachuca tanks, and to supply the Moore filter tanks, one Ingersoll-Rand air compressor with double cylinders, 18¾ in. dia. and 16 in. stroke, delivers 700 cub. ft. of air at 9,000 ft. elevation, when making 100 revolutions per minute. When making 130 revolutions per minute the capacity of the compressor is 900 cub. ft. This quantity of air is however not required until the capacity of the plant is enlarged. The volume of air required for energetic agitation in the Pachuca tanks is estimated at 100 cub. ft. per minute with pressure at 25 lb. per sq. in. The pulp is agitated 36 hours with ·3% *KCN* solution and allowed to stand at rest from 12 to 24 hours, as it has been found that during this period the silver extraction increases about ·20 grams. The compressor receives power from a 100 horse-power motor. There is about ·18 horse-power per day consumed per ton of ore agitated by air. Before agitation is commenced the solution is brought up to ·3% *KCN* and 250 grams of acetate of lead per ton of ore is added. The average value of the solution and mineral in the Pachuca tanks in October was 337 grams silver and ·33 gram of gold per ton. After the material had been treated there were 71 grams silver, and after the slime had been discharged from the filter it contained ·55 grams of silver per ton.

The filtration plant contains two units of Moore A No. 3 type of vacuum filters; the filter leaves which are 10 ft. × 6 ft. have a capacity of 2·5 tons of dry slime per day. There are 160 filter leaves in operation and 80 held in reserve. The leaves are arranged in four baskets of 40 each. Each unit has three tanks 12 ft. × 22 ft. × 7 ft. for the filter leaves, and each tank has a capacity of 75 cub. meters.

In order to move the baskets there are two 25-ton capacity travelling cranes, each supplied with a 30 horse-power variable speed General Electric Co. motor for lifting, and a 5 horse-power motor for travelling and transferring the baskets.

Filtration is divided into three periods: the first, formation of the slime cake, which requires from 20 to 30 minutes to become 1 in. thick, according to the condition of the filters. In the second period 1 hour is consumed in washing the cake with barren solution. The third period is a 15-minute water wash given to the cake on the filters.

The first tank contains the agitated slime pulp to be filtered; the second tank contains the barren solution coming from the zinc precipitation boxes; the third contains water for washing out the barren solution, after which the cake is discharged from the filters and washed away. The discharged cake contains about as much water as slime, that is 1 to 1. On each basket is a spiral tube 16 in. in dia. and 20 ft. long, in which is maintained a vacuum to hold the slime cake on the filter leaves during the 5 minutes required to transfer the baskets from tank to tank. Each unit has a vacuum pump which draws the various solutions through the filter leaves. During October the water discharged assayed 4·5 grams of silver per ton; the cake washed assayed 50 grams and unwashed 55 grams per ton. In order to clarify the solutions, sand filters are used prior to precipitation in the zinc boxes.

Precipitation.—Zinc shavings, ·006 in. thick, are used in precipitation.

Two tons of solution to the ton of ore are precipitated.

The heads in the boxes assay from 200 to 300 grams of silver; the tailings, 2 grams.

The clean-up is made weekly.

The short zinc which remains on a 20-mesh screen is returned to the boxes; the shorts on 40-mesh are melted.

The precipitate is pressed and dried to 20% moisture on a 24 in. × 24 in. Shriver press.

Melting.—Coke furnaces and Dixon's No. 300 graphite crucibles are employed.

The precipitate is melted with the following flux: 15% broken glass; 6% borax glass; 4% soda carbonate. The short zinc which remains on the 40 mesh screen is fused with 20% broken glass; 8% borax; 6% soda carbonate.

The bars, without remelting, assay from 920 to 950 grams silver, and 5 grams of gold to the kilo of silver.

The slags are crushed quarterly in a battery and concentrated on a Wilfley table. The concentrates are melted and the tails treated in the cyanide plant or sold.

RESUME.

The resume of the November results is given below:

Tons crushed, 8,393.	
Assays:	
Ore contents:	
Silver	·901 kgm
Gold	4·430 gm
Concentrates:	
Silver	25·348 kgm
Gold	141·600 gm
Heads of Pachuca tanks:	
Silver	·397 kgm
Filter discharge:	
Unwashed, silver	·071 kgm
Washed, silver	·063 kgm
Extraction:	Per cent.
In concentration silver	24·19
Cyaniding:	
Crushing	31·74
Agitation, silver	36·18
Total extraction, by assays, silver ...	92·11
Total extraction, by concentn., silver	91·43
Total extraction, by concentn., gold...	97·08
Consumption per ton of ore:	
Sodium cyanide, 128 per cent. ...	1·078 kgm
Lead acetate	·317 kgm
Lime	6·706 kgm
Flint pebbles	1·227 kgm
Zinc	·843 kgm
Coke	1·646 kgm
Borax	·084 kgm
Soda carbonate	·044 kgm
The cost per ton was:	
General expenses	$ ·78
Grinding and concentrating ...	1·18
Cyaniding	1·32
Pumps and compressor	·15
Filtration, including royalty ...	·33
Precipitation and melting ...	·44
Total, Mexican money ...	$4·20

—E. GIRAULT, *Mining and Metallurgical Institute of Mexico.—Mines and Minerals*, May, 1910, p. 618. (A. R.)

MANUFACTURE OF IRON AND STEEL.—"1. There are large iron ore deposits in various parts of the Colony, but the most accessible, near Pretoria and Airlie, are, so far as outcrop samples indicate, very silicious, and although capable of producing a good pig iron would be expensive to smelt.

2. The limestone deposits suitable for blast furnace purposes are very limited, and owing to their occurrence the mining costs are high. There are large deposits of silicious limestone, but these could not be economically used alone for iron smelting, although to a limited extent they might be used in admixture with the more pure varieties.

3. It has not yet been definitely determined if a hard metallurgical coke suitable for blast furnace practice can be made from local coals, although experiments on a small scale give very promising results. In any case the coke would not be a high-class coke and it would be expensive, owing to the costs incurred when mining only selected portions of the coal seams.

4. Assuming a satisfactory coke can be made, pig iron suitable for foundry purposes and for the manufacture of steel by the Basic process can be made from Pretoria and Airlie ores, in admixture with not less than about 20% of rich non-silicious iron ores which exist in comparatively small quantities in several parts of the Colony. Such a pig iron would cost very considerably more than the same quality of iron produced in Europe.

5. Pig iron cannot be commercially manufactured in the electric furnace from local ores with current costing anything approaching the price at which it can be at present generated in the Colony.

6. Taking into consideration all the local conditions respecting the supply of raw materials, cost of labour, and especially the restricted nature of the present market, and the large variety of finished products it would be necessary to manufacture in order to maintain a fair output, I do not consider that an iron and steel plant to manufacture rails, sleepers, bars, wire, galvanised sheets, etc., could be commercially successful.

7. It would be quite impossible to export either pig iron or manufactured steel in competition with older iron-producing countries much more favourably situated. It would not be possible to compete with imported material near the chief seaports without a very substantial bounty from the Government or protection in some equivalent form.

8. If all the Railway Departments of South Africa would jointly agree to take substantial yearly quantities of rails and sleepers, at fair market prices, this would form a sound basis for the establishment of an iron and steel industry, which, with the assistance of a reasonable bounty, should expand as the requirements of South Africa increase.

9. A small electric furnace plant designed to produce high-class steel from the large accumulations of scrap in the Colony should give excellent results, and such a project should receive every possible support from the Government.

In view of the possibility of future developments, I would strongly recommend that systematic prospecting of the most accessible ore deposits be undertaken without delay, to prove them both as to quality and quantity, and that the question as to the possibility of producing a metallurgical coke be definitely settled by coking experiments in modern coking ovens."—F. W. HARBORD.—*Transvaal Mines Department Report*, June, 1910, p. 17. (A. McA. J.)

STUDIES IN CONCENTRATION. — "Theodore J. Hoover presents a discussion of various points in connection with the calculation of the proportion of recovery in concentration. He commences with a demonstration of his method of calculating the percentage of recovery and ratio of concentration when only the percentage assay-values of the ore, concentrate and tailing are known, and not the relative tonnages. This is quite a simple mathematical

problem. If we take x, y and z, as the undetermined weights of the ore, concentrate, and tailing, and a, b and c as the percentage of metal contained in each as ascertained by the assay of a sample, then the constitution of the ore, concentrate, and tailing, and the amount of metal coutained in the three, are expressed by the two formulæ:

$$(1)\ x = y + z$$

$$(2)\ ax = by + cz$$

The ratio of concentration is expressed by the fraction $\frac{x}{y}$, and the percentage of recovery by $\frac{100\ by}{ax}$. Eliminating z from the two equations, by multiplying (1) by c, and substracting, we get $(a-c)\ x = (b-c)\ y$. Thus $\frac{x}{y}$, the ratio of concentration, is represented by $\frac{b-c}{a-c}$. Substituting this ratio in the formula for the percentage of recovery we get the percentage expressed thus: $\frac{100\ b\ (a-c)}{a(b-c)}$. These two formulæ are independent of the value of x, y and z. It will be seen also that if we have the percentage assay-values of the ore, concentrate, and tailing, we can deduce the relative amounts of concentrate and tailing for, say, 100 parts of ore. This is done by substituting 100 for x and $100-y$ for z in equation (2) and solving for y.

