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The Society, as a body, is not responsible for the statements and opinions advanced in any of its publications.

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Proceedings
AT
Ordinary General Meeting, September 19, 1908.

The Ordinary General Meeting of the Society was held in the Chamber of Mines, on Saturday, September 19th, Mr. R. G. Bevington (President), in the chair. There were also present :—

39 Members: Messrs. A. McA. Johnston, E. H. Croghan, W. R. Dowling, K. L. Graham, C. B. Kingston, A. Richardson, C. B. Saner, H. A. White, Prof. J. Yates, A. Avent, H. D. Bell, E. Browning, J. S. Cellier, F. W. Cindel, A. A. Coaton, G. A. Darling, L. Evans, G. Goodwin, J. H. Harris, J. I'Ons, T. Johnson, J. A. Jones, G. A. Lawson, Hy. Lea, J. Lea, S. Morison, P. Morrisby, W. D. Morton, E. Pam, C. F. Parry, J. F. Pyles, W. H. Roe, C. E. Rusden, R. Stokes, W. Taylor, J. F. Walker, J. P. Ward, and J. Watson.

10 Associates: Messrs. J. Atkinson, J. Cronin, W. J. Dunnachie, N. Newland, W. S. V. Price, H. Rusden, W. A. C. Tayler, W. E. Thorpe, C. Toombs and W. Waters.

9 Visitors and Fred. Rowland, Secretary.

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

NEW MEMBERS.

Messrs. W. E. Thorpe and W. Taylor were elected scrutineers, and after their scrutiny of the ballot papers, the President announced that the candidates for membership had been duly elected, as follows :—

BENNETT, RANDOLPH WHITE, Witwatersrand G. M. Co., Ltd., P. O. Box 1, Knights. Cyanider.
BLEWETT, EDWARD, c/o Wilkins & Co., Chemists, Selukwe, Rhodesia. Chemist and Cyanide Manager.
CARR, W., Salisbury G. M. Co., Ltd., P. O. Box 1022, Johannesburg. Mine Manager.
CREED, JOHN PERCY, Village Main Reef G. M. Co., Ltd., 12, Eloff Street, Johannesburg. Cyanider.
EDWARDS, A. L., P. O. Box 74, Barberton. Assayer and Cyanider.
KERR, THOS., Jumpers G. M. Co., Ltd., Cleveland. General Manager.
STRANG, JOHN, East Rand Proprietary Mines, Ltd., P. O. Box 66, East Rand. Cyanider.

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

ALLAN, JOHN RICHARD, Turf Mines, Ltd., Schuller Street, Forest Hill, Johannesburg. Miner.
LINDBERG, BIRGER, Lebong Soelit, Ketaun, West Coast, Sumatra, Dutch East Indies. Mining Engineer.
SMITH, SYDNEY A., Turf Mines, Ltd., P. O. Box 5887, Johannesburg. Miner.
WOLVEKAMP, HENDRIK, Lebong Soelit, Ketaun, West Coast Sumatra, Dutch East Indies. Mining Engineer.

And the following Student :

LIPSCHITZ, CECIL, Transvaal University College, P. O. Box 1176, Johannesburg.

GENERAL BUSINESS.

The President: It is very pleasing to see that we have so many gentlemen coming forward as Members and Associates. It is very satisfactory that we are able to keep up our list of members, and replace any resignations. In fact, we are not only able to do this, but we are increasing our membership.

MINES TRIALS COMMITTEE.

The President: There is one correction I wish to make with regard to what I said on the evening of my inaugural address respecting the Mines Trials Committee. I then stated that I understood the Mines Trials Committee interested itself in everything connected with mining. I have lately been informed that it does not do so, but that its scope practically embraces everything from the collar of the shaft to the residue dumps and slimes dams. I am very sorry to find that the Committee does not interest itself in appliances connected with winning the rock, that is to say, rock drills and other similar appliances, as well as the reduction works plant. I certainly think that the Committee should have laid itself open to deal with everything connected with

mining, both on the surface and underground. I must apologise to any gentlemen who have been misled, and I know there are some who have already placed matters connected with rock drills before this Committee only to be informed that it could not deal with them. I can only say that I was misinformed, and that I regret very much if these gentlemen have been put to any inconvenience.

NOTE ON A PROBLEM DURING SHAFT SINKING.

By Chas. B. Saner, M.I.M.M. (Member of Council).

Shaft sinking on the Rand has on the whole been more or less plain sailing, and there are few recorded cases of how great difficulties met with have been surmounted.

On a certain mine on the West Rand very soft stuff was cut through with great difficulty, but unfortunately there have been no published details as to how the job was accomplished: on the East Rand water in one or two instances was a great hindrance, but no paper has been written showing the method of overcoming this bugbear. This note is a short account of how a difficult and dangerous stage in vertical shaft sinking was passed through in the shaft of the Turf Mines, Ltd., without loss of life and without undue loss of time, and will, I hope, encourage others to give the Society the benefit of their experiences in other such cases.

Shaft sinking and development is to my mind one of the most important and least acknowledged branches of mining; because on the excellence or badness of the work depends the future economical working of the mine to a great extent. If the main thoroughfare is bad and poorly laid, the running costs must be high; be it rail-road or shaft-road, one is horizontal, the other is vertical, but both are for the same purpose. The efficiency or otherwise of the shaft will affect the maintenance costs throughout the life of the mine.

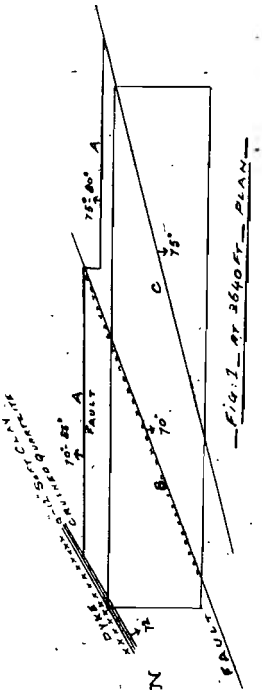
The Turf Mines shaft is being sunk by hand labour of three 8-hour shifts, 75 to 85 boys are employed in lashing and drilling, 35 to 45 holes of 30 in. to 40 in. in depth being blasted per shift. Hand labour is much speedier and less expensive than any other method at present utilised.

The shaft is a seven-compartment one with its long axis running N. and S., that is across the formation, the north or pump compartment being No. 1, and the south compartment being No. 7, the bearers being similarly numbered.

The reason for sinking the "deeper deep" vertical shafts of the Rand with the long axis across the formation is for greater strength; no long ledges of rock are exposed on the strike, as is often the case when the long axis is parallel with the strike of the formation, the excavated area measuring in this instance 48 ft. N. and S. and 8 ft. E. and W. that is only 8 ft. of formation dips into the shaft, and as stage winding is to be adopted in these shafts, it is not necessary to consider the question of the bend.

For some time the shaft in being sunk was cutting through series of almost vertical faults with approximately S.E. and N.W. strike and dip S.W., cutting diagonally across the shaft.

On referring to Fig. 1 (plan of shaft at 2,640 ft.), it will be seen how the country rock around the shaft was cut up.



Fault "A" was dipping steeply east 75° to 85°, with very distinct striation marks, and was cut through by Fault "B," which had about ½ in., of dyke matter on its line of cleavage. "C" was a fault plane crossing the strike of the formation in a diagonal direction.

The formation, although normally with a dip of about 34° S., was shattered and cut up badly around the dyke, faults and cleavage planes being thrown out in all directions and at all angles. The dyke on first being cut, was soft and shaley, with quartz seams running through; on the line of contact was 6 in. to 12 in. of

pulverised clayey matter adjoining 12 in. to 36 in. of crushed and altered quartzite.

At 2,630 ft. a fault striking N. and S. dipping east appeared on the whole east side of the shaft. The rock was continually scaling off from this fault, and as the shaft was breaking wider and wider and a dyke face with a dip of 72° S.W. had appeared in the S.E. corner, it was decided to put in a set of bearers at 2,662 ft., hitches of 12 in. to 48 in. on the east side and 6 in. to 12 in. on the west side, and bearers 9 in. wide under the end pieces, and 7 in. wide under each divider by 27 in. to 36 in. deep, which were all firmly wedged in place.

On referring to Fig. 2, the reason for the great difference in length of hitches will be appreciated; the N. and S. hitches were 12 in. deep, being cut out of good solid ground, while the others were in "baulky" ground and so cut deeper for greater hold and security.

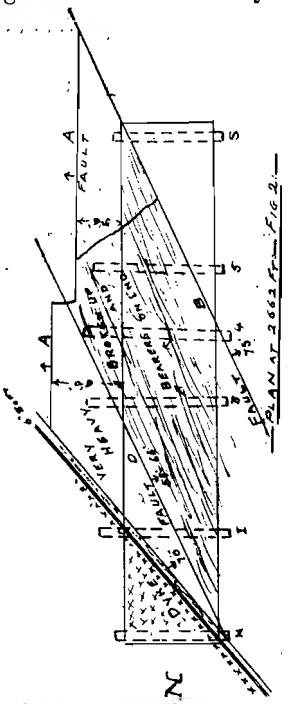


Fig. 2 gives a clear idea of the disturbances and the positions of the bearers. These bearers were wedged tightly from the west side, so as to resist any tendency of the east side to heave off to Fault "A" into the shaft.

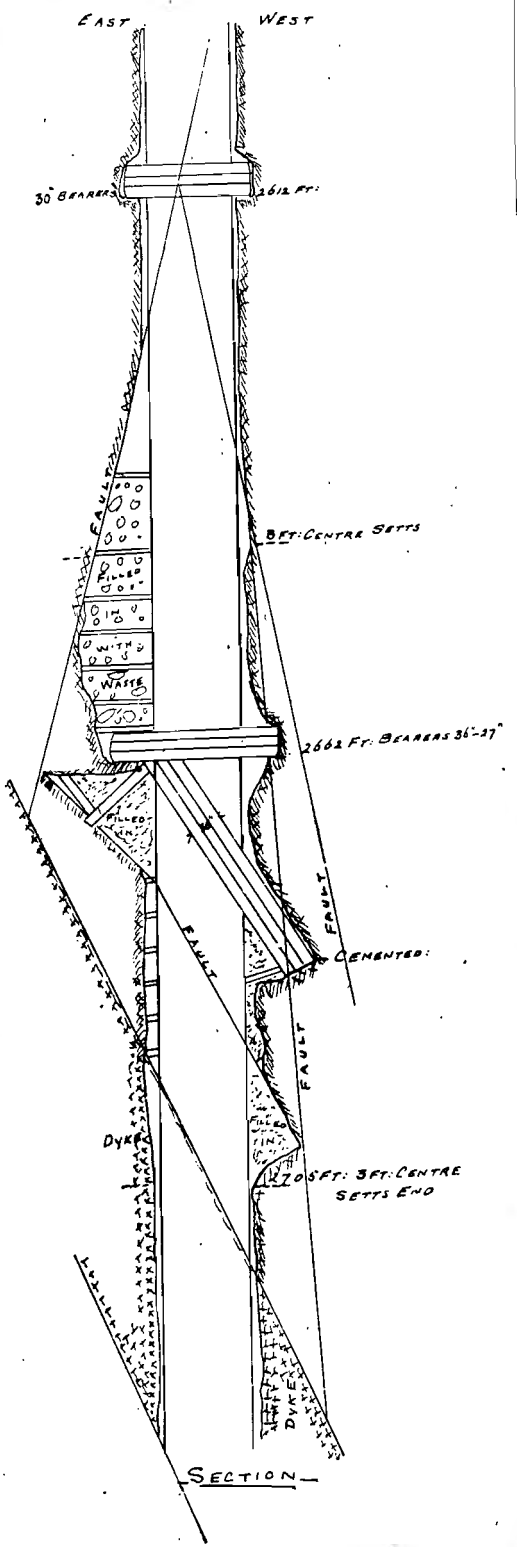
On the 2,662 ft. bearers a solid "bottom" was placed; that is, cross timbers from one bearer to another were firmly packed in, and all intervening spaces were closely filled with blocks, thus making a compact floor between sets and shaft side; then the sets were closely lagged with 3 in. to 4 in. Australian hardwood (stringy bark) planks and filled in behind with waste rock from the shaft bottom.