As a rule the percentage of recovery and the ratio of concentration are ascertained by measuring x, y, a and b. There are many occasions when this cannot be so conveniently and easily done as by this method based on sampling. Mr. Hoover points out that his method depends for its success on the degree of accuracy of the sampling and of the assay; we do not suppose that the possibility of error need be greater than when the ore and concentrate are weighed daily and their contents sampled. The two methods might with advantage be used as a check one on the other. Mr. Hoover proceeds to show how his method can be employed for studying the accuracy of analyses, and how errors are introduced into the figures for extraction by inaccurate samplers and chemists."—*Mining Magazine*, July, 1910, p. 74. (A. R.)

HIGH TEMPERATURE MEASUREMENTS AND GAS THERMOMETRY.—"Before the publication of the recent measurements in the Geophysical Laboratory, the region of accurate gas thermometry ended with the temperature 1,150°. The common practice for some years has therefore been to measure temperatures beyond that point by extrapolating the parabolic curve out to the desired temperature—often as high as the melting point of platinum (1,755°). Extended extrapolation is always a matter of grave uncertainty, but no other convenient method was available, and after all, observations of this kind with the thermoelement could be translated in terms of an observed scale whenever one should be provided. It happens in this particular case that the extrapolation (from 1,150° to 1,755°) comes out just 50° low at the platinum melting point.

Now that the gas thermometer is able to make direct measurements of temperature up to 1,550°, it is merely necessary to include several thermoelements in the furnace with it, to evaluate their temperature curves by direct comparison, and afterward to determine with them a series of standard fixed temperatures like the melting points of metals, minerals, or salts, which are available for general use.

This procedure has been followed in the recent investigation with the results which are tabulated below. The pure metals are readily obtainable and the minerals easily made up, so that anyone may now calibrate his thermoelement by noting its electromotive force at the (known) melting temperature of three or four of these substances, preferably choosing such as will include the region in which he expects to use the thermoelement.

Such a calibration requires to be checked from time to time by repeating one or more of the melting point observations, particularly if the element is exposed to contamination from vapours of other metals. This contamination is particularly disastrous if a thermoelement is exposed to the vapour of iridium, which explains the remark made above, that in our experience a bulb made from an alloy of platinum and rhodium is to be preferred to the iridium alloy in use at the Reichsanstalt.

The standard melting points in the table below are taken from observations by Day and Sosman and include, for comparison, the standard melting points published by the Reichsanstalt in 1900 and now in general use:—

Substance.	Point.	Atmosphere.	Crucible.	Temperature.	The Reichsanstalt Scale.
Zinc	Melting and freezing	Air	Graphite	418·2° ± 0·3	419·0°
Antimony	,,	Carbon monoxide	,,	629·2 ,, 0·5	630·6
Silver	,,	,,	,,	960·0 ,, 0·7	961·5
Gold	,,	,,	,,	1,062·4 ,, 0·8	1,064·0
Copper	,,	,,	,,	1,082·6 ,, 0·8	1,084·1
Diopside (pure)	Melting	Air	Platinum	1,391·2 ,, 1·5	
Nickel	Melting and freezing	Hydrogen and Nitrogen	Magnesia and magnesium aluminate	1,452·3 ,, 2·0	
Cobalt	,,	,,	Magnesia	1,489·8 ,, 2·0	
Palladium	,,	Air	Pure magnesia	1,549·2 ,, 2·0	1,575·2
Anorthite (pure)	Melting	Air	Platinum	1,549·5 ,, 2·0	
In addition, the following temperatures were incidentally obtained:					
Cadmium	Melting and freezing	Air	Graphite	320·0 ,, 0·3	321·7
Aluminium	Freezing	Carbon monoxide	,,	658·0 ,, 0·6	657·
Platinum	Melting	Air	—	1,755	

A new estimate of the melting point of platinum, which has not yet been directly determined with the gas thermometer, is included in the above list. It was obtained in this way: There is a remarkably close agreement between independent determinations of the temperature interval between the melting points of palladium and platinum:—

Nernst and von Wartenberg	204°
Holborn and Valetiner (at the Reichsanstalt)	207°
Waidner and Burgess (at the Bureau of Standards)	207°

If we, therefore, simply add 206° to the palladium melting temperature (1,549°), we obtain 1,755° as the melting point of pure platinum, with an absolute error of perhaps no more than ± 5°.

By way of conclusion, the following estimate is offered of the accuracy of existing standards of temperature within certain temperature intervals:—

0° – 100° ± 0·002°
100° – 300° ,, 0·05°
300° – 1,100° ,, 0·8°
1,100° – 1,550° ,, 2°
1,550° – 1,750° ,, 5°.'

—A. L. Day.—*Metallurgical and Chemical Engineering*, May, 1910, Vol. viii., 5, p. 260. (J. A. W.)

The Silica Filter in Slimes Treatment.—"An unusually interesting paper on an improvement in cyanide practice has been contributed by Mr. E. Gybbon Spilsbury to the transactions of the American Institute of Mining Engineers. It relates to a porous diaphragm which has been found of great value in the aeration of slimes, and has been successfully used for revolving filters and other purposes, to which a material of that kind is obviously well adapted. The following notes refer to the employment of the filter in slimes treatment:—

'The recovery of gold and silver from their ores by means of the cyanide process has been so successful in the last few years that any radical improvement would seem impossible; yet the appliance to which I wish to call attention in this paper is really a radical departure from the methods now in general use.

'The most modern and approved of these, known as the all-sliming method, depends for its success on the grinding of the ore so fine that practically 90% of it will pass through a 200-mesh screen. The slimes thus produced are then agitated and aerated in tanks of various types.

'The main object of this treatment is to insure such a thorough admixture of the pulp and solution that every particle of the ore is surrounded by a volume of solution sufficient to insure the dissolving of the whole of the gold and silver content. In addition, in order to expedite the action of the cyanide, and to oxidise such elements as would, if left in their active state, become cyanicides, air is blown in under pressure. In the Pachuca tank, this air is the active and sole means of agitation.

'The chief objections to all the methods of agitation used heretofore are the expense of operating and maintaining the mechanical devices for keeping the pulp in suspension, and the length of time required to obtain a fairly complete extraction of the values. These objections are inherent to any method of agitation effecting a circulation of the pulp and the solution together by giving the whole mass a circular movement, as in the low-tank system, or a verticular circulation, as in the Pachuca-tank system. In both methods, the particles of ore are kept travelling in the same direction as the solution, and with very little difference of speed; so that, while the whole mass is in violent motion, the relative positions of the ore-particles to their surrounding medium of suspension change but very slowly, and consequently the length of time required for a given extraction is much greater than would be necessary if the movements of the solution and the pulp were not coincident.

'The improvement here described is a purely mechanical one, devised to meet this requirement. It rests simply on the discovery of a method of manufacturing a diaphragm of silica sponge, which, while strong enough to support heavy weights, is so evenly porous throughout that air can be passed through it with practically little resistance, and in which, nevertheless, the pores are so minute that no solid matter, however, finely divided, can pass through or even into it.

'In practice, this diaphragm is placed in the tank as a false bottom, resting on light channel-iron bars, 4 in. above the real bottom. The plates are either 12 × 12 or 12 × 20 in. in size, and are secured to the channel-iron supports, along the lines of intersection, by 0·25 in. carriage bolts. When the plates are all laid, oakum is driven into the joints, which are then made completely tight by pouring in liquid cement.

This simple operation completes the whole installation. The charge is now run into the tank, and air is admitted from below, under a pressure of from 2 to 6 lb. only, depending on the depth of the charge. Immediately the whole charge becomes a seething mass of uniform but gentle agitation, in which every particle of ore is in constant motion.

'No pressure is exerted on any of the air particles after they have passed through the pores of the diaphragm. They simply levitate up through the mass by reason of their lower specific gravity. No distinct streams or lines of agitation are preceptible to the eye, but the charge shows a very distinct increase of volume, amounting to more than a foot of height in a 6 ft. tank; and the surface becomes covered with a coating of foam or air-bubbles, the thickness of which depends on the volume of air blown through the diaphragm. The full capacity of the sponge varies from 5 to 5·5 cub. ft. of air per minute per square foot of area at 1 lb. pressure.

'Under these conditions the action of the cyanide is very rapid and intense. In 80% of the runs made an extraction of over 50% of the combined gold and silver values has been obtained within the first hour of agitation; and while, in experimental work, our treatment is usually carried on for 12 hours, six hours will probably be found to be the economical period in general practice.

'It is found that the consumption of cyanide per ton of ore treated under this method is much less than in either the Pachuca or the mechanically-agitated tanks. This, I believe, is due chiefly to the briefer exposure of the cyanide to the oxidising effect of the air, but also to the circumstance that we are able to treat effectively a much thicker pulp than the other methods of agitation will permit. With a proportion of 1·5 of solution to 1 of ore, we can obtain the quickest extraction; but, for facility of charging and discharging tanks, we generally make the mixture 2 to 1.