The method of filling in was as follows: The skip was loaded at the bottom in the usual way, and by signal slowly raised to the required position, where a white man was stationed. The clamping arms preventing the skip from tipping were then disengaged, a hook attached to a wall plate by a chain on the opposite side of the shaft to where the waste was to be tipped, was placed in a bracket on the bottom side of the skip, and then the signal to "lower slowly" given; the chain and hook took the weight and thus overturned the skip; the lip of the skip landing on a wall plate and the contents falling behind the lagging between fault and timber, thus making a solid side, and without a fragment of rock falling upon the 75 to 85 boys working below.

Previous to the above hitches being cut, it was decided to shorten two sets, which had been hung at 6 ft. centres to 3 ft. centres to catch the "heaving" ground more securely. The hanging bolts in the top set were first loosened, then the studdles were removed and brought to surface in the skips. Chains were lashed round the east and west wall plates of the lower set in Nos. 2 and 6 compartments and attached to the north skip of one engine and the south skip of the other engine by means of shackles below the skips; by signals given simultaneously to the driver of each engine, the whole was raised together; that is wall plates, end pieces and dividers in one lift; when raised to the required position, the long hanging bolts were taken out and the short ones put in and the set hung. The same was done with the lower set. The great secret was to get the two engines to work exactly together; a ticklish job and accomplished successfully and without mishap; this saved a great deal of time, as it obviated raising all the pieces of timber separately to surface and sending them down again.

As soon as all was secure and safe, sinking was resumed; but we had only got a few feet below our 2,662 ft. (or No. 24 set) of bearers, when the greater part of the east side below the bearers slid away into the shaft, fortunately immediately after the 3 p.m. blast, along the Fault "D," undercutting bearers No. 3, 4, 5. (See vertical section.) On careful examination the east side appeared so heavy and dangerous, that all sinking was again stopped; and to make assurance doubly sure, it was decided to come up the shaft above the region of these disturbing faults and put in another heavy set of bearers.

So at 2,612 ft. this was done. A complete set of eight bearers 30 in. deep in good solid hitches was put in; we usually place bearers under each



end piece and every alternate divider that is five in all, and about 100 ft. to 120 ft. apart. Moiling and cutting hitches in solid quartzitic rock is a slow, expensive, and expert job, especially with timber, which must not be damaged, close around, and there is always great difficulty in getting natives to moil out the hitches, because of the confined space and awkward positions; the method of cutting hitches is as follows: Having lined out the exact position carefully from the dividers above, diagonal or stripping holes are drilled about 6 in. to 12 in. deep, these are shaken with one-half to one stick of gelatine; then the hitch is squared by drilling holes 6 in. to 9 in. deep, as close together as possible, and the intervening rock cut out with chisels or popped with explosives, and finally the bottom is moiled or chipped square; a "lead" is cut away at one end to bring in the bearer timber; this lead may be either vertical or diagonal, it is usually the latter.

A "lead" is vertical when there are no dividers close to the hitch, so that the bearer piece can be dropped into the hitch. But usually diagonal so as to get the bearer in under the divider, and when once in the hitch, no side blow can dislodge the bearer. A "lead" is a recessed way cut out of the solid rock to give room for a bearer to be brought into a hitch.

A "bearer" is a foundation girder and should always be several inches longer than the shaft is wide; the object of the bearer is to support the superincumbent weight and transfer this weight through the hitches on to the solid rock. A "hitch" is a step cut out of the solid rock to receive the end of a bearer.

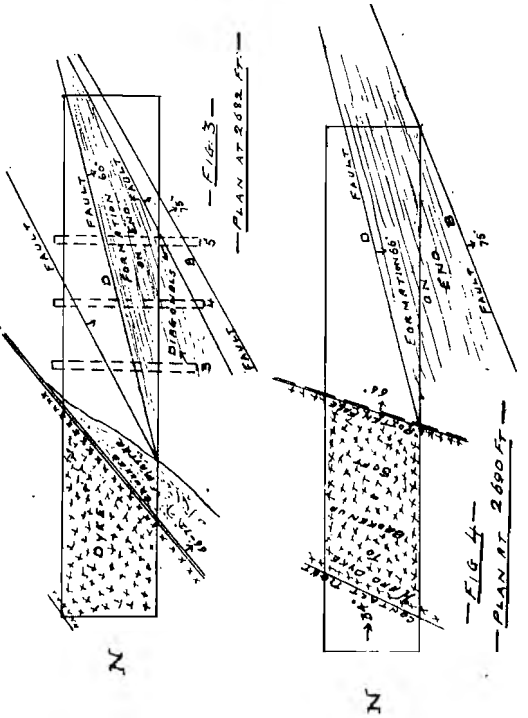
The great thing is to get the bearer piece as good a fit as possible in the hitch, so that it should have no chance of being pulled out, in case of skip accident; a bearer piece should never be able to give way, but must rather break before being able to be dislodged from its hitches. Finally, the bearers 30 in. deep made up of three 10. in. pieces of pitch pine, were firmly placed in solid hitches under every divider and with the hanging bolts $1\frac{1}{8}$ in. diameter, and 12 in each set in place and each set well blocked, sinking was resumed, but with great caution and very lightly charged holes.

The undercut, eastern side, below the 2,662 ft. bearers was temporarily supported by three legs 14 in. by 16 in. on a 12 in. by 14 in. sill 15 ft. long, well spragged. At 2,682 ft. three deep hitches 4. ft. to 5 ft. were cut in the west side of the shaft and diagonal bearers were firmly placed with their heads under Nos. 3, 4, 5 bearers at 2,662 ft. and the toes against the fault on the west side and cemented in. These three diagonal bearers were two 36 in. deep and

one 24 in. deep, consisting of 12 in. pieces of pitch pine, a great thickness of timber, and may be too strong, but it is better to be sound in time than to be sorry when too late.

These diagonals took a long time to get into position, as a neat fit was essential, and they had to clear the steel guides. They are at an angle of 57° upwards to the east, so supporting the heavy eastern wall of the shaft as well as the ends of the horizontal bearers; when these diagonals were in place the temporary legs were removed.

The ground between Faults D and B (see Figs. 3—4) was all on end and greatly disturbed



and broken up, and as sinking was continued the trouble was transferred to the west side of the shaft, but fortunately only the southern half.

The sets at 3 ft. centres were continued to 2,705 ft., and, where possible, close lagged with 3 in. stringy bark and extra blocking behind every divider, firmly wedged in place and afterwards filled in with waste and ashes.

The timber was kept within 6 ft. of the bottom during this sinking, hence very light holes and careful blasting was necessary. The method of procedure was as follows: The north and south ends of the shaft were cut down with light holes so as to get as far away from the hitches on the west side as possible before proper blasting was done; then as the west side was so disturbed, a channel was cut along the east side and the ground stoped away from the west side, with light holes looking east.

By this means the shaft was slowly sunk until regular formation through the dyke was reached. No one unless he has actually worked with them can imagine the difficulty of handling heavy and large timbers in such confined spaces; when the sets were being kept close to the bottom, it was a difficult and slow job to hang the wall plates, each half plate being about 23 ft. long. They were dropped vertically to the bottom, suspended by a wire rope in pairs below the skip, and the ends dragged north and south till the plates were horizontal on the bottom, they were then detached from the skip and hauled up by block and tackle horizontally into position and suspended on the hanging bolts.

I am a firm believer in good solid hitches and well placed bearers, for on the soundness of these the whole shaft depends, and I deprecate the practice I have sometimes seen adopted of scamping the hitches, and only wedging the so-called bearers in place. Good bearers and solid hitches are a firm foundation for the shaft.

I may mention that before the bearers were wedged in place, four heavy plumbobs were suspended from the surface to ensure that the shaft timbers were in their correct vertical position before final blocking. Two bobs were lowered in No. 2 compartment, 10 ft. 6 in. from north corners on the east and west sides and 6 in. from the wall plates. Two other bobs were lowered in No. 6 compartment, a similar distance from the wall plates. Saw cuts had been placed in the set to be plumbed at 2,662 ft.

The lines were lowered from surface from reels, light weights being attached for this purpose, then the heavy bobs were put on, and while coming to rest the line was examined carefully from bottom to surface to make sure all was clear, the skip ascending slowly. After an hour, to allow the bobs to settle, careful measurements were taken and the set was found to be $\frac{1}{16}$ in. too much to the north and $\frac{1}{4}$ in. too much to the east.

The work now completed is a safe and good one and reflects great credit upon Mr. Coombe and the timbermen for the efficient and speedy manner in which it was done. No man shirked his job, though for several days and nights double shifts were worked, and the danger of working in a confined space below unknown masses of heavy ground was great, and if it had started to move, all would have been crushed like rats—that is the luck of the miner!

The President: Mr. Saner is very much to be congratulated on the lucid way in which he has placed this matter before us. This is one of

those problems in mining which are very difficult to contend with and very interesting when the work is safely accomplished. Mr. Saner, and also Mr. Coombe, are to be congratulated on having safely performed a hazardous operation, and I trust they will have no further trouble in that quarter.

ZINC DUST PRECIPITATION AT CERRO PRIETO.

By Robert Linton (Associate).

The Cerro Prieto plant of the Black Mountain Mining Company, situated near Magdalena, Sonora, Mexico, is the only plant in Mexico operating on a large scale, and one of the few plants on the Continent, which is successfully using zinc dust for precipitation. The following account, which is confined to the matter of precipitation alone, and is not intended to cover any other features of the cyanide practice at that plant, may therefore be of interest to the members of the Society.

The ore is crushed in weak potassium cyanide solution in a 120-stamp mill, and the coarse sands reground in tube mills; the pulp is treated in a sands-slimes plant, the sands by percolation, and the slimes by decantation and filtering in vacuum filters. Weak solutions are used, 0·15% to 0·2% for leaching the sands, 0·075% for crushing and for slimes treatment. The gold dissolves rapidly, over 60% of the total contents being extracted in the mill. The gold solutions carry generally 2 to 3 dwt. gold and about four times as much silver; they contain ordinarily from 0·07% to 0·1% KCy and are of about 0·2% alkalinity.

The solutions drained from the sands and decanted from the slimes flow to the gold sump. A clarifying filter is placed in the top of the tank to remove any slimes in suspension. The filter is of the ordinary type used for filter bottoms, a wooden grating covered with cocoa matting and burlap, on which is spread 2 in. of coarse gravel and 6 in. of sand. The occasional cleaning required is quickly and easily done by scraping a thin layer of sand from the top. As the filter is in the top of the tank, this can be done without emptying the tank, and only requiring the flow of solution to be shut off for a few minutes. The solution filters through by gravity, is drawn off as convenient and pumped to the precipitation tank on the hill above the plant. The latter has a storage capacity of 450 tons, and has in the bottom a clarifying sand filter similar to that just described.

The gold solution is drawn off from the bottom of the precipitation tank through a 6 in. pipe connected to the zinc press. Zinc dust in the form of an emulsion is added at the point where the solution leaves the tank, the emulsion being prepared as follows:—A small quantity of solution is lifted by compressed air through a 1-in. pipe and discharged into a cone at the top of the tank. Zinc dust is fed into this cone at regular intervals and thoroughly mixed with the solution by agitation with compressed air introduced through a ¾ in. pipe reaching nearly to the bottom of the cone. The emulsion overflows into a 3-in. pipe leading into the 6 in. outlet pipe, is added to the outflowing gold solution, and becomes thoroughly mixed with it in flowing down to the zinc press. This apparatus maintains very satisfactorily the essential regularity of zinc dust feed.

The zinc press is of the Dehne type, 35 in. square on outside of frames, having 24 leaves with 4 in. frames intervening, the total filtering surface being 367 sq. ft. Cotton sheeting 72 in. wide, made by the Pepperell Mills, Bradford, Maine, is used for filter cloths. Two cloths are used over the leaves. Each time the press is cleaned new cloths are put on next to the leaves, the inner cloths put on over them, and the outer cloths burned. The press stands 31 ft. below the bottom of the precipitation tank, affording a head of from 31 to 47 ft., depending upon the height of solution in the tank. No pump is used in connection with the press, the flow being maintained by gravity alone. The pressure corresponding to head would vary with the height of solution from 13·4 to 20·4 lb. The working pressure of the press, or the back pressure of the solution flowing through it, rises from zero, when the press is started, to a maximum of 14 lb. before the break.

After a break of the press, precipitation is started by feeding an excess of zinc dust into the feed cone, then opening the valve to the press, allowing the mixture to flow through slowly, in order to form a thin film of zinc dust on every filter cloth. The high rate of zinc dust feed is kept up for six hours, the quantity of solution fed being meanwhile gradually increased. At the end of six hours, the rate of feeding zinc dust is cut down and gradually lowered until the press is broken again. Before starting the press the gold solution is always titrated for KCy, and when necessary enough fresh cyanide added to insure the immediate setting up of precipitation. After precipitation is once started, comparatively weak solutions (as low as 0·07% KCy) are precipitated without difficulty. The precipitation is very rapid and complete. As much as 1,200 tons of solution has been passed through the press in 24 hours, with an average value left in the effluent solution of less than 1 gr. gold and a mere trace of silver. This means that the solution is flowing

down through the pipe line at a rate of 135 ft. per minute, that the entire volume of the press is being displaced every three minutes, and that the total time that the gold solution is in contact with the zinc dust is less than five minutes. The rate of flow corresponds to 3·2 tons of solution per square foot of filter surface per 24 hours.

Lead acetate is added in suitable proportion to the zinc dust, which assists the precipitation. The silver contents of the solution doubtless facilitate as well the rapidity of the precipitation.

With accumulation of precipitate in the press the rate of flow through it gradually decreases with a corresponding increase of pressure to a minimum of about 700 tons per 24 hours, the average being usually about 900 tons per 24 hours for the entire time from starting the press until it is started again, including the time required for cleaning. This tonnage, it will be understood, is handled from the one precipitation tank and through the one zinc press.

The consumption of zinc dust is 0·4 lb. per oz. of gold and silver precipitated. Calculating this on the actual metallic zinc available in the zinc dust, the consumption is 5½ parts of zinc to one of gold and silver. The press is broken and cleaned four to six times per month. The precipitate is dried, briquetted with borax and litharge, and melted and refined in an English cupelling furnace.

Without going into an extended discussion of the respective merits of zinc dust and zinc shavings for precipitation, a few of the advantages and disadvantages of each may be noted, together with a few of the essential points to be noted in using the former.

Success in zinc dust precipitation requires careful supervision and a thorough working out of questions of feed of zinc to the solution, and of zinc and solution to the press. The aim is to obtain a granular product which, when it forms a coating on the cloths or settles to the bottom of the frames, shall offer the least obstruction to the flow of solution through it. If the product is very fine, or contains slime carried in from unclarified solutions, the coating on the cloths soon becomes dense and impervious to the solution flowing through the press, and the capacity of the press consequently diminished.

Gravity feed appears to offer an advantage over the use of a pump to force the solution through the press. The tendency of a pump is to break up the product and produce the fineness that it is aimed to avoid, while when the zinc and solution flow quietly to the press, as is the case in using the gravity feed, the condition is the most favourable for forming the granular product, and the fullest capacity of the press obtained. To the same end a thorough clarification of the solution prior to precipitation is necessary so as to avoid any clogging of the filter through slime settling in it.

The refining of the zinc dust precipitate involves more labour and expense than the refining of the zinc shavings product. The latter is usually melted by the cyanide foreman, whereas melting the zinc dust product practically means the addition of a smelting department, on a small scale it is true, to the operations. More flux is used, more fuel, and more labour is required. The additional cost has proved a serious factor in many cases, and has even in some cases led to the abandonment of zinc dust precipitation altogether. On the other hand, there is the saving of all the trouble and expense connected with the zinc boxes; and if the refining is done efficiently and the precipitation effected with a minimum of zinc dust, the cost of refining may be practically balanced in the saving made by having no zinc boxes to look after. To this must be added a further advantage, and the most important one derived from zinc dust precipitation, the increased security against theft.

Zinc dust used for precipitation should be regularly tested for total contents of metallic zinc, zinc oxide, and lead. It is frequently very impure, and unless the operation is regularly controlled by such tests there is likelihood of incomplete precipitation on the one hand or excessive consumption of zinc on the other. Too much zinc is not only a waste in itself, but materially increases the refining cost. The limited supply available makes it often difficult to obtain a satisfactory quality, and has been a hindrance to its more general use. Any considerable increase in the consumption of zinc dust could easily create a demand which the present productive capacity could not meet, but the eventual result would be the manufacture of zinc dust as a separate commodity instead of an undesirable bye-product of zinc smelting, and the making available of a more reliable supply, both as to quantity and quality, than the market now affords.

The President: We have to thank Mr. Linton very much for a most interesting paper, and I am very pleased to see that we are getting contributions from such distant parts as Mexico. There is no doubt that in these times Mexico is waking up with regard to metallurgical processes, as you will have noted from the able paper presented to the Society by Mr. Caldecott on the treatment of silver ores in Mexico. From all that we hear and from the correspondence between Mr. Caldecott and various gentlemen in Mexico, it is evident that a very great interest is being taken in metallurgical work there, and I feel

gratified that we have this paper from Mr. Linton, which I hope will be the forerunner of many more from that part of the world.

DISCUSSION.

Mr. Walford R. Dowling (*Member of Council*): I have read the author's paper with considerable interest, and only regret that he did not go into the subject more fully. Although the paper only deals with precipitation, he mentions the cyanide strengths used for dissolving as 0·15% to 0·2% for sands and 0·075% for crushing and slime treatment. I would like to ask whether these strengths are necessary for dissolving the gold and silver, or are they maintained so high for the purpose of precipitation? I like his method of bringing about the mixture and contact of solution and zinc dust, but am not clear as to how the zinc dust is fed into the mixing cone. Is some kind of mechanical feeder employed, or is this done by hand?

With reference to the sheeting used for filter cloths, apparently the outer sheeting is burnt at each opening of the press. At that rate, with four openings per month four complete sets of sheeting are burnt per month. In this country the item would amount to something in cost.

I can appreciate the point of aiming at a granular product to ease filter pressing, but do not see any mention of how this result is obtained other than gravity feed of solution. I have been trying to find out for a number of years what are the conditions which give an easy pressing of acid treated gold slimes, without much success. Gravity feed is probably better than pump feed, but in any case, if the slime is granular, the pressing is easy. My difficulty is how to keep the slimes always granular.

With reference to the notes on smelting, the author does not say why it is necessary to treat the zinc dust precipitate differently to zinc shaving precipitate, and why it involves greater expense. I understand that zinc dust in America is not more costly than zinc shavings, and where about the same quantity of either per ton is required the case improves for zinc dust. In South Africa zinc dust is still more expensive than zinc shavings, and would have to become much cheaper before the method could be considered.

I am afraid my contribution to the discussion consists mostly of questions, but I hope the author will be able to enlighten us on the points raised.

Mr. McArthur Johnston (*Vice-President*): I should like to put two questions before the Society to-night, so that Mr. Linton may have the opportunity of replying as early as possible. He does not definitely state here whether or not amalgamation takes place in the mill. This point might, I think, be made a little more clear. Then he says the silver contained in the solution doubtless facilitated the rapidity of the precipitation. Can Mr. Linton give us any figures showing the relation existing between the amounts of gold and silver precipitated by the zinc, and to what extent the quantity of silver present facilitates precipitation of the gold?

Mr. H. A. White (*Member of Council*): I notice that no mention of acid treatment of the zinc gold slimes is made and would suggest that this should considerably reduce the trouble and expense of the clean-ups.

The President: I think that point is an important one. We will transmit these queries to Mr. Linton, and in due course get his reply.

Mr. J. A. Jones (*Member*): I would like to know how the screen assay is obtained there. It will be interesting to us on the Rand to know how they get it in other parts of the world. Also how theoretical extractions and actual recoveries compare, and are the copper plates, if any, attacked very much by cyanide?

The President: Mr. A. Richardson has kindly undertaken to read the following paper:

SMALL MINES OF RHODESIA.

By B. I. Collings (*Member*).

In bringing a few notes on this subject before the Society, it is to be understood that no very new or very perfect methods of mining or metallurgy will be dealt with, but rather an attempt will be made to describe the nature of the ore deposits, the economic and the industrial peculiarities of the country, and their effect on the methods of mining and milling generally adopted. The subject may best be dealt with under four separate headings:—

1 geological, 2 economic, 3 mining, 4 metallurgical.

Geological.—The gold mines of the country are, as far as I am aware, all located in a geological horizon corresponding to that known as the Swaziland system in the Transvaal. This system is recognised as the oldest series of rocks in South Africa, and it is in it that the gold mines of the Barberton, Klein Letaba, and Murchison districts of the Transvaal are situated. In Rhodesia it is represented by schists, banded ironstones, quartzites, conglomerates, and occasional slates; the conglomerate beds however, seldom bear any similarity to the banket beds of the Rand.