'Records of work done with this porous diaphragm on a commercial scale, in one of the large mills of the Guanajuato Development Co., show what remarkable extractions are obtained by this method.

'One of the important questions we had to study in the use of this material was, whether the pores of the sponge would not sooner or later become filled and choked by the very finest particles of ore, thus impairing the efficiency of the plates.

All our experience hitherto goes to prove that no such stoppage need be feared. The pores are so minute and so irregular in shape that apparently no solid matter can find entrance. In treating certain classes of ore we do find that, after a certain number of hours, the air-pressure begins to increase, by reason of a closing of the surface-pores of the diaphragm; but examination under the microscope has shown that this is due to a gradual deposit of lime carbonate on the surface of the plates, formed at the moment

of contact with the air with lime in solution. The removal of this coating, however, offers no difficulty. It can be done, between charges, either by sweeping the surface with a wire broom or by washing it with a weak solution of hydrochloric acid. In either case, the complete removal of the deposit takes place instantaneously, and the air-pressure drops to the normal.

'The credit for the successful application of this porous medium to the cyanide process is due to J. E. Porter, of Syracuse, N.Y., who, having acquired the material for an entirely different purpose, conceived at once its possibilities in the cyanide field, and, by a long series of careful experiments, developed the many advantages of its present adaptation to that field.

Besides the employment of this silica sponge in the treatment tanks, its usefulness has been demonstrated in the filtering of the solutions, and as a clarifier. Several types of silica-sponge filters are now building of which probably, the simplest consists of a table, from 25 to 30 ft. long and 8 ft. wide, the top of which is an air-tight pan, connected with an exhaust pump. The pulp being run over the table, the solution is drawn through, leaving a dry cake of the desired thickness. Wash-water is then flowed over this cake, and it is washed in the usual manner. When the cake is finally dried, the table is tilted to a vertical position, and the cake is blown off by air-pressure. The mineral sponge always contains a certain amount of moisture, which causes a film of water to exude when the air-pressure is turned on: and this film, acting as a lubricant, aids the cake to free itself, so that the separation is immediate.

This filter has a capacity of from 40 to 50 tons of dry pulp per day. The resulting cake contains less than 23% of moisture when discharged. The simplicity of construction of this filter, its indestructible filtering medium, and the absence of the innumerable valves and fittings required by all filters of the leaf-type, will recommend it strongly to mill men generally. I think it will increase the saving of values wherever it is introduced."—E. GYBBON SPILSBURY, *American Institute of Mining Engineers. — South African Mining Journal*, June 18, 1910, p. 461. (A. R.)

MINING.

THE RELATION OF COSTS TO PROFITS.—"*Hypothesis.*—Minimum waste milled increases the costs and profits per ton milled (waste and reef), but the profits are increased in a greater ratio, and, although the pay limit is higher, some of the unpayable ore reserves becomes payable, or maximum reef milled decreases the costs and increases the profits per ton of reef milled, and some of the unpayable ore reserves become payable.

Introduction.—Undoubted a new era has been heralded on the Rand recently by the commendable reductions in working costs, and the pace has been set so strong that it may be wise to pause and endeavour to ascertain whether profits per unit of gold may not be sacrificed on the high altar of costs. I am fully cognisant of the fact that this subject has received much attention, but I am not aware of any similar table having been prepared.

Basis of Calculations.—I have taken as the basis of my calculations a mine which has 300 claims, which contain two reef bodies on a 30° dip. The average reef thickness of each ore body is 24 in., which will produce a value of 6 dwt. over a 60 in. stope width. The reduction plant consists of 200 stamps and four tube mills, with a crushing capacity of 40,000 tons per month. From this starting point I have calculated what would result if the external waste (waste above and below the reef channel at the stope face) were gradually reduced from 36 in. to 12 in.—vide columns A, B, C, D, E, F, G,—and at the same time the maximum amount of waste was sorted out on the surface, which will ensure minimum waste or the maximum reef being milled. Columns A and G are examples of two extreme cases, but they have been purposely included in the accompanying table to emphasise the principle I am endeavouring to establish.

Waste Sorted out on the Surface.—The basis mine (A) sorts 20% of waste on the surface from stopes of an average width of 60 in. The C mine sorts 20% also, but reduces the stope width to 54 in. by leaving 6 in. of waste in the hanging and foot, which, when compared to the basis mine, is equivalent to sorting 10% on the surface; this, together with the 20% sorted, makes an equivalent of 30% sorted compared to the 20% of the mine taken as the basis. Mine G only sorts 10% on the surface, but through reducing its external waste to 12 instead of 36 in., is credited with 50% less waste milled.

Ton Duty in Stopes.—I have assumed, for simplicity, that all the rock is broken by machines, and that if the duty in the 60 in. stope were 8·75 tons it would gradually drop to 5·25 tons in a 36 in. stope. I shall provide the reason of this assumption later.

Lb.-inch Duty.—The above ton duty is based on an equal efficiency—that is 292 lb. broken per in. of stope width. The term in. dwt. (should be dwt. in.) is familiar on the Rand, and also, I consider, that as efficiency is the only criterion of work done that the term lb. in. should replace the ton duty. This latter comparison is dependent mostly on the stope width and the duties in two different stope widths are not comparable, but if the ton duty is divided by the stope width and expressed in lb. for convenience, the resultant factor is a direct reflex of efficiency. The formula is:—

Lb. in. duty = lb. broken per in. of stope width per machine per shift.

$$= \frac{\text{Fathoms} \times 36 \times \text{Stope width in in.} \times 2{,}000 \text{ lb.}}{12 \times 12 \times \text{Machine shifts} \times \text{Stope width in in.}}$$

$$= \frac{500 \times \text{fathomage}}{\text{Machine shifts}}$$

One of the most uncertain factors in determining the ton duty is eradicated in the calculation, viz., the stope width. In a similar manner the efficiency cost of breaking ground is expressed by the formula:

$$\text{Cost per ton in} = \frac{80 \times \text{cost}}{\text{Fathoms}} \text{shillings}$$

which also does away with the stope width factor and gives a true reflex of efficiency.

Tons Mined.—In order to mill 40,000 tons per month, the 60 in. mine would mine 50,000 tons, when 20% was sorted, and the 36 in. mine would mine 44,444 tons, as 10% only, or 4,444 tons are sorted on the surface. The first advantage is here noted that 5,446 tons less would be hoisted, etc.

Feet Developed.—As the stope width decreases the lateral development must be increased to develop an equal tonnage. My argument here is that the tons developed should equal the tons stoped plus 25%, inasmuch as 5% of the tons developed would be occupied by pillars, which could only be extracted at a later date; 10% is represented by worthless bodies of reef exposed; and 10% for contingencies, such as dykes, faults, etc. It is, therefore, seen that the development footage would be increased from 1,561 ft. per month to 2,219 ft., and since the tonnage per

THE RELATION OF COSTS TO PROFITS.—Minimum of Waste Milled increases the Costs and Profits (but the Profits are increased in a greater ratio) and unpayable Ore Reserves become payable.

Stope widths (in inches)	60—A.	60—B.		54—C.		54—D.		48—E.		48—F.		36—G.	
	Mine taken as the basis.	Costs and Profits.	Equal Profits.	Costs and Profits.	Equal Profits.	Costs and Profits.	Equal Profits.	Costs and Profits.	Equal Profits.	Costs and Profits.	Equal Profits.	Costs and Profits.	Equal Profits.
Percentage sorted on surface	20	30	—	20	—	25	—	15	—	20	—	10	—
Equivalent percentage waste not milled (through decreased stope width)	—	—	—	10	—	10	—	20	—	20	—	40	—
Percentage decrease of waste milled	20	30	—	30	—	35	—	35	—	40	—	50	—
External waste above reef in stopes	19	18	—	15	—	15	—	12	—	12	—	6	—
Reef	24	24	—	24	—	24	—	24	—	24	—	24	—
External waste below reef in stopes	18	18	—	15	—	15	—	12	—	12	—	6	—
Ton duty	8·75	8·75	6·74	7·87	6·22	7·87	5·69	7·00	4·92	7·00	4·50	5·25	2·63
Tons stoped	46,829	53,643	—	46,483	—	49,651	—	43,418	—	46,137	—	39,940	—
Tons mined	50,000	57,143	—	50,000	—	53,333	—	47,059	—	50.000	—	44,444	—
Tons milled	40,000	40,000	—	40,000	—	40,000	—	40,000	—	40,000	—	40,000	—
Tons developed	62,439	71,524	—	61,978	—	66,201	—	57,891	—	61,516	—	53,253	—
Feet developed	1,561	1,788	—	1,722	—	1,855	—	1,809	—	1,922	—	2,219	—
Value of tons from stopes	6·00	6·00	—	6·59	—	6·59	—	7·33	—	7·33	—	9·54	—
Increase in value	—	—	—	·59	—	·59	—	1·33	—	1·33	—	3·50	—
Value of tons milled	7·05	7·99	—	7·73	—	8·19	—	8·08	—	8·54	—	9·76	—
Value of yield	6·63	7·51	—	7·30	—	7·74	—	7·68	—	8·11	—	9·37	—
Value of yield (shillings)	27s. 11d.	31s. 6d.	—	30s. 8d.	—	32s. 6d.	—	32s. 3d.	—	34s. 1d.	—	39s. 4d.	—
Increase in value	—	3s. 7d.	—	2s. 9d.	—	4s. 7d.	—	4s. 4d.	—	6s. 2d.	—	11s. 5d.	—
Stoping per ton stoped	4s. 6d.	4s. 6d.	5s. 10d.	5s. 0d.	6s. 4d.	5s. 0d.	6s. 11d.	5s. 8d.	8s. 0d.	5s. 8d.	8s. 9d.	7s. 6d.	15s. 0d.
Shovelling and tramming, per ton mined ..	1s. 6d.	1s. 6d.	1s. 6d.	1s. 8d.	1s. 9d.	1s. 8d.	1s. 9d.	1s. 9d.	2s. 0d.	1s. 9d.	2s. 0d.	2s. 0d.	2s. 6d.
Development per ton mined	1s. 9d.	1s. 9d.	1s. 9d.	1s. 11d.	1s. 11d.	2s. 0d.	2s. 0d.	2s. 2d.	2s. 2d.	2s. 2d.	2s. 2d.	2s. 9d.	2s. 9d.
Total costs, per ton milled	16s. 1d.	18s. 0d.	19s. 8d.	17s. 3d.	18s. 10d.	18s. 3d.	20s. 8d.	17s. 7d.	20s. 5d.	18s. 5d.	22s. 3d.	19s. 6d.	27s. 6d.
Total profits, per ton milled	11s. 10d.	13s. 6d.	11s. 10d.	13s. 5d.	11s. 10d.	14s. 3d.	11s. 10d.	14s. 8d.	11s. 10d.	15s. 8d.	11s. 10d.	19s. 10d.	11s. 10d.
Percentage increase in costs	—	12	—	7	—	14	—	·9	—	15	—	21	—
Percentage increase in profits	—	14	—	13	—	20	—	24	—	32	—	68	—
Cost per ton of reef	32s. 2d.	31s. 7d.	—	31s. 1d.	—	30s. 10d.	—	30s. 0d.	—	29s. 7d.	—	26s. 4d.	—
Profit per ton of reef	22s. 11d.	23s. 6d.	—	24s. 4d.	—	24s. 7d.	—	25s. 9d.	—	26s. 4d.	—	30s. 1d.	—
Pay limits at stope faces	3·60	3·61	—	3·85	—	3·86	—	4·12	—	4·09	—	4·82	—
Ore reserves which become payable with decreased waste milled	3·61	3·61	—	3·93	—	3·93	—	4·34	—	4·34	—	5·54	—
Gain in ore reserves	·01	·00	—	·08	—	·07	—	·22	—	·25	—	·72	—
Ore reserves which become unpayable with increased waste milled	3·17	3·17	—	3·45	—	3·45	—	3·79	—	3·79	—	4·82	—
Loss in ore reserves	·43	·44	—	·40	—	·41	—	·33	—	·30	—	0·00	—
Life of mine (years)	23·10	20·22	—	20·83	—	19·53	—	19·64	—	18·48	—	15·60	—
Percentage decrease of life	—	12·5	—	9·8	—	15·5	—	15·0	—	20·0	—	32·5	—
Equivalent stamps	200	225	—	220	—	231	—	230	—	240	—	265	—
Equivalent tube mills	4	4·5	—	4·4	—	4·6	—	4·6	—	4·8	—	5·3	—
Net life profit	5,829,6[illegible]8	5,842,771	—	5,981,064	—	5,965,018	—	6,173,971	—	6,219,185	—	6,676,972	—
Increase in profit	—	13,163	—	151,456	—	135,490	—	344,363	—	389,577	—	847,364	—
Yearly dividend	50·47	57 79	—	57·43	—	61 09	—	62·87	—	67·30	—	85·60	—
Present worth—6% and 3½%	2,839,694	3,047,613	—	3,074,171	—	3,163,973	—	3,265,845	—	3,384,921	—	3,913,632	—
Present worth per share	£5 13 7	£6 1 11	—	£6 3 0	—	£6 6 7	—	£6 10 8	—	£6 15 5	—	£7 16 7	—
Percentage increase in present worth	—	7·3	—	8·3	—	11·4	—	15·0	—	19·2	—	20·2	—
ORE RESERVES—													
Payable ore reserves—Waste	4,158,675	4,158,675	—	3,541,249	—	3,541,249	—	2,938,797	—	2,938,797	—	1,524,848	—
,, ,, Reef	2,772,450	2,772,450	—	2,832,999	—	2,832,999	—	2,938,797	—	2,938,797	—	3,049,695	—
,, ,, Total	6,931,125	6,931,125	—	6,374,248	—	6,374,428	—	5,877,594	—	5,877,594	—	4,574,543	—
Percentage decrease of waste	—	—	—	14·8	—	14·8	—	29·3	—	29·3	—	63·4	—
Percentage increase of reef	—	—	—	2·2	—	2 2	—	6·0	—	6·0	—	10·0	—
Percentage increase of payable reef claims worked	—	—	—	2·2	—	2·2	—	6·0	—	6·0	—	10·0	—
Percentage decrease of nominal ore reserves ...	—	—	—	8·0	—	8·0	—	15·2	—	15·2	—	34·0	—

ft. is less, this item would decrease from 62,439 to 53,515 tons, as a minimum of waste would be mined in the latter case.

Value of Tons from Stopes.—If the basis mine produced rock of 6 dwt. over the stope face, assuming the reef value at 31·96 dwt. and the waste value ·7 dwt., then the 36 in. mine would increase its stope grade to 9·54 dwt.

Value of Yield.—The yield for the 60 in. stope would be 6·63 dwt., or 27s. 11d., and that of the 36 in. stope 9·37 or 39s. 4d., or an increase of 11s. 5d. per ton milled. I have assumed the extraction to be 94% and 96% respectively.

Stoping Costs.—If a ton of rock costs 4s. 6d. to stope in the 60 in. stope, I have—for the time being—given a cost of 7s. 6d. or a 3s. increase for the 36 in. stope.

Tramming and Shovelling Costs.—I have added 6d. a ton for the decreased stope width.

Development Costs.—I have used the cost of 56s. per ft., and consequently the 36 in. mine would incur an increase of 1s. per ton milled for the extra development.

Reduction Costs.—The reduction costs would vary but little.

Cost per Ton Milled.—The above costs for the 60 in. mine total 16s. 1d. per ton milled, and they increase to 19s. 6d. for the reduced stope width mine.

Profits per Ton Milled.—The profits are, however, increased when mining the same reef at a uniform value, with similar mining and reduction conditions, from 11s. 10d. to 19s. 10d. per ton milled, or an increase of 8s per ton.

Equal Profits.—I shall now endeavour to prove that although in actual practice the above results may not be achieved, yet at the same time it is possible to prove that the principle advocated is true. I state that the profits would increase from 11s. 10d. to 19s. 6d. Assume that this deduction is untenable, and that the costs would rise higher than 19s 6d. Then, if the profit in the 36 in. stope were only 11s. 10d., the costs could rise from 19s. 6d to 27s. 6d., which would enable the stoping costs to rise from 4s. 6d. to 15s. per ton, when the width was cut down from 60 in. to 36 in., which, in my opinion, is reducing this possibility to an absurdity and quite without the pale of reason. It would also be seen how simple it is to reduce the costs from 19s. 6d. to 16s. 1d., without an increased efficiency. I consequently claim that although I do not maintain that results achieved would be realised to a penny, yet the principle is vindicated.

Percentage Increase in Costs and Profits.—Mine B increases the costs 12% and the profits 14%, or a ratio of 1 to 1·2. Mine C drops the costs from 18s. to 17s. 3d., through shovelling, tramming, hoisting, etc., less waste, which is a 7% increase on mine A, and the profits are increased in the ratio of 1 to 1·9. and so on to the 36 in. mine, which increases the profits over the costs by the ratio of 1 to 3·2, which tends to prove my assumption that by increasing the waste milled by the costs and profits are increased in a greater ratio.

Costs and Profits per ton of Reef Milled.—The stope width of 60 in. is composed of 24 in. of reef and 36 in. of waste, but 20% is sorted out on the surface, which accounts for 12 in., consequently every ton going to the mill consists of 24 in. of waste, that is half reef and half waste. If the cost per ton milled of waste and reef is 16s. 1d., the cost per ton of reef is 32s. 2d. In a similar manner it will be seen that the majority of the rock broken, hoisted and milled in the 36 in. stope is reef which works out at 74% reef and consequently although the cost per ton of waste and reef milled increases to 19s. 6d., the cost per ton of reef decreases from 32s. 2d. to 26s. 4d. And similarly the profits increase from 22s. 11d. to 30s. 1d. for every ton of reef milled. This proves my alternative hypothesis that maximum reef milled decreases the cost and increases the profit per ton of reef milled. Our mines have been laid out to mine and mill the maximum of reef, not waste.