Most of the mines in Rhodesia work quartz reefs, which may be of any colour or appearance, but gold is occasionally met with in other matrices; in conglomerate as at the Eldorado and the Riverlea, in schist as at the Giant, and in altered granulite as at the Ayrshire; in fact, any rock in the gold belts of Rhodesia, if mineralised, is worth testing for gold. There is very seldom any definite formation in which one may more particularly expect to find gold, but it appears as a rough general rule that the payable mines are to be found either in schist near the contact of one or other of the big granite masses of the country, or close to some minor intrusion of an acid igneous rock. These granite masses, which are probably of similar age to the old grey granite of the Transvaal, are intrusive into the rocks of the Swaziland system, and though they are, generally speaking, very poorly mineralised themselves, it appears highly probable that they played an important part in the genesis of the ore bodies found in the metamorphic rocks. Contacts of all kinds are promising places to prospect, and these may often be recognised, even where there is a heavy overburden, by a change in the character of the bush, trees growing more readily on some soils than on others. There has been practically no prospecting for virgin reefs, nearly all the work having been confined to opening up ancient workings, of which there must be many thousands in the country.

The reefs are more or less lenticular and vary in width from a few inches up to 20 ft. or more. The distribution of the gold in them seems to follow no rule whatever, and base metals are usually present, pyrites being the most common, but galena, chalcopyrite, mispickel, and stibnite, are occasionally met with, although seldom more than two of these minerals are present in the same ore. A curious feature to be observed in some of the quartz reefs carrying gold at surface, is the disappearance of all values at depths of 30 or 40 ft., and, in others, the disappearance of the gold at water level; these reefs cannot all be regarded as gash veins, as they often continue in depth although barren. It is possible that the gold in them may have been precipitated by organic agencies, which would be effective only close to the surface. They often show a thin film of visible gold in the cracks of the quartz, which is very deceptive, as it pans disappointingly; it is known locally as paint gold.

It is not to be understood that a large proportion of the reefs lose their gold within 30 or 40 ft. of the surface, but it greatly increases the value of a prospect when it has been proved below water level. It must be admitted, however, that even where the gold holds below water level, a falling off in values with depth is by no means an uncommon phenomenon, and some mines that have been highly payable at surface, have been comparatively poor at depths of 700 or 800 ft.: but it is impossible to predict whether in any particular case an impoverishment may take place, as many of the older mines show as good values as ever in depth, while most of the small workers' mines are not yet deep enough to enable one to express an opinion.

It is idle in the vast majority of cases to search for the extension of a reef for any great distance.

Few mines in the country possess a pay shoot of greater length than 600 ft. or 700 ft., and in most instances the values only hold over much shorter strikes than this, a pay shoot of 200 or 300 ft. being considered quite a sound proposition for the small man. Payable mines are quite frequently found in groups within a mile or two of one another, but even in these cases the individual mines often differ greatly in the character of reef and mode of occurrence of gold, both free milling and refractory ores being encountered in the same belt of country.

In prospecting for virgin reefs, it must always be borne in mind that the ancients have been over the ground previously, and it is unreasonable to hope that any conspicuous outcrop carrying free gold will have been left untouched. Virgin reefs of value are very difficult to find, and many of those discovered have only been struck by accident. In examining an ancient working on which no present-day work has been done, the absence of any quartz on the dump is not a bad sign, but rather the reverse, as it often means that the reef was all of it sufficiently rich for the ancients to take away and crush. Ancient workings, however big, are not necessarily a sign of high gold values, and it seems probable that even in those far-off days there were "wild-cats."

The quartz reefs now being worked are seldom of any great width, a stoping width of 3 or 4 ft. being usual. The width in some mines is fairly consistent throughout the pay shoot, but more often the ore occurs in a series of lenses. When the ore body consists of ironstone, schist, or conglomerate, the widths are often very great, but the ore is of correspondingly low grade. Very few small workers have as yet attempted to tackle those big impregnations, and where they have been worked, only the richer portions have been exploited. There is little doubt, however, that the ultimate future of the country, from a mining point of view, rests with these big low grade ore bodies, and it is probable that as the small workers extend their operations, this type of mine will be more largely worked. In the gold belts the country is full of reefs, of

which probably not more than one in fifty could be pronounced payable, some being too narrow, and others too poor. At the same time, a series of unpayable reefs will sometimes through weathering and disintegration, particularly where rich stringers are present, give rise to a payable rubble area. Although seldom of high grade, these rubble deposits are very cheaply worked, and are of great value when taken in conjunction with reef mining proper, as a stop-gap when development is behindhand, a state of things which, I regret to say, is the rule rather than the exception in the small mines. These rubble deposits often extend over many acres, but are usually payable only in patches; the depth varies from 6 in. up to 6 ft. or more. A certain amount of material classed as rubble consists of the sorting heaps of the ancients. The sampling of rubble is a very laborious operation; the area should be well trenched and big samples taken. Many expensive mistakes have been made in the estimation of rubble, and great caution is recommended before tackling a proposition of this sort.

To those accustomed to the comparatively even values of the Main Reef series of the Rand, an example or two will be given, showing the necessity of taking as little as possible for granted in quartz mining.

Case I.—Two large ancient workings were situated, in a line of smaller ancient workings, about 200 ft. apart from one another. Shafts were sunk in each of these to a depth of 70 to 80 ft., and excellent values were obtained. A 200 ft. shoot of good ore was assumed. On driving to connect the two shafts the values fell off at once, and further examination showed that the two shafts had been sunk in chimneys of ore caused by the intersections of two somewhat insignificant looking stringers.

Case II.—A shaft was sunk on a line of ancient workings, and, having bottomed the old workings, was continued for a further depth of 40 ft. on the reef. It was then decided to drive in order to determine the length of the payable shoot before striking the water. The drive in one direction showed abnormally high results, many ounces to the ton, and it was assumed that there were 40 ft. of backs of this grade ore. On starting to prepare to stope, the old workings were broken into within 18 in., and the assumed backs were found to be almost totally non-existent. It is true that neither of these cases should have proved a trap for a careful man, but there is a strong temptation for the small man, to whom money is scarce and time is valuable, to rush at conclusions and take chances that under other circumstances he would not feel inclined to do. At the same time, if a man is to do any good in a quartz mining country he must take risks. The man who takes chances and makes the fewest mistakes does best, and the man who waits till he is absolutely sure before acquiring any interest, will wait a very long time and often acquire nothing.

Economic Conditions.—The rate of wages paid in Rhodesia to whites has been higher than that obtaining in the Transvaal, and although the larger mines are now making a general move to lower these rates, it is probable they will still continue to be above those paid on the Rand. Miners and mechanics have received 25s. a shift in the past, and although the company owned mines have recently cut these rates, the small workers will probably continue to pay the old rates, as the hours worked are so much longer, and Sunday work is pretty frequent.

Native labour is cheaper than on the Rand, but probably not so efficient, while there is invariably a great shortage during the summer months. Surface labour is paid from 10s. to to 25s. a month, while underground boys receive from 15s. to 50s. a month, but rates vary considerably in different districts. Police boys, boss boys, and other special boys, receive 80s. a month or more. It is customary to start a boy at the lower wage, and gradually raise him according to his abilities. Boys are fed and housed by the mines as in the Transvaal. Many of the boys employed are recruited by the Rhodesian Native Labour Bureau, a semi-official concern similar in character to the Witwatersrand Native Labour Association, but the native labour position in Rhodesia is by no means a satisfactory one. Most of the boys supplied by the Bureau come from North-Eastern and North-Western Rhodesia, while most of the Southern Rhodesia boys come voluntarily to work. The high railway rates make machinery and stores expensive, and competition amongst the merchants is not carried to the same extremes as in the Transvaal.

Practically all the small mines use wood fuel; some of the bigger mines which have been working for many years, having cut all the wood in their vicinity, find it cheaper to use coal. The only producing coal mine is Wankies, situated on the Bulawayo-Victoria Falls Railway some 200 miles from the nearest gold mine. There are other well-known coal deposits much nearer the gold belts, but the Chartered Company is not in favour of their being opened up, as it is interested in the Wankies mines. In some parts of the country no charge is levied on the mines for cutting wood, but there is considerable difference of opinion between the farming and mining communities as to both wood and water rights. The position is rather complicated, as farms are held under two or three different forms of title, but in

Mashonaland, in almost every case, the miner has full wood and water rights without payment. Water is obtained either from rivers or from the mine, dams being very seldom seen.

In the Penhalonga district near Umtali, everything is driven by water power, but this is quite the exception. With regard to transport, most of the small workers ride their own cord wood and mining timber, which is cut in the vicinity of the mine, using oxen or donkeys for the purpose. At the stations and principal sidings transport is undertaken by forwarding agents and storekeepers. During the rainy season the transport of heavy machinery to any great distance is rendered almost impossible, as the roads are merely tracks cut through the bush, with frequent vleis where the wagons sink up to the axles in mud.

The climate is not unhealthy, and women and children are to be seen in every district. Good health is largely a matter of diet, and fresh meat and vegetables are much more easily obtained than they were ten years ago. One never hears of men compelled to leave the country on account of fever. Lions very seldom give trouble, but transport riders occasionally lose donkeys or oxen through them. White ants are troublesome, but there are various preparations, such as carbolineum, solignum, etc., which keep them away. Tar is also effective in preserving wood from white ants.

In the Rhodesian law, provided the outcrop of the reef is pegged, the claim holder can follow the reef on the dip as far as he pleases. There is therefore no such thing in Rhodesia as a deep level claim. Practically the whole country is open for pegging, and the holders of the surface rights have no claim at all on the minerals. In order to be able to peg ground, it is necessary to obtain a prospecting licence. Each prospecting licence costs £1, and enables the owner to peg out a block of 10 claims, measuring in all 1,500 by 600 ft. An individual can obtain as many prospecting licences as he pleases, on paying for them. The first step in pegging ground, after discovering one's reef, is to post a discovery notice. This protects the ground for a radius of 1,000 ft. for 30 days. Before the expiration of the 30 days, however, it is necessary to peg out the actual area required, and then apply at the nearest mining office for registration, giving a sketch and describing the position of the ground. This registration costs a further £1 for a block of 10 claims. Within six months of posting the discovery notice, it is necessary to do 30 ft. of development, or in default, obtain an inspection certificate at the cost of £5. During the next six months a further 30 ft. of work must be done, or in default an inspection certificate at the cost of £15 must be obtained. After the first year 60 ft. of work must be done annually, and inspection certificates are only granted on work done and cannot be obtained by payment. There are no claim licences to be paid as in the Transvaal. In the case of a number of contiguous blocks registered in the same name, the work can all be concentrated on one block. Many of the claims pegged years ago and held by companies are held under more easy terms than the above; inspection certificates being obtainable by payment, and work done on one block of claims is allowed to protect other blocks, whether contiguous or not. Blocks of claims containing ancient workings on which no work has been done, can be bought at from £50 to £250 or more, but ancient workings are by no means a guarantee of payability. Where properties are worked on a share or tribute basis, it is almost invariably arranged on a percentage royalty on the gold won. This is the only satisfactory way, and the one that least lends itself to any possibilities of dispute. The usual royalty asked is in the neighbourhood of 10%.