Pay Limits at Stope Faces.—The pay limits rise from 3·60 dwt. to 4·82 dwt.

Value of tons at Stope Faces.—The 6 dwt. value over 60 in. increased to 9·54 dwt. in the 36 in. stope, or an increase of 3·54 dwt. Since the pay limit increases only 1·22 dwt., there is a direct gain of 2·32 dwt. in the increased stope value over the pay limit. In other words, although the pay limit increases, the value of the stope increases in a greater ratio.

Ore Reserves which become Payable.—The 60 in. stope would require 3·61 dwt. to be payable. Cut this face down to 36 in. and the value becomes 5·54 dwt., which is ·72 dwt. greater than the pay limit of 4·82 dwt. in the 36 in. mine. And conversely, if 4·82 dwt. were being broken in a 36 in. stope—this is equivalent to 3·17 dwt. in a 60 in. stope—which means that the narrower stope is mining rock which would be unpayable by ·43 dwt. if the width were carried to 60 in. I consequently consider that the principle is proved which enunciates 'some of the unpayable ore reserves become payable when a minimum of waste is milled.'

Life of Mine.—The 60 in. mine would have a life of 23·1 years, whereas the 36 in. mine would be worked out in 15·6 years, or a percentage decrease of 33%.

Stamps and Tube Mills.—If the plant were 200 stamps and 4 tube mills, and it was required that more reef should be milled, the 60 in. mine would have to add 65 stamps and 1·3 tube mills to mill as much reef as the 36 in. mine. How much simpler and less costly it is to reduce the waste milled. Instead of spending money on extra stamps, spend more money on the sorting plant first. I am conducting experiments to see what additional waste can be sorted out of the product going to the mill after coarse and medium waste has been eliminated.

Increase in Life Profit.—Although the reduced width of mine has a decreased life of 7·5 years, yet yet in the shorter time it would yield £847,000 more profit from the same reef.

Present Worth of Shares.—The present worth of 6% and 3½% is increased from £5 13s. 7d. to £7 16s. 7d., or 20·2%.

Nominal and Actual Ore Reserves.—The nominal ore reserves, that is the reserves of reef plus waste, are decreased from 6,931,125 tons to 4,574,543 tons, but the actual reserves, that is the reef tons, are increased from 2,772,450 tons to 3,049,695 tons, which is accounted for by 10% of the unpayable ore reserves becoming payable.

Payable Reef Claims Worked.—10%, or 30 more claims would be worked.

Conclusion.—In conclusion I am firmly convinced that if the development is kept well ahead, the stope widths are reduced to a practical minimum and the maximum waste sorted out on the surface, the resultant profits per unit of gold will be at their maxima, no matter what the costs per ton of waste and reef are, provided the maximum efficiency in working the mine is attained. Low costs without efficiency will not bring the large tonnage of low-grade Rand ores within the limits of payability."—H. MUSSON THOMAS.—*South African Mining Journal*, April 3, 1910, p. 257. (A. R.)

HAMMER DRILLS IN OVERHAND STOPING AND RAISING.—"Before the Institution of Mining and Metallurgy, Mr. H. B. Williams, Associate, read a paper from which we extract the following:—The numerous advantages obtained by the recent substitution of hammer drills in place of hand labour and ordinary piston drills having now been sufficiently demonstrated by means of exhaustive trials and practical operation in stoping and raising at the Granite Gold Mines, British Columbia, I herewith append an account of my experience of their performance. Overhand or back-stoping only is practised at these mines, the reef consisting of hard quartz from 8 in. to 5 ft. in width, containing 2% to 10% of sulphides of iron and arsenic, while the country rock is a dark close-grained granite. The vein underlies at angles varying from 35° to 50°. Figures being available to show the cost of stoping as practised in previous years at the mines, this paper is written with the object of a general comparison of results by the various methods employed, and in particular with regard to those obtained with hammer drill. The principal wearing parts are the piston hammer, costing $4, the chuck $3½, tappet $2, and complete valves $10. I now propose to deal with the advantages of the hammer drill over piston drills manifested in the tests underground, which led to the final adoption of the former at the mine for stoping and raising. The lightness and general handiness of the 2 in. hammer drill, its weight being only 60 lb against the 125 lb. of the 2 in. piston stoping drill, is a point in its favour. There is a considerable saving of time in setting up. In a shift of eight hours, it can be safely estimated that the loss of boring time in setting up, changing steel, and taking down a piston drill is two hours; with the hammer drills half-an-hour only per shift is sufficient, less time is lost in changing steel, there being no U-bolts, bars, or mountings to manipulate. Inexperienced hand-labour miners can learn to run the hammer drills in a few shifts, a great advantage in a country where good machine men are difficult to procure. Greater drilling efficiency has been obtained. With the small 2 in. one-man piston drill, a shift's work comprising a total footage bored of between 25 and 30 ft. was considered satisfactory, whereas the hammer drills bore 10 4-ft. holes, or an aggregate of 40 ft. per shift. The bits of the drill steel are not dulled by the quick, light blows of the hammer drill to the extent which occurs with the heavier blows from the piston drill. Owing to the fact that the hammer drill is so compact and requires no bar or mountings, holes may be bored to take the greatest advantage of slips in the ground in low stopes, and greater efficiency in breaking ground has resulted. The construction of the hammer drill renders its adaptability most evident for carrying considerably lower stopes than with the piston drill, a great saving of cost in stoping narrow reefs, such as those of the Granite Mines. I have been struck with the small quantity of fine dust made by the hammer drill in the stopes and rises. I account for this improvement over piston drills by the fact that there is no churning movement of the drill steel in the hole, as is the case with the piston drills, the bit of the drill steel being held continually in the bottom of the hole while boring. The air is exhausted back from the face of the stope and no air can escape at the chuck end of the machine. It may also be noticed that the hammer drills possess the advantage of producing coarser cuttings than the piston drill. In air consumption I have found that with a receiver pressure of 80 lb., there is only a slight advantage in practice in favour of the 2 in. hammer drill over the 2 in. piston drill. The advantage in general handiness, increase in progress and consequent reduction of cost in raising with the hammer drill are even more marked than in the case of stoping by its use."—*S. A. Mining Journal*, March 7, 1910, p. 299. (A. R.)

COLLIERY GAS INDICATORS.—"I. In the incandescent wire system we find that two small loops of platinum wire of equal magnitude can be made to glow by the passage of an electric current through them simultaneously. This current is generated by turning the handle of a small dynamo used in the manipulation of the system. One of the loops is surrounded by an air-tight brass cylinder, with a glass disc at one end; the other by a cylindrical copper gauze, also with a glass disc at one end. These two glass discs face each other at a few inches apart, and when the wires are made to glow the light from each falls upon the two sloping surfaces of a small movable A-shaped block covered with white paper which stands between them. When there is combustible gas in the air which circulates through the wire-gauze cylinder, the loop within that cylinder glows with greater intensity, or, in other words, emits more light than the opposite loop in consequence of the combustion of the inflammable gas in contact with it. As a result, that side of the movable block nearest to it is brighter than the other side. If the block is now moved towards the other loop until its two sides are equally bright, then its relative distance from the loops is a measure of the intensity of the light emitted by each, and by an easy process of comparison, arrived at by experiment, can also be made a measure of the proportion of combustible gas in the air. In testing a scale is provided, fixed in a line along which the apex of the block is moved, which is divided and marked to show percentages of the methane. If there is any difficulty in judging the relative brightness of the two sides of the block, the system is defective.

2.—In the alcoholic flame system we have a combination of a complete safety lamp and a small alcohol lamp. The latter may be screwed into an eccentrically situated recess in the bottom of the former, and its wick holder projects up through a tube which it exactly fills and terminates close to the wick holder of the safety lamp. The two wicks are thus brought so close together that if either of them is ignited the other immediately takes fire also. Either of these can be extinguished by drawing down or re-lighted by pushing up. It is thus possible to test for gas with alcohol flame alone, and then to re-ignite the oil flame and extinguish the alcohol flame without opening the lamp. The safety lamp itself is capable of detecting gas of 3% and over, but by the introduction of the alcohol tester fire-damp from 0·5% to 3% may be detected. About one-third of the inside of the glass is enamelled black so that a background may be formed to facilitate observation of the gas caps. In testing by means of the oil flame the wick is drawn down until it just loses its bright tip; the black glass then indicates the gas cap. If no distinct cap is seen, this does not prove the absence of gas, as it does not indicate its presence when less than 3% is present. The brass plug is then withdrawn from the oil lamp, and the alcohol tester screwed into the recess. This, by an ingenious device, opens its own wick tube, and the heat from the oil flame causes the alcohol to ascend and ignite by contact with the oil flame. The oil flame is then extinguished, and the test by alcohol

accomplished. The gas caps will be as follows, taking half-inch alcohol flame as the standard:—

Per Cent. of Gas.	Inch Gas Cap.	Remarks.
0·5	·60	Very pale, not clearly seen.
1·0	1·00	Pale blue, clearly seen.
1·5	1·44	Pale blue, clearly defined.
2·0	1·68	Clear blue cone.
2·5	2·00	Distinctly defined gas cap.
3·0	—	Oil flame used.

The alcohol flame is a simple and effective gas indicator, and can be applied very easily as a tester.

3.—In the hydrogen flame indicator we have a good practical invention. In construction it is an ordinary oil or benzoline lamp, with a copper tube passing vertically through the oil vessel, and terminating close to the wick holder, and connected to a steel cylinder fixed to the side of the lamp at the other. The steel cylinder contains compressed hydrogen, which is retained in it by means of a screw valve. When it is desired to test for smaller proportions of fire-damp than can be detected by the oil or benzoline flame, hydrogen is allowed to flow through the copper tube by slightly opening the valve. The hydrogen jet thus produced immediately ignites at the flame of the lamp. The wick of the latter is then drawn down until its flame is extinguished, and the hydrogen flame alone remains. The flame is then regulated to the proper height by means of a valve, and the heights of the caps produced by it are read off on a scale at the back of the glass. The percentage is judged as much from the brightness as from the height of the cap. The gas caps produced by a hydrogen flame (reduced) are as follows:—

Per Cent. of Gas.	Inch per Cap.	Remarks.
¼	·7	Pale and hazy.
½	·7	More definite, and outline easily perceived.
1·0	·9	More clearly seen.
2·0	1·2	—
3·0	2·1	Becoming larger and more distinct.

The incandescent system is quoted as one which is very liable to become defective or even useless during a test. In the alcohol flame system it is said when the apparatus becomes even slightly heated, by the heat of the flame or the temperature of the mine, a vapour is given off by the alcohol which is an explosive agent when mixed with air. Further it is liable to become easily extinguished if not handled with extreme care.

For the hydrogen flame it is claimed to make a test with absolute certainty. The flame is non-luminous and capable of emitting considerable heat with safety; this serves to enhance its reputation as a most favourable fire-damp indicator. It is not readily extinguished and can be taken in safety into currents which would extinguish an ordinary safety-lamp. The hydrogen flame left burning would re-ignite the oil lamp without danger in suitable current. It is capable of long duration of flame in an atmosphere which is irrespirable. Will continue to burn in atmospheres containing from 50 to 60% of carbonic acid gas, when about one-quarter of the amount would extinguish an oil or alcohol flame. It is considered an efficient and effective lamp for gas testing, but condemned by some for the foregoing reason. It is very sensitive when detecting low percentages of CO_2, exhibiting a change of colour, which is more evident as the proportion of CO_2 increases. It is capable of being taken into any mine as a fire-damp detector, and can be utilised under many other conditions. No waste of time in making a test—not necessary to exceed 30 seconds. It is not inferred that it would be prudent to place the apparatus in an atmosphere where the oil flame becomes extinguished; I only quote its capabilities."
—T. H. KNEEBONE, *Science and Art of Mining*, March 5, 1910, p. 356. (A. R.)

THE TREATMENT OF NATIVES ON MINES.—"We take the following from the report on the health conditions on Rhodesian mines:—

The members of the committee were Dr. A. M. Fleming, C.M.G., Chief Medical Officer, Rhodesia, Dr. Donald Macaulay, M.L.A. (Transvaal) and Dr. W. Gladstone Clark, M.A. (Bulawayo).

Prevalence of Scurvy.—The figures given in the last reports available, namely, for the year 1908, show that there were 1,671 cases of scurvy on the mines in Southern Rhodesia, with 207 deaths, and this may be considered a low estimate, as many cases are unreported, or died of secondary complaints such as dysentery and pneumonia, and in proof of this, during our commission we found on several mines a large number of cases of undoubted scurvy which did not figure on the hospital sheets. We have no reason, moreover, to suppose that there has been any appreciable diminution in its incidence since this report was published.

Infectiousness.—Evidence taken both in Rhodesia and in the Transvaal is unanimous as to the fact that a large proportion of the alien recruits from the tropical zones arrive either with the disease upon them or develop it within the first three months of service. Outbreaks occur from time to time amongst native miners who have been resident for longer periods than this, but careful enquiry will generally bring to light some error in the dietary existing on the particular mine concerned.

Deferred Pay.—It would be convenient here to refer to the system of deferred pay, as this would appear to have a certain bearing upon the incidence of scurvy. It is a striking fact that habitual labourers and those natives who add to their diet by purchases of foodstuffs from local stores rarely suffer from scurvy, whilst those natives whose pay is reduced by remittances to their homes or who are of a parsimonious nature, are particularly affected.

Diet.—We would especially emphasise the necessity for the issue of fresh vegetables, further reference to the growing of which will be found in clause 65. As far as possible, vegetables for which the natives have a predilection should be given, and fortunately these are the most easily grown. We would specially mention potatoes, pumpkins—in fact, all gourds—sweet potatoes, wild spinach; also leeks and onions, where obtainable, are valuable anti-scorbutics. The difficulty of obtaining fresh vegetables for the mines has been impressed upon us, and we therefore recommend that at seasons of the year, and in districts where fresh vegetables are unobtainable, ½ lb. of meat and a double ration of Kafir beer be added to the diet scale as a substitute.

Kafir Beer.—The nutritive value of Kafir beer and its action as an anti-scorbutic have been amply demonstrated; we consider that it should be a compulsory ration on all mines. The preparation should be in the hands of the mining authorities, and it should be issued with regularity and under supervision. In this connection we found on visiting the Rand that an excellent beverage called "mahauwe" was prepared in some of the compounds from the residue of the mealie porridge. This is a mildly fermented beverage, is easily prepared, and has a

distinct nutritive value. A description of how this should be prepared will be found in Annexure "C."

Native Diet in Kraals.—It was alleged by some witnesses that the diet of mine natives compared more than favourably with the diet the native was accustomed to in his kraal. We have enquired into the native kraal diet, and though a greater quantity of food may be consumed by the native on the mine, still, in his kraal his food is richer in those fresh constituents to the presence of which we attach so much importance. Apart from this consideration, it seems superfluous to insist that the scale of feeding obtaining in the native kraal is no criterion as to a native's wants whilst engaged in mining.

Prevalence of Pneumonia.—During the year 1908 there were reported to the Government Health Department no fewer than 2,251 cases of pneumonia, with 686 deaths, giving an incidence of 72·93 per 1,000, and a death rate of 22·23 per 1,000 labourers. Our investigations have shown that these figures are an under-estimate.

Overcrowding as a Factor.—Overcrowding in compounds produces ideal conditions for the spread of an admittedly infectious disease. This overcrowding is brought about in two ways; firstly, by the provision of inadequate accommodation and insufficient air space, as we found on some mines; secondly, by the habits of the natives, who prefer to crowd into huts with their brothers rather than occupy the places set apart for them. There is more pneumonia during the winter months, when labour is most plentiful and overcrowding most prevalent.

Structural Defects of Dwellings.—The effects of overcrowding are aggravated by the structural nature of the huts and the personal habits of the native. We consider that the ordinary hut full of the heterogeneous collection of odds and ends with which the native surrounds himself, badly ventilated and badly lighted, with an absorbent floor which he uses as a spittoon, and which he impregnates with nutrient material from the refuse of his food, forms an ideal incubator for the germs of an infections disease.

"*Stricter and More Intelligent Compound Inspection.*"—We recommend that, where possible, there should be a daily inspection of all natives in the compound, and an inspection by a medical officer or other competent person once a week. The sanitary condition of many of the mines we visited is open to improvement. Clothing parades and disinfection of clothing, and the prohibition of natives changing their clothes with natives, should be enforced.

General Conclusions.—Food should be abundant and properly cooked. The fine grinding of meal is of the utmost importance. The cultivation of mine gardens for the supply of vegetables should be encouraged. Natives who have their wives on the mines seldom suffer from sickness, and tend to become habitual labourers. The office of compound inspectors should be removed from the Mines Department and placed under the Health Department. Such officers should be given powers of summary jurisdiction in cases of minor offences under the regulations."—*S. A. Mining Journal*, May 7, 1910, p. 286. (A. R.)

HYDRAULIC IMPREGNATION OF THE COAL FACE AND HYDRAULIC BLASTING IN COAL SEAMS BY THE MEISNER PROCESS.—In 1890 Meissner devised a process for rendering coal dust harmless, by forcing water under a pressure of 10 to 20 atmospheres into a borehole about 40 in. in depth for a considerable time. The mouth of the borehole was fitted with a stuffing gland. The water impregnated the coal face in such a manner as to enable the coal to be brought down free from dust. The process, however, suffered from certain deficiencies and after a time was given up.

On account of the Rodbad explosion last year experiments were again started at a colliery near Dortmund, and later were extended to a number of other collieries in Westphalia. Deeper boreholes were used than in 1890. The packing gland was also improved and extended further into the borehole. In dusty bituminous seams the experiments have been quite successful, but in compact non-porous structures the method has been found impracticable.

When the coal is porous and mild, a pressure of 20 atmospheres are found sufficient, but in more compact seams 40 atmospheres are often necess ry. Again, in some seams the impregnation and loosening of the coal are complete in about 10 minutes whereas in others 5 to 6 hours are necessary.