As to what constitutes payability, this depends very greatly on reef conditions, locality, etc., and these vary very widely. A 2 dwt. recovery on rubble may under favourable conditions be payable, while a 10-dwt. recovery on a reef where conditions are unfavourable may barely pay expenses. Under average economic conditions, a 10-stamp mill with a cyanide plant working a 3 ft. reef should work at from 15s. to 20s. per ton milled.

The capital required to start a small mine depends largely on the class of machinery put up, but I will give a rough estimate of the cost of three types of plant very often seen. Tremain steam stamp outfit with Dodge crusher, boiler, and pump, etc., price f.o.r. Salisbury, £720. Capacity, 300 tons per month. Five-stamp Californian mill, 1,050 lb. falling weight, with engine, crusher, boiler, pump, etc., price f.o.r. Salisbury, £1,450. Capacity, 750 tons per month. Four foot Chilian mill, with engine, boiler, crusher, pump, etc., price f.o.r. Salisbury, £1,400. Capacity, 800 tons per month. These figures are very rough, and the capacity depends naturally on the kind of ore crushed. A galvanised iron cyanide sands plant to treat 600 tons a month will cost erected between £400 and £500. To erect a 5-stamp mill and cyanide plant and develop a mine to keep it adequately supplied, cannot be done for less than £3,000, and this sum is usually considerably exceeded. Huts, tools, camp equipment, rails, piping, livestock for transport, and similar items, all run away with money, but fortunately credit is more easily obtained than in the Transvaal.

Mining Conditions.—These are, as far as the small mines are concerned, exceedingly simple. Shafts sunk on the reef are the rule. Sets made of round native timber poles are used, the favourite wood for this being "mopani," an exceedingly hard wood and difficult to work, but it is one of the very few that resist the attacks of borers. Most native woods become quite rotten within a year or so from these pests. The shafts of the small mines have generally only one hauling way, a single drum hoist with a ½ ton skip being sufficient to keep the usual small mine going. Native timber props are used in the stopes, but owing to shortage of development too few pillars are often left. Headgears are made of native timber or sometimes of 9 in. by 3 in. deals, and are seldom over 25 ft. high.

In most of the mines the reefs dip pretty steeply, and it is not often one strikes a quartz reef dipping at less than 50°. As a rule, the gold is free, and the values are followed by the pan, an assay plant on a small mine being very unusual. Drilling is all done by hand, compressed air being beyond the means of the small worker, and in most cases they are working in oxidised or semi-oxidised ore, where hand-drilling is in every way more economical. The character of the hanging wall varies in every mine, while the distance apart for levels depends more upon the fancy and resources of the man working the mine than on strictly mining principles; it is seldom more than 100 ft. Gelignite is the explosive generally used, the price of which is 42s. 9d. a case at the nearest station. Gelatine is 58s. 9d.

Nothing is known concerning the ancient workings, but it appears probable that some of them are over 200 years old. The method of mining was by fire and water, and charred pieces of wood are occasionally found in the old stopes.

Metallurgical Conditions.—Under this heading I will deal with surface plant generally. The special requirements of the small worker are portability and cheapness in first cost and erection. First of all I may mention that sorting is rarely indulged in by the small worker, as where the reef is narrow the casing for a few inches on both sides usually carries values.

One of the striking features of the surface plants on the small mines is the number of types of crushing machinery employed. Tremain steam stamps, Huntingdon mills, Chilian mills, pneumatic stamps, and last, but not least, Californian stamps, are all to be met with. The advantages of the Tremain mill are portability and low first cost, its chief disadvantage being heavy steam consumption. Huntington mills are low in first cost and use little power, and in crushing soft clayey ores their output is large and cost of spares small; for hard ores they are very inefficient and expensive. Chilian mills are higher in first cost than Huntingtons, but for soft or moderately hard ores, treating large tonnages, they are very satisfactory. Both the Huntington and Chilian mills, when treating the harder ores, are used to best advantage as secondary crushing machines. The pneumatic mill, it is claimed, costs less and has less weight, for a given capacity, than the Californian mill, and is probably well adapted for very hard ores. Two, three, and five stamp mills of the Californian type are frequently seen, and the old gravity mill is under many conditions very difficult to beat, particularly when subject to the rough usage machinery receives in the hands of some of the small workers, who are too apt to call in the aid of a 7-lb. hammer whenever things go wrong.

Frenier sand pumps are largely used for lifting tailings, and for heights up to 18 or 20 ft., they are admirable, requiring little power or attention. The Frenier spiral sand pump consists of a spiral ribbon of steel plate in form like a spiral clock spring. On each side is a steel disc, which is joined to the spiral by continuous air-tight joints, thus making a spiral tube of steel with a rectangular section of constant area throughout. It is mounted on a hollow horizontal shaft, which has an opening to connect with the spiral tube. There are no valves, but the water and sand are raised by virtue of a hydrostatic head in each turn of the spiral, a part of each turn being filled with water and the rest with air (the pump being partly immersed in water and partly in air). The pump works best at about 20 revolutions per minute.

The cyanide tanks on the small mines are invariably made of galvanised iron, and are generally placed directly on the ground, and are filled and emptied from the sides. For this reason it is better to have the units small, and 15 ft. is the largest diameter tank that can be conveniently worked in this manner. The cost of a 15 ft. by 4 ft. G.I. sands tank complete with filter frames and matting is about £40 to £45. The battery pulp is usually first run into a settling tank fitted with a slimes gate, and the sand thus collected is transferred to the treatment tanks; but in some cases it is run into a series of pits, where it settles and is afterwards dug out. Most small workers have very little idea what their percentage recovery is, as many cyanide plants are run without any assays ever being taken, but below the oxidised zone it is probably well below rather than above 80%, for very few small mines have slimes plants, as their first cost is so heavy. Blanket strakes, or canvas tables, are only occasionally employed, and their

use combined with some small amalgamating pan might probably be extended with advantage.

Duplex steam pumps are generally used for cyanide solutions, as it often happens that there is no suitable line shaft from which to run a centrifugal pump. The usual type of boiler used is the locomotive, one of its advantages being that it requires no brick-work setting. Slide valve engines are almost invariably used, and where fuel is expensive, feedheaters are installed. Gas producer plants are spoken of very favourably, and will probably be largely used in the future. Electricity is seldom used, but might often be advantageously employed for working isolated pumping stations; most small workers, however, understand very little about it, and are therefore afraid of installing it. Water is a very important question on many mines, it being necessary to pump sometimes three or four miles. One often sees the mistake made of putting the mill by the water instead of carrying the water to the mill, but in many instances a three or four mile pipe line is altogether beyond the resources of the small man. When debating on the introduction of labour-saving appliances, local conditions must be considered, and in Rhodesia where labour is cheap and machinery expensive, and where moreover the life of a mine is such an uncertain factor, labour-saving appliances are not justified to the same extent as in most other fields. With limited resources, the deciding question is often, what *can* be done, rather than what *ought* to be done.

In Rhodesia one must regard almost every individual mine as a law unto itself; character of ore, character of walls, character of gold, all vary; in no single way does it appear possible to generalise on them. For instance, I will take a mine with which I am personally connected. It is called the Cam and is situated in the Hartley district. The ore consists of about 30% quartz and 70% chlorite schist, the walls being of schist. The recovery over the plates is at present less than 20%, while the slime charges are more than half as high again in value as the sand charges. The ore contains stibnite and mispickel, and shows at depth no gold in the pan, the reef width is about 6 or 7 ft., the footwall is defined, but the values gradually merge into the hanging. The stibnite occurs in patches and carries high gold values, which have up to the present proved irrecoverable. The bullion contains practically no silver and has been as high as 991 fine. At one place in the mine a cross-reef occurs and values are obtained over a width of 30 ft. or 40 ft. Within little over a mile of us is the Eiffel Blue mine, which works a quartz reef in what I am told is diorite. Here 80% of the gold is caught over the plates, and the slimes are not rich enough to treat. The ore contains no stibnite, while the bullion contains an appreciable amount of silver.

The above notes will, I hope, enable members of the Society to form some idea of the conditions affecting the small worker in Rhodesia.

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The President: The thanks of our Society are due to Mr. Collings for placing this paper before us. It is very interesting indeed and should call forth a good deal of discussion; Mr. Collings points out in one place that the ultimate future of the country rests on the big low grade ore bodies. I do not think that that only refers to Rhodesia. Reduced working costs and improved facilities in working generally, will allow us to develop our own large bodies of low grade ore and that will very materially affect the life of the mines of this country. I should like to make one remark on the subject of mining papers. It has been stated elsewhere that there is a great paucity of mining papers in Johannesburg. I cannot agree with the gentleman who made that statement. Here are we to-night with two such papers before us, and we have had several others recently, one of which is under discussion now. I refer to the paper by Mr. Tom Johnson who, I am sure, will give us a very able reply to-night. This contribution has given great food for discussion and reflection. I must say that I cannot agree that there is with us a paucity of mining papers, though it may be that in another place there is.

NEW TYPE OF NATIVE COMPOUND BUILDING OF ALL METALLIC CONSTRUCTION.

(*Read at April Meeting, 1908.*)

By C. B. Kingston, B.A., B.A.Sc. (Member of Council).

REPLY TO DISCUSSION.

Mr. C. B. Kingston: Replying to Mr. Schmitt's discussion of my short paper. The question of the merit of this type of building as a habitation for natives may perhaps be a

* See this *Journal*, vol. viii., Sept., 1907, pp. 90-91.

matter of opinion, but as stated in my paper, the design received special commendation from a well-known authority on native housing. His opinion has been supported by our experience. The health of our boys has been extremely good since they moved into the new compound.

During the past winter, the rooms, which are heated with ordinary stoves, have been quite warm and comfortable. They will, doubtless, be warm on hot summer days, but so will any other building with a single roof of corrugated iron. The iron building has this advantage, that its walls do not radiate heat during the night as brick walls do, but cool quickly, after sundown, and remain cool during the hours when the boys are resting.

On the plans submitted, the windows are stated to be equivalent to 10% of the floor space.

Sloping windows, set high in the roof, receive the sun's rays more directly than vertical windows, and transmit the light directly into the heart of the room. This point received special consideration, and after discussion with the health authorities, it was decided that windows set as these are set would produce an excessive illumination if made with an actual area equal to 10% of the floor space.

Iron of 24 gauge was not used without first testing the protective coating after bending to the required curve, and although well aware that heavier iron is commonly used for curved roofs, I am satisfied that the 24 gauge iron is heavy enough for the building in question, while the riveted joints appear to be perfectly satisfactory.

Care is taken to prevent the boys using open fire boxes, so that the roof may not be exposed to dangerous fumes.

As to the prophecy that the life of the building will be shorter than expected, I am sorry that Mr. Schmitt is unable to take a more cheerful view, but now that the building is up, we can only wait and see what happens.

Touching the cleansing of brick walls, I fear there are cases in which the brick walls of compounds do require something to be spent upon their maintenance. I have unhappily not shared Mr. Schmitt's experience, that "good bricks can be had for the same price as bad ones," although I have found in some parts of the country that even bad bricks are not to be had, except at the price of good ones. It may, I hope, be assumed that none of us use any second class material when first class material is procurable.