As impregnation is proceeding the coal can be heard to tear, break and even detonate. The process is attended with great difficulty in seams where the adjacent rock is cleaty and friable, since under these circumstances the rock can be easily wetted and even begins to creep, necessitating great care. The main results of the process are greater safety in shot firing, reduction of consumption of explosives and absence of dust during loading and haulage.—BERGASSESSOR TRIPE, Dorstfeld—*The Colliery Guardian*, July 8, 1910, p. 67. (T. D.)

Reviews and New Books.

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

ORE MINING METHODS. By WALTER R. CRANE. Cloth, octavo, pp. 225. Ill. (New York: John Wiley & Sons. London: Chapman & Hall.) Price 12s. 6d.

"This book contains descriptions of methods of timbering and supporting rock in place, methods of stoping and mining in narrow and wide veins and bedded and massive deposits, including stull and square-set mining, filling and caving, open-cut work, and a discussion of costs."—*Mining Magazine*, July, 1910. (A. R.)

GEOLOGY: SHORTER COURSE. By Profs. CHAMBERLIN and SALISBURY. 21s. (London: John Murray.)

"The scientific and literary partnership of Profs. Chamberlin and Salisbury, the heads of the Department of Geology in the Chicago University, is a felicitous one, and has been productive of excellent results upon geological teaching in America. Their advanced course not only achieved American celebrity but speedily attained to considerable vogue in this country; indeed, we can hardly remember another foreign work of the same type which sprang so instantaneously into prominence and received so frank and cordial a welcome. The shorter course has been prepared in response to an obvious demand, and is also excellent, the thankless work of abridgement having been carried out with judgment and with skill. We have, in fact, nothing but praise for the book, which is adequate in its presentment of principles, informed with a multitude of illustrations drawn from the labours of the best equipped geological survey in the world, and written in a manner which administers a useful stimulus to the imagina-

tion, without in the least trangressing the restraint proper to scientific writing. The stratigraphical record as here presented is, naturally, almost exclusively American in its inspiration, and to English students, accustomed to the English classification, the descriptive names will jar with the unfamiliarity of a foreign language. But such differences are inevitable to the science, and form its chief, perhaps its only, drawback. Moreover, they tend to destroy provincialism of thought, and are thus of independent utility. A point which constantly recurs in this work, which is of great importance, is the presence or absence of deep-sea deposits in the sedimentary record. Our authors hold—and M. Emil Haug in his recently published traité partially concurs with his view—that, so far as at present determined, the derivative rocks exhibit no examples of formations parallel with those now accumulating upon the ocean floor. Even the true white chalk, which occurs in the Secondary rocks of the United States, though not in such great vertical extension as with us, the authors attribute to conditions of comparatively shallow but clear water, and not to the true ocean depth. This view, which is gaining constantly greater currency among geologists of the modern school — that the gigantic changes of sea-level formerly postulated never occurred — is obviously fundamental, and carries with it the conclusion that the relative positions of sea and land have varied in geological time only within comparatively narrow limits."—*London Mining Journal*, June 11, 1910. (A. R.)

MEASUREMENT OF PULP AND TAILING. By W. J. SHARWOOD. Pamphlet, pp. 24. Ill. (London: *The Mining Magazine.*) Price 2s.

"This is a reprint of Mr. Sharwood's Articles that appeared in *The Mining Magazine* for November, December and January. The information contained in the articles has proved to be of great value to metallurgists and this issue in separate form has been made to supply a demand."—*Mining Magazine*, July, 1910. (A. R.)

NATAL: REPORT ON THE MINES AND MINERAL RESOURCES (OTHER THAN COAL). By F. H. HATCH, Ph.D., M.I.C.E. Published by order of the Natal Government. (London: Natal Government Agency.)

"As the result of a personal examination the public would naturally hope for some definite expression of opinion from a trained and experienced expert as to the probable result, favourable or unfavourable, of further development on those deposits which have been partly opened up. It is remarkable in a report of this kind that as regards the majority of the properties described there is not the faintest indication of opinion as to what might be the outcome of further development. Only in instances where developments have proved conclusively the worthlessness of a property is there any definite opinion expressed.

Plans and sections of development work in progress were in the majority of instances non-existent; and this deficiency, due to the existing regulations of the mining department, is pointed out in the report, but the writer of it should have supplied the deficiency so that his readers might obtain a clear insight of the structure of the ground b ing explored. Such insight can very rarely be obtained from verbal descriptions. They leave too much to the imagination of the reader. Facts conveyed by correct plans and sections are definite in themselves and in their relation to one another.

From the descriptions of the iron ore beds and of several of the non-metallic minerals it is evident there are indications of deposits on which further investigation might lead to the establishment of profitable industries, but, as is properly pointed out in the report, their exploitation is hampered by the lack of any geological maps. Without reliable surveys no systematic records, either geological or statistical, can be maintained. A proper survey of the Colony would be of the greatest possible benefit to individual enterprise.

To those intimately associated with Natal the inconclusive nature of this report must be disappointing; but it should in no wise act as a deterrent on their efforts to develop the resources of the Colony, and it is to be hoped that the Government will, by a proper organisation of the department concerned, render more efficient assistance to prospectors and miners than they have done in the past."—*London Mining Journal*, July 9, 1910. (A. R.)

REDUCTION OF WORKING COSTS ON THE RAND. By HENRY HOWARD. 1s. (London: *Investors' Guardian*, Limited.)

"This is dated May, 1910, and is an examination of existing conditions and the prospects. The author states that the work, which has appeared in the form of contributions to the *Investors' Guardian*, is the result of information gained during his last visit to the Rand in the autumn of 1909. The brochure sums up very well what has been done under the big mills low grade low costs policy of the last two or three years. From 1902 to the autumn of 1909 the average cost of all the Witwatersrand mines was reduced from 25s. 9d. to 17s. 2d. per ton. Authorities are cited to support the expectation of a further reduction, but it may be well to remember that during the last few months the leaders of the industry have been giving intimations of changes in policy, which would involve higher rather than lower costs."—*London Mining Journal*, June 11, 1910. (A. R.)

FROM PROSPECT TO MINE. By ETIENNE A. RITTER. 8s. 4d. (Denver, U.S.A.: The Mining Science Publishing Company.)

"Chapter I. treats on Prospectors and outfits, and these subjects are particularly well dealt with. Chapter II. is on Veins and Ore-bodies; Chapter III. on 'Taking up Claims and the Apex Law." This chapter, based on the laws of the United States, graphically describes all the procedure necessary to be executed and work accomplished before a claim can be said to be properly registered and owned by the prospector or taken up, and this chapter also defines the text of the patent grants of a lode mining claim.

Chapter IV. treats on "Starting work underground," and contains many practical suggestions as to the best course of procedure under different conditions. The author rightly says that a serious and a frequent mistake is not to have the work directed in the first instance by a good mining engineer, who need not stay at the mine all the time but inspects it regularly and keeps posted on the process of the work; and we also endorse the advisability of having assays taken regularly when a vein is being developed. There is also a great deal of truth in the author's view put out—that the future of the mining industry belongs to some powerful prospecting and mining companies able to stand the losses incurred in failing to open a mine from a large amount of prospects,

and yet able to keep on until they get a bonanza which will not only make up for all the money previously lost, but bring besides a tremendous profit; in such a company, as in an insurance company, the working of the law of averages will bring success.

Chapter V., on "Mining Companies," furnishes a few useful practical notes of advice, and also words of caution for both the prospector and also the investor as to the flotation of metalliferous mining companies, and the methods adopted for the getting up and the issuing of prospectuses with a view to the raising of share capital.

Chapter VI. deals with Pointers as to Local Conditions, and Chapter VIII. with Mining Machinery.

Chapter X.—the last one in the book, on "Philosophy and Poetry of Mining"—should be read with the warning note with which this review opens."—*London Mining Journal*, July 9, 1910. (A. R.)

HINTS ON SURVEYING; AND THE KEEPING OF MINE PLANS AND REPORTS. By J. W. TEALE, M.I.M.M. 6s. (London: *The Mining Magazine.*)

"A thin octavo volume containing 18 pp. of subject matter and 21 of specimen forms for monthly returns, etc. The reading matter consists of a few desultory remarks on mine plans in general; the hints on surveying, which occupy the major portion, comprise a list of conventional signs usually employed by mining surveyors. As regards the forms proposed for the returns, they may probably serve their purpose; but these are matters of individual taste, and most managers have some hobby of their own. The book is very neatly bound."—*London Mining Journal*, July 9, 1910. (A. R.)

Abstracts of Patent Applications.

(C.) 245/10. James Dawson & Son, Ltd. (1), James Dawson, junior (2). Improvements in conveyor belts.

The invention for which the above application is made relates to improvements in the construction of conveyor belts.

According to this invention the completed belt is constructed of a belt, the full width of the completed belt, to either or both sides of which are attached, longitudinally, two or any greater number of belts of smaller widths. The smaller widths may be attached to the full width by means of rivets, screws, or sewings, or by means of balata or like substances.

All the widths are constructed of textile fabric, and each may be of any number of plies suitable for the purpose for which the belt is required. The textile fabric is usually impregnated or spread with balata, gutta percha, india-rubber or like substance.