With regard to the cost stated, I am prepared to give and take a little, but not much. Our compound, as it stands, cost about £3 10s. per boy, but as it includes some reconstructed buildings, I have not the exact figures for a compound of all new construction. I consider, however, that similar accommodation can be supplied, all new construction, for £3 15s. per boy in the case of a compound such as the one in question. The suggestion that increased cost of construction may be regarded with composure, when it results from the preferential use of local material indicates a fine public spirit, but it involves an important question of general policy.

I hardly think that Mr. Schmitt would seriously maintain that an engineer is justified under any circumstances in increasing the cost of any piece of work to benefit a local industry, at the expense of his company, without first securing the shareholder's approval of this generous policy.

The President: Before proceeding to Mr. Tom Johnson's reply to the discussion upon his paper, Mr. Rowland will, with Mr. Johnson's permission, read a contribution from Mr. J. M. Phillips.

NOTES ON RAND MINING.

(Read at March Meeting, 1908.)

By Tom Johnson (Member).

DISCUSSION.

Mr. J. M. Phillips (*Member*): Prof. J. Yates in referring to that part of my contribution wherein I state, "but I prefer underhand stoping for machines, as there you have better footing for rigging up, and all the holes are water holes, which one cannot have either in overhead or breast stoping," says, "that, of course, is not correct." I did not mean that *no* water holes could be drilled in either overhead or breast stoping, but that by drilling underhand all the holes are "water," whilst in "overhead" or "breast" stoping the stopes cannot be kept in proper order unless some of the holes are drilled "dry."

From practical experience I have found that in "underhand" stoping 100% of the holes are "wet," in "breast" stoping 95% are "wet," in "overhead" stoping—six-hole bench—70% may be wet, but with a four-hole bench only 50% can be reckoned on.

No matter how well a hole may be placed it cannot be absolutely relied on to break according to one's intention. One may "kick" far beyond expectations whilst another may fail to bring its burden, or there may be a missfire, and thus your ideal stope is knocked out of shape; and even in breast stoping it will be found necessary to drill one or more dry holes to bring it back to its proper shape again. Even if it were possible to drill every hole wet, which I contend is impossible in practice, it is not

always economical to do so, and although from a health point of view every man should drill wet holes where ever possible, yet if a man is on contract he will drill dry holes, if by so doing he can break more ground, and nothing less than the "sack" will stop him.

I am sorry that the illustrations showing how an overhead stope can be worked by wet holes only is not reproduced in the *Journal*, as it would be very interesting and perhaps instructive to myself and those other outside members who cannot attend our meetings.

I agree with Prof. Yates in saying that it is very unusual to see a stope being worked by drilling all the holes dry, and I cannot think that the cases are as many as Mr. Coombe states.

Mr. Tom Johnson (*Member*): Several of the contributors to the discussion on this paper expressed the opinion that in writing it I intended to wake up some of the members. I shall feel amply repaid for my trouble if I can justly claim to have done so. In the paper I left many openings for attack, hoping that members would seize on them and express some opinions which would take some combating, as I believe one can never get value from a paper unless the discussion causes the writer to either prove his words or withdraw them. As several say, the paper covered a wide field, and it would, no doubt, have been better if some of the subjects had been dealt with alone; but before going on my holiday I only had a short time at my disposal, and thought I might as well provide something for members to think about, as it would be beyond the usual time before I could hope to reply. Several of my critics seem to wish to saddle me with advocating two shafts for a property; I don't do anything of the sort. Some also have a habit of dismissing a subject without investigation, as in the matter of boilers underground, for instance, when they are probably without any experience of it. I welcome any critic who will go for me good and strong, if he will bring some argument forward.

It might interest members to hear of a little job that was done in an upcast shaft at Strangeways Hall Colliery, Wigan.* This upcast shaft, with a furnace situated in a drift at 1,200 ft. from surface, was enlarged from 10 ft. to 15½ ft. diameter whilst being available to wind coal or men. The shaft is 2,163 ft. deep, and the top 1,620 had to be enlarged. Coal winding took place 8 hours and stripping 16 hours out of the 24—four seams being dependent on the shaft for ventilation. Air for the sinkers was carried down in two 18-in. dia. pipes for a distance of 300 ft., below that no pipes were used, the fresh air travelling down the side of the shaft and returning up the centre with the ventilating current from the four seams. I wish to draw attention to the manner in which the ventilation of the portion being sunk was effected. Here we have a heated column of air rushing up the centre of the shaft and a fresh-air current going down the sides of the same shaft. At the time the paper was written the fresh air was travelling down the 300 ft. of ventilating pipes and 368 ft. free. This natural brattice can be found in some of our incline shafts which are being sunk with hammer boys. I mention this work to show what can be done in upcast furnace shafts.

Prof. Yates thinks I am wrong in comparing Kimberley and the Rand, because of our compound shafts and the many loading stations. I am far from agreeing that the compound shaft is the best, as may be seen in my reference to bends; also, I claim that we are wrong in using so many loading stations at one time, for so many loading stations are surely not necessary if, as Mr. Saner mentions, a level is only in use for a year or so. If compound shafts are the best, I am wrong in my comparison, as I do not think the same amount can be pulled from a compound shaft because of the bends and the several loading stations. In dealing with questions like this we must look to, not only what has been done, but what is being done and is proposed to be done in the future: it seems to be forgotten that shaft hoisting, under as good conditions as at Kimberley, is quite possible at several places here on the Rand, and I think it will surprise many of our members to know that there are not many of our hoisting engines that run more than 25% to 30% of the available time, even amongst those that are supposed to be doing a lot of work. If any one doubts this statement, let him time a few engines and see how long it takes to hoist a trip, and then get the average number of trips per 24 hours; he will learn something about the waste of engines. Winding costs are over 1s. per ton, and I am of opinion that some of this can be saved by giving our hoisting engines more work to do.

It gave me great pleasure to read Mr. Edwards' contribution to the discussion, as I like to see people alive to the supposed dangers of any new procedure. The danger of flushing in tailings is not so great in the proposed system as he believes, for what I am proposing is, I believe, quite different from the case he quotes. I am alive to possible trouble, but the manner of carrying out the work would provide the means of dealing with it. I note in the July *Journal* that the late Mr. Gluyas tried the dumping of tailings into the

* Trans. Inst. M.E., vol. xxxv., part 3. Enlarging upcast shaft from 10½ ft. to 15 ft. dia. whilst available for winding men and coal.—Mr. Chas. F. Bouchier.

stopes, but this is quite a different thing to water stowage.

Mr. Phillips thinks that to hoist an average quantity of 5,000 tons daily through two compartments from a depth of 3,000 ft. is not possible, but he is wrong, as it has been proved that greater quantities have been hoisted from less depths. I am of the opinion that we on the Rand can, if we try, beat all known records in hoisting rock. If Mr. Phillips had started with a useful load of 8 tons, say, instead of a puny 4 tons, he would have found there was plenty of time. If it is thought that a load of 8 tons, plus skip, say, 4 tons, is too great, I would point out that there are plenty of instances in practice where greater loads are on the rope end. The question of hoisting rock wants looking at, not from the point of view of what we see being done here on the Rand, but from that of the best practice elsewhere. As we are able and willing to show others something worth while copying, so we should copy any improvement that others can show us, when it is suited to our conditions; we must not wrap ourselves in any cloak of prejudice to the detriment of the industry. When we know of instances of 3,000 tons of coal being hoisted in 10 hours, are we to sit down and say it cannot be done here when we have better facilities for doing so now opening up. What is possible for others to do, is, given as good conditions, possible for us to do.

Mr. Phillips asks how is it possible to handle the tools, men, etc., in addition to the rock? He says he has been on mines where it takes a separate engine to handle men, tools, etc.; so have I, and for that reason I fancy I know a little of what I am talking about. In many places we still hoist men at midday, we do not do it during the night shift, is it therefore necessary? Is there not also too much time spent in handling material, drills, etc.? I think underground furnaces, machine repair shops, detachable bits, etc., will make it unnecessary to keep engines for this odd work; there is too much odd winding allowed because we do not feel it with so many engines around. A changing gear and a couple of spare skips for handling drills would be a great help to many mines at the present time. If we get away from some of our old ideas and better our system of working we shall not want so many persons for the work as we use at present.

As to handling men, I have known the skips changed and 1,000 persons put down in $1\frac{1}{4}$ hours with one pair of engines here on the Rand.

I figure out the problem of hoisting 5,000 tons of rock, handling the men, examining ropes, etc., with one pair of engines, depth of hoist 3,000 ft. in the following manner:—

Quantity 5,000 tons from a depth of 3,000 ft.; rock 8 tons, skip 4 tons, maximum speed 3,500 ft. per minute, 15 seconds acceleration, 10 seconds retardation, 64 seconds running time and 26 seconds loading time, or a total of 90 seconds per trip, 40 trips per hour, giving 320 tons per hour, say, 16 hours for rock hoisting. Say, number of persons per shift 2,000, 50 persons per trip equals 40 trips, maximum running speed 3,000 ft. per minute, 13 seconds acceleration, 10 seconds retardation, running time 72 seconds, loading 48 seconds, making a total of 130 seconds per trip, giving 30 trips per hour, equals 1 hour 20 minutes each time of raising or lowering the men, equals 5 hours 20 minutes for the two shifts, leaving 2 hours 40 minutes for daily examination of ropes, with Sundays for weekly examination. If more persons are needed, the number per trip could be raised to 80, which in the same time would mean 3,200 lowered or raised. These times can be improved upon as the rock loading ought to be done in 15 seconds, and men loading in 30 seconds per trip. I have not worked out whether the above conditions would give the greatest useful effect for the engines, but I do not think they are far from what would be good practice, but a little heavier load and slower maximum speed might be better. Under the above conditions the extra load due to acceleration, if uniform, would be about 3 tons for the rock hoist.

The ropes for above work may be either parallel or taper, running on a spiral or spiral-cylindrical drum or running on a parallel drum with a balance rope. It must be understood that with a parallel drum and taper ropes the balance rope could not be fastened to the cages, but would be run separately by a light rope running on a Koepe pulley fixed on the drum shaft, the balance rope being run in any part of the shaft that was most convenient. With a spiral or spiral-cylindrical drum there is great weight but no tail rope is needed; with parallel drum and taper ropes we have a lighter drum, but also the weight of Koepe pulley, running rope for balance rope, and balance rope heavy enough to balance its own hoisting rope, and the rock hoisting rope. With parallel rope and drum the balance rope can be fastened to the bottom of the cages or skips, this makes the changing of cages and skips more difficult. If it is considered preferable to use the parallel rope and drum, probably with a flat rope as balance rope, and providing means for facilitating the changing of cages and skips, then the rope would be locked coil, $5\frac{1}{2}$ in. circumference, $1\frac{3}{4}$ in. diameter, 45 lb. per fathom, weight 11·25 tons, total load 23·25 tons, breaking strain 188 tons, and factor of safety 8; the drum 17 ft. diameter, grooved

$1\frac{7}{8}$ in. pitch, the ropes to chase across the drum. The breaking strain I have taken from a maker's table, the long tons of which I have converted into short tons, giving a reduction of strength of about 10·7% for difference of table strength and assumed actual strength as used in the example. The factor of safety of 8 for new ropes going on is high enough for these heavy ropes.