The intention of the invention is to secure greater longitudinal pliability than is possible in a belt of the same strength made in one width, and to increase the life of the belt as if one of the smaller widths wears out it can be easily replaced.

(C.) 325/10. Ottokor Serpek. Improvements in electrical resistance furnaces. 8.7.10

This invention relates to the manufacture of nitrides, especially aluminium nitride.

A furnace is described which produces the nitrides from mixtures of suitable materials through the agency of the heat produced by an electric current in resistances constructed of carbon and aluminium nitride. These resistances are so shaped as to stir up the melted ingredients when the furnace is revolved, thereby securing an easy and regular distribution of the heat generated.

(C.) 354/10. Charles Anderson Case. Improvements in a panning and concentrating machine. 22.7.10.

This application refers to an improved panning and concentrating machine consisting of a screen and a series of pans, mounted co-axially with openings near the lowest points of the conical pans and a means of applying a circularly swaying motion. Balls are used to aid the separation of the concentrate and to force it through the openings provided.

(C.) 367/10 and 368/10. William Woodward Robacher. Improvements in or relating to filtering apparatus particularly intended for the cyanide process of extracting precious metals from their ores. 29.7.10.

These applications refer to vacuum filters of the vertical rotary type, the lower half of which is immersed in slime pulp, and provided with scrapers acting on the upper part for removal of the partially dried solid. 367 refers to a filter with the filtering medium forming the walls of the rotating member, and 368 to one with the filtering medium forming the periphery. The principle feature of the design is the use in both of a rigid, porous mineral material as the filtering medium.

(C.) 576/09. David Gilmour. Improvements in fixing cam pulleys to cam shafts. 13.12.09.

This refers to a means of fastening pulleys on cam-shafts, and consists of a tapered bore in the pulley boss to fit on a taper turned on the shaft. The grip of the pulley is obtained by screwing up a nut on a thread cut on the cam-shaft whilst a key may also be fitted if desired.

Selected Transvaal Patent Applications.

RELATING TO CHEMISTRY, METALLURGY AND MINING.

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

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

(C.) 410/10. William Arthur Beard. Improvements in and relating to rotary engines and the like. 19.8.10.

(C.) 411/10. William Edge. Improvements in hinging machines. 19.8.10.

(P.) 412/10. Willie Herbert Stewart. Drill heating and smelting furnaces. 19.8.10.

(C.) 413/10. Elizabeth Barnston Parnell. Improvements in the treatment of ores. 19.8.10.

(C.) 414/10. John Gillies (1), Claude Daniel McPhee (2). Improvements in scratch headed fuses. 20.8.10.

(P.) 416/10. Philip Lawrence Lazarus. An improved sanitary pit. 22.8.10.

(P.) 417/10. John Broad Roberts. Anti-phthisical lubricator and appliance. 23.8.10.

(P.) 418/10. Adam Richard Stacpoole (1), Joseph Mahoney (2). Improvements in and appertaining to the shoes and dies of stamp mills. 24.8.10.

(P.) 419/10. Norman William Pill. Improvements in apparatus for the manufacture of inflammable gas by carburetting air. 24.8.10.

(C.) 421/10. William Edward Harding (1), John Hunter (2). An improved chuck for rock drills and other purposes. 25.8.10.

(P.) 423/10. William Charles Stephens. Improvements in or connected with cradles for rock drills and other machines. 25.8.10.

(P.) 424/10. Hans Charles Behr. Improvements in pneumatic percussive apparatus. 26 8.10.

(C.) 425/10. William David Coolidge. Improvements in tungsten and the manufacture thereof. 26.8.10.

(C.) 426/10. Emil Deister. Improvements in ore concentrators. 26.8.10.

(P.) 428/10. Cecil Hardwood Fenn (1), David Gavine Hunter (2), John Hathorne Wilson (3). Improvements in rope jockeys. 27.8.10.

(P.) 429/10. Reitz Gannon. Improved process to bring about the complete combination and convert cyanide mine dump sands into hard substances such as concrete waterproof plaster and the like. 29.8.10.

(C.) 430/10. Holberry Mensforth. Improvements in internal combustion engines. 29.8 10.

(P.) 431/10. James Auld. Improvements in driving means for the ore feeders of stamp mills. 2.9.10.

(C.) 432/10. Robert Emmott. Improvements in crushing, pulverising and disintegrating machines. 2.9.10.

(P.) 433/10. William Henry Rufus Munnery. An improved double action stone crusher. 3.9.10.

(P.) 434/10. Arthur Martin Ludwig Dammrich (1), Charles Hansen (2). Improvements in means for conveying and distributing water or other liquids in mines and the like. 3.9.10.

(P.) 435/10. Arthur Martin Ludwig Dammrich (1), Charles Hansen (2). Improvements in means for conveying and distributing water or other liquids in mines and the like. 3.9.10.

(P.) 436/10. Albert Edward Jordan. Improvements relating to hauling and like engines. 3.9.10.

(P.) 437/10. Frank George Scineke. Improvements in means for straightening or altering the direction or course of a bore hole. 3.9.10.

(C.) 439/10. Boris Hellmann. Improvements relating to the treatment of petroleum for the purpose of modifying certain characteristics thereof. 5.9.10.

(P.) 441/10. Anthony Richardson. Vacuum brake improvements. 7.9.10.

(C.) 442/10. Robert James Worth (1), Worth, McKenzie & Co. (2). Improvements in connection with the valve controlling gear of winding engines for collieries or the like purposes. 7.9.10.

(C.) 444/10. William Spiers Simpson (1), Howard Oviatt (2). Improvements in the direct production of iron and steel from the ore. 9.9.10.

(C.) 445/10. Charles Cheers Wakefield. Improvements in or relating to automatic regulators for acetylene gas generators. 9.9.10.

(C.) 446/10. Robert McNitt. Improvements in methods for reducing metals. 9.9.10.

(C.) 447/10. John Collins Clancy. Improvements in the treatment of precious metalliferous ores. 9.9.10.

(C.) 448/10. Michael Joseph Owens. Improvements relating to the production of articles of glass and to apparatus therefor. 9.9.10.

(C.) 449/10. William Emil Bock. Improvements relating to the production of articles of glass and to apparatus therefor. 9.9.10.

(C.) 450/10. Arthur Thomson. An improved means for collecting the dust from holes drilled by rock drills and like machines. 9.9.10.

(C.) 451/10. Thomas Rouse (1), Besseler, Waechter & Co., Ltd. (2). Improved preparation of soluble alkaline silicates containing ferric oxide. 10.9.10.

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ANDERSON, J. S., *l/o* England; Abosso G. M. Co., Ltd., Abosso, Gold Coast Colony, W. Africa.

ANDERSON, P. von N., *l/o* Benoni; P. O. Box 678, Johannesburg.

ATKINSON, C. E., *l/o* Selukwe; East Gwanda Mines Ltd., West Nicholson, Rhodesia.

BEATTY, J. W. S., *to* P. O. Box 1064, Johannesburg.

BROWN, J., *l/o* Fordsburg; Village Deep, Ltd., P. O. Box 1064, Johannesburg.

CHOMLEY, W. B., *l/o* Day Dawn; Lancefield G. M Co., Beira, W. Australia.

CLARENCE, J. V. G., *to* P. O. Box 27, Jeppes.

COCKBURN, P. A., *l/o* Queque, Rhodesia; Le Hang Chang Mine, via Francistown, B.P.

GRAHAM, H. W., *l/o* Johannesburg; c/o Bank of Africa, 113, Cannon Street, London, E.C.

GRAY, J. WENSLEY, *l/o* New Zealand; Wallis Co., Ltd., Ntranang Obuase, via Sekondi, Gold Coast Colony.

JENKINS, C. W. BOWEN, *l/o* North Sydney; Yambulla Gold Mine, Yambulla, near Eden, New South Wales.

JOHNSTON, JAMES, *l/o* San Francisco; Room 1002, 30, Church Street, New York.

JORY, J. V., *to* P. O. Box 124, East Rand.

JUPP, H. L., *to* 31a South Street, Vogelfontein, P. O. Boksburg.

HAY, E., *l/o* Johannesburg; Ginsberg G. M. Co., Ltd., P. O. Box 16, Knights.

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OLDS, H. F., *l/o* England; Lone Star Mine, Gadzema, Rhodesia.

OLIVER, E. L., *to* 9, First Street, San Francisco, California, U.S.A.

PAYNE, A. E., *l/o* New Primrose G. M. Co.; Consolidated Langlaagte Mines, Ltd., P. O. Box 15, Langlaagte.

POWTER, H. B., *to* P. O. Box 5, Knights.

ROBERTSON, J. R. H., *l/o* Penhalonga; East Rand Proprietary Mines, P. O. Box 220, East Rand.

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WERTHEIM, C. J. M., *to* Kaswarie Straat, 3, The Hague, Holland.

WHITEHOUSE, J., *l/o* Utrecht; Village Deep, Ltd., P. O. Box 1064, Johannesburg.

WRIGHT, A. L., *l/o* Florida; P. O. Box 4804, Johannesburg.