I do not know if the Government Inspectors would think favourably of a reduction of the factor of safety for these heavy ropes, but I think if pulleys and drums of proper size were used it would be quite safe to reduce it. I fail to see why, if a factor of 6 is sufficient for the lighter ropes in use at present, that 5 should not be sufficient for these heavier ropes when taken off. The following example will show my reasoning. Assume we have two ropes of the kind used for the above example, one $5\frac{1}{2}$ in. circumference, breaking strength 188 tons, factor of safety 8, load 23·25 tons; this rope is used until the factor of safety is reduced to 6, and there is still a strength of 141 tons left in it; and the other a rope of $3\frac{7}{8}$ in. circumference, breaking strength 90 tons, factor of safety 8, load 11·25 tons; this rope is used until the factor of safety is reduced to 6 as before, the strength of the rope being 67·5 tons; why should we require so great a difference in the strength of the two ropes when taken off? If we were allowed to work the larger rope until the factor of safety were 5 there would still be a strength of 117·5 tons left in the rope, surely a safe enough margin; under these conditions would the dangers of hoisting men be greater than now? This principle is carried out in testing the strength of boilers; boilers working at pressures up 75 lb. are tested to double working pressure, but a boiler for 150 lb. is tested to 225 lb., that is 50% over working pressure.

I should prefer to use the parallel drum with taper ropes and Koepe pulley, as, although it is heavier than the parallel drum and parallel ropes, it is the handiest for changing skips and cages, also the sump is clear under the skips, in this case the ropes would be of the following sizes:—

Lower 1,000 ft., $4\frac{7}{8}$ in. circumference, 23·5 lb. per fathom, load 12 + 2 tons, breaking strength 112 tons, factor of safety 8; second 1,000 ft., $5\frac{1}{4}$ in. circumference, 27 lb. per fathom, load 12 + 2 + 2·26 = 16·26 tons, breaking strength 134 tons, factor of safety 8; upper 1,000 ft., $5\frac{1}{2}$ in. circumference, 29 lb. per fathom, load 12 + 2 + 2·26 + 2·42 = 18·68 tons, breaking strength 147 tons, factor of safety 8; the drum to be grooved $1\frac{7}{8}$ in. pitch, the ropes to chase across the drum; the balance rope would be 12·0 ton weight, run by a locked coil rope of $3\frac{1}{4}$ in. circumference and a breaking strength of 80 tons; 12 tons gives a little overbalance which is good. All hoisting ropes to be subjected to Messrs. Vaughan and Epton's fatigue test in addition to ordinary test before being used.

I am very sorry if I led Mr. Phillips astray with regard to rail guides, but in speaking of rail guides and rope guides at the same time I thought it would be understood what kind of rail guides were meant, as they are common enough in these times; I meant rail guides as used in collieries, the slippers on the cage encircling the head of the rail; I said two guides per skip, but four would be better. I am afraid Mr. Phillips has not read the paper carefully, as he charges me with inconsistency with regard to the two shaft sinking. I said, if on a property two shafts had to be sunk I would sink them together, meaning such a property as is generally understood, say, 200 to 300 claims; in the other case I was speaking of two shafts going down on the incline, each serving several properties as mentioned; these shafts would have the development drivages going off from them all the time, as is the present practice; the winzes, of course, would be the second outlet, facilitating ventilation. This, to my mind, is quite a different proposition from the other, and I cannot take any blame for inconsistency because he has failed to see the difference; he seems to have missed the point altogether in this matter of incline shafts, and I will take a few points he mentions and try to prove to him how easy it is to be mistaken. For inclines for great depths he fails to see any advantage of the incline over the vertical for the following reasons:—(*a*) more footage for incline shafts; in the case I cited most of the footage is already partly done, or if a complete new shaft was to be sunk under the footwall of present stopes, there are so many points of attack open that the footage costs would be no greater for the greater incline distance than for the necessary vertical distance (also see below); (*b*) more timber and ropes; I do not think so, in most inclines I know of there is a great deal less timber than in the vertical, as to ropes, it is a moot point; (*c*) wear and tear of rope greater; like (*b*); (*d*) extra costs of rolls for rope; (*e*) heavier and closer timber; no; (*f*) loading stations at every two or three stations at least; does Mr. Phillips seriously mean that, if he were to sink a deep shaft, say, 3,000 ft. he would compound it; I believe not. Would he not have to go on the incline in some manner and use loading stations? (See Fig. I.)

As is well known, for these deep shafts the companies start close to the northern boundary to sink, as at (*b*), Fig. I., and reach the reef at (*c*), and may sink for another level and put in permanent loading station, then they have to go on the incline; how many stations are saved by doing this instead of coming down on the incline from (*a*)?

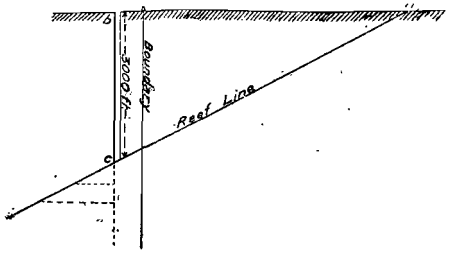

FIG I

Taken from a purely mining point, the incline method of attacking the ground mentioned actually requires the shortest footage (see Fig. II.).

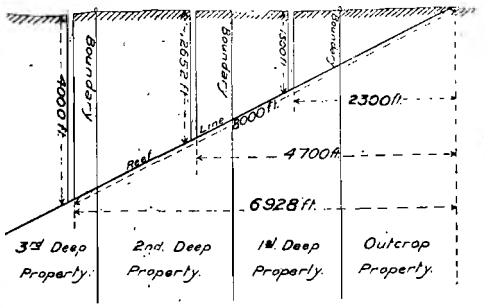

FIG. II

In the sketch we have the surface pegged out by the Outcrop Co, The First Deep Co., Second Deep Co. and Third Deep Co.; the Outcrop Co. sinks its incline and the Deep Cos. their verticals. Now, if instead of the three Deep Cos. sinking vertical shafts the incline shaft had been carried on large enough to serve the several companies, which would take the greatest amount of sinking?

Let us calculate the footage that is done by the four companies and what the footage would be in my scheme of incline work taking the dip at 30°. Say, the Third Deep sinks its shaft just inside its northern boundary and takes 4,000 ft. to reach the reef, the distance from outcrop being 6,928 ft.; the Second Deep sinks its shaft, say, 4,700 ft. from outcrop, and takes 2,652 ft. to reach the reef; the First Deep sinks at 2,300 ft. from the outcrop, taking 1,300 ft. to reach the reef; this gives us for the three deeps 4,000 + 2,652 + 1,300 = 7,952 ft. without outcrop. The distance on the incline to reach the point where the Third Deep shaft gets the reef is 8,000 ft., not much difference, is there? especially when one comes to think of the fact that practically the whole of the 8,000 ft. of incline shaft sinking is done in addition to the vertical work; this latter fact seems to have escaped Mr. Phillips, as it has done many others. The shafts are still open to the danger of subsidences, but are not vertical shafts liable to subsidence if the pillars are taken from about them.

As to whether I am right or not in saying there is too much timber used in our shafts is, of course, a matter of opinion, but there are a few miles of timbering in our shafts where nothing but blocking touches ground; could not the wall and end plates be lighter in these cases, and could not the distance centre to centre be expanded a little. At 6 ft. centres timbering is costing, I should say, £3 per foot in the seven-compartment shafts, and if we put them at 9 ft. centres, which could easily and safely be done for nine-tenths of the distance, something would be saved on timber alone, besides wages and time.

Long Incline Hoisting.—I said, "using endless ropes and cars on inclines a smaller shaft is permissible for a maximum tonnage," this is true on short inclines, and there is no doubt that better work with the present tonnages could be done with the endless rope system in many of our incline shafts than with skips. On long inclines, with big tonnages, skips would probably pay best because of the expense of handling. I expected someone would have asked, if, as I say, 5,000 tons could be hoisted and men handled in two compartments, 6 ft. × 5 ft., of a vertical shaft, what could be hoisted from the same depth through an incline of 30° dip, and what size would the compartments be? The same amount of rock could be hoisted, the unit amount handled being greater. The load would be: rock 12 tons, skip 7, rope 16, and friction 7, equals ·42 × ·5, equals dead load 21 tons; rope 5 in. circumference, breaking strain 155 tons, factor of safety 7·4, maximum rope speed 4,000 ft. per minute, acceleration 10 seconds, retardation 10 seconds, time of running 100 seconds, filling 33 seconds, making 133 seconds per trip, 27 trips per hour or 320 tons, as before, the size of compartments being 6 ft. wide by 7·5 ft. high from sills. These large total loads are nothing uncommon as I know of instances where a load of 18 long tons* was hung under the capping.

Mr. Saner asks how it is that in other places they are sinking large shafts for smaller quantities and shorter distances than I mention in my paper? If he had read my paper carefully he would have found that I said "the amount of rock to be hoisted should not be the measure of the size of the shaft, but that the amount of air needed for the number of persons to handle the rock should be the guide." I particularly drew attention to the fact that the three compartments mentioned were not large enough to pass the necessary air without an extravagant waste of power, clearly implying that I did not consider such a shaft large enough; as a matter of fact, I never calculated what the necessary size

* Winding Ropes and Capels. W. Routledge, Trans. I.M.E., vol. xxxiv., part i.

of shaft should be, as that has no place in this paper. Mr. Saner quotes the Bentley Collieries' shafts as being 20 ft. in diameter for 2,000 tons per shift, but he forgets to mention the fact that one pair of engines is to be used for this work; if we were only doing as well it would be a great deal better than at present. How many cases from the whole of the mines of the Rand can he quote me of where two pairs of engines working two shifts per day are doing that amount—not many, I am sorry to say. I do not know why the Bentley people put down their shafts 20 ft. in diameter. There are several points that they would have to consider, two vital ones being the size of cage to carry the cars holding the coal, and the amount of air necessary to provide an adequate ventilation; now we know that for either of these reasons a colliery shaft would need to be larger than a gold mine shaft for the same tonnage, for the bulk of coal is greater than rock and has to be carried in cars, also the amount of air necessary is greater to carry off the gases liable to be met with. If giving an opinion why those shafts are 20 ft. in diameter I should say it is for one of the above reasons, but even if their shafts were too large for the work, that is no excuse for our doing likewise.

Mr. Saner thinks the comparison of Kimberley and the Rand absurd; not hardly so bad as that; he, himself, is manager of a shaft where the hauling will, or should be, under as good conditions as Kimberley, and how much rock is it proposed to hoist through the six hoisting compartments with the three pairs of engines? Also, take the South City, for example—if 1,000 stamps are erected the required tonnage of rock will be about 12,000 tons per working day; then three sets of engines should do that, and handle the men, tools, etc., that are necessary for the work. At these and other shafts there is only one station to hoist from, and the width of seam or reef, dip and facilities for handling the rock under ground do not enter into the question: if two engines are put down to do the work that one can do, it is no use talking of width of seam, dip, etc. If the rock cannot be got to the box it only makes matters worse; there is some excuse for having a little extra power to handle the rock one can get to the box, but to put in engines to hoist rock one cannot get to the boxes is madness.

Mr. Saner says mining banket is not mining coal. I have done both and agree with him; but in hoisting who has the best of it—the man who handles the greater volume for the same tonnage and a product that suffers by being knocked about, or the man who hoists the lesser volume, say, four-sevenths of the other, and a product which benefits by being knocked about? Consider the shaft area occupied by the cages to hoist 4 tons per trip in both these cases; in the Bentley district the cars would hold about 1,000 lb. each, and each would be, say, 3 ft. × 2 ft., and eight would be required, four per deck, 6 ft. per car, say, 24 sq. ft. area without clearance; for rock we should take a skip 2 ft. 6 in. × 4 ft., say, 10 sq. ft., say, about two-fifths the space (see Fig. III.). It is something of this kind that

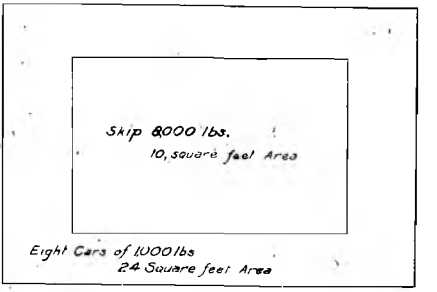

FIG. III.

troubled the Bentley people. Mr. Saner thinks it folly to spend money in elaborate arrangements when a station is only to last a year or so; well, I have seen 26,000 tons come off two levels in a month, and some of our members think three levels are not enough for a 200-stamp mill, but if, as he says, a level will be worked out in a year or so, I think there is much better work being done than some of us are aware of. As to the cost of fitting up boxes, I think money spent on boxes, within reason, is a good investment. I know I have been thankful for box room often enough, and the way I look at it is how much per ton will it cost, and how much am I likely to gain. Assume that on a mine with 250 ft. backs it is decided to put in boxes only at every second level; at the odd levels a crosscut has to be driven to such a point that a pass going down at 55° will strike the box of the level below; how much extra money will it cost in comparison with making two boxes, and when you spread it over the tonnage of two levels of a mine 5,000 ft. long with 12 ft. of stoping width it comes out at $\frac{1}{100}$d. per ton for every £100 spent extra.

Mr. Saner thinks I am wrong in saying that we are not in the front rank in mining matters, because we have some of the finest mining men in the world here. It does not matter, to my mind, how good these mining men may be; they are human like the rest of us, and liable to have the same failings. I do not think they claim to be infallible, nor do I think they should be, they would have no business here if they were. I fancy there is room for improvement, and that improvements can be made, especially in our methods of handling rock, so in my own clumsy way I am trying to point out where I think something can be done. Mr. Saner says, given plenty

of room above the collar, accidents should be few or none with good drivers; but what about room above the tip—it is there we require a little room, and most of us are loth to leave room there, as we want all we can get under the tip. Considering the number of trips an engine has to make during a shift, the weight to be set in motion, the velocity to be attained, and the short time between starting and stopping the load, I think it is expecting too much from the engineman to go on year in and year out without an accident, no matter how good a man he may be.

I have long been in favour of applying to coloured labourers the no-work no-food motto, but have always seen the difficulty of cheap food, as I do not think the Government would allow the companies to control the sale of food to the coloured workers, as it savours too much of truck.

Mr. Weston says Prof. Yates has pointed out that it is a different thing to hoist rock from one point than from eight or ten; it may seem strange to Mr. Weston, but I am quite aware of the fact. The point that requires elucidation is whether the large number of points are necessary: I believe not, and suggest for the main hoisting, one; for the subsidary or secondary, as few as possible—say, three on most mines. I know of many places where there are two shafts—one shaft having a rock hoist, and the other a rock hoist and a man hoist—neither working 30% of their time when they ought to be running 60% to 70%; this is not done because of faults, dykes, etc.

As to reading technical literature, I am pleased to say I do, and have done so for many years, and if Mr. Weston had done so he might have had a nodding acquaintance with some of the points raised in my paper. His remarks *re* circular shafts are not practical, and I would strongly advise him to pay a visit to one in process of sinking; and I hope there are not many more in the position of wanting to know, in these days, how rope guides are fastened in colliery circular shafts; there is one thing, they are not left hanging like the end of the bell line. As to the colliery arrangements reposing on the bottom of the shaft, I fancy the colliery people manage to get through without this happening.

As to handling rock on inclines, he has something to learn, but perhaps he has not had the opportunity of seeing cars hauled up a 45° incline as I have. He seems to have a vivid imagination of the mess there would be; I should expect an imagination like that would help him to devise means for preventing the mess. I know steam boilers underground are not so greatly in favour in collieries as they used to be, there being two very grave dangers in coal mines to guard against, that is danger of firing the coal and explosions of firedamp, dangers that are not present in our gold mines. There are some still in use, and as far as I can gather could not be beneficially changed on the score of economy. At one colliery I know of there are underground three Lancashire boilers, 28 ft. × 7 ft., which were taken down whole, except for the mountings, and have proved very satisfactory. Mr. Weston speaks of properties being divided by a fault or dyke with 500 to 1,000 ft. throw; but this is quite the exception, and when things like that occur the owners have a very sensible way of re-arranging claims to nullify the effects of such faults. He asks, have I never heard of bratticing a shaft? I have very often, and I know enough about ventilation to know that brattices are only a makeshift, and are of little use after starting away from the shaft. Has Mr. Weston never heard how hot our deep shafts are before they are connected, and how glad everyone is when the connection is effected. There are a few people going to learn something of ventilation when we get a handy electric drill going in our mines. Several ask me, could I develop as much ground from two shafts close together as I could when they are the usual distance apart. No, I could not, but if Figs. IV. and V. are studied a little, it will be found that I should not make a bad show, compared with the system of opening out that we are used to. Mr. Weston wonders if I am poking fun in suggesting ten machines on a face, I am not; I believe the time will come when it will be necessary to use large numbers of machines on each face, as the faces will have to advance much more quickly than now or they will not be kept open.

A short time ago Mr. Richardson read a paper in which some very useful points were discussed about the size of pillars and the weight they would carry in deep level mining; I am not sure whether I am quite correct or not, but I believe the general feeling was that we should not be able to leave pillars enough and work profitably at the same time; my idea is that we shall have to clear out everything and pack, if possible, moving the faces along at a fast rate. If the system of using more machines for each face is slowly introduced, I believe it will be found to be no more expensive than our present way of doing it; anyhow, it would not cost much to pick out a good worker who could be trusted to try the scheme fairly, and let him start in a stope not far away from the shaft, with three or four machines and helpers, and build up to the full amount of machines that could be worked on the face, to find the limit. I believe it will be found that four benches per machine, as sketched, will be enough, for I do not think a machineman can be found, running two or three machines, who keeps four benches per machine. If more machines are used

on a face it is reasonable to suppose that a less number of faces would be necessary for the same tonnage, also that any particular face would move faster and the block of ground be exhausted sooner, so that the blocks may be left larger, which would be done by either having the drives or winzes, or both, farther apart. Then as to brake roads, shaking shutes, etc., being more efficient, if more rock is broken more will need to be removed in the same time, and moving the greater quantity in the same time, there should be a reduction in the standing charges. It costs the same to fix up the rails at the face end of a brake road, or a shaking chute, for a small amount as for a larger amount, also if plenty of rock is about the labour force accomplish more and the gear is more protected. It is all a question of keeping down the unit cost.

Mr. Weston also asks what would the miner in charge of the face do if he found one workman drilling more ground than another? He must either sack the miner or the stope would get out of shape. Perhaps he would allow one of the above things to happen, but, fortunately, a miner would have the common sense to arrange that the best drillers would be put where the most drilling was necessary, as is done now by both machinemen and hammermen, if they know what they are paid for. As to handling machines, etc., I have always noticed that the good men are mostly those who move their gear the least distance, generally a bench or so away. He also thinks a machine ought to have 80 ft. of face to work on, I cannot see why; if a stope is in benches I fail to see why if a machine, call it No. 1, is on a bench, the other benches should wait for No. 1 machine; what is the reason the other machines should not be used on some of them if they can be broken with advantage? if the same mind is directing the whole of the machines the benches may as well be broken in a shorter time by the use of more machines instead of waiting for No. 1 to be free. Speaking of the diagrams of stoping holes, he says he would be under the necessity of sacking a man if he found him with holes placed as I show them, and I fully believe he would, judging by his remarks on mining. He is not the first to be deceived by the look of the second hole to go having apparently little to do, it is that little to do that counts. I have often seen men try his system and be quite surprised to find the holes looking at them when they got back to the face. Mr. Weston must have been more fortunate than I, and some others I know, in the ground he has worked in, for I have seen men in low stopes drilling on his system on contract and not making wages, and when they have been placed on day's pays and made to drill as sketched, in less than a week they wished to go on contract again. As to it not being necessary to fan back holes a little, how is the bench to be held to size in most cases? the machine is wider than the hole and the bench breaks off at the hole, so if the hole does not look in a little the bench is smaller at the back than at the front; if Mr. Weston has been used to ground that breaks back I must again compliment him on his good fortune, or is that the reason why a machine wants 80 ft. of room so as to form new benches as the others get too small? In some cases where the bench is wider than the amount to be drilled then there is no need to fan the back holes. If the amount of ground can be broken in a 38-in. stope, as he implies, which means practically a bench 6 ft. across for five holes, I am wondering why breaking costs are not lower. Mr. Weston thinks handling the machine under the arm is wasting time; he is not alone in that, but that does not make him right: where there is no parting in the foot, good stoping cannot be done without putting the machine under the arm for the bottom holes, and the man who does not put the machine under does not know how to handle it, or is troubled with the fault of many of our Rand miners—tiredness. The chisel bit has the four reaming edges he speaks of. Several critics think I am wrong in having used 2½d. and 3½d. steel. I used 2½d. steel for the 8 ft. drills and over, and for shanking; and 3½d. steel for the other drills. Some time ago a paper was read before the Society on safety fuse, cheap fuse being condemned in the same manner that cheap steel is being now. I found the price of fuse go up, but should not like to swear that the quality did.

Development from two shafts close together, or from a single shaft (see Figs. IV. and V.).—If in sinking, No. 1 shaft gets to the reef first, start the second and first crosscuts, then as soon as the second shaft gets to the reef, start driving both ways until points of first winzes are reached, say, 400 to 500 ft. distant. Start off these first winzes as development shafts, say, 10 ft. wide, opening out the levels from these; if at any time it were found that these did not give length enough between, then the next winze on one side, or both sides, could be used, but I do not think it would be necessary; in the meantime carry on the sinking of No. 2 shaft until bottom is reached, then start number three and four crosscuts, cut stations and ore passes; the main incline would be started from number three and carried on, a temporary box being put in there. As soon as number four station was ready, and connection made with number four drives, the development rock from fourth and below would go to number four station, and when ore passes were ready the rock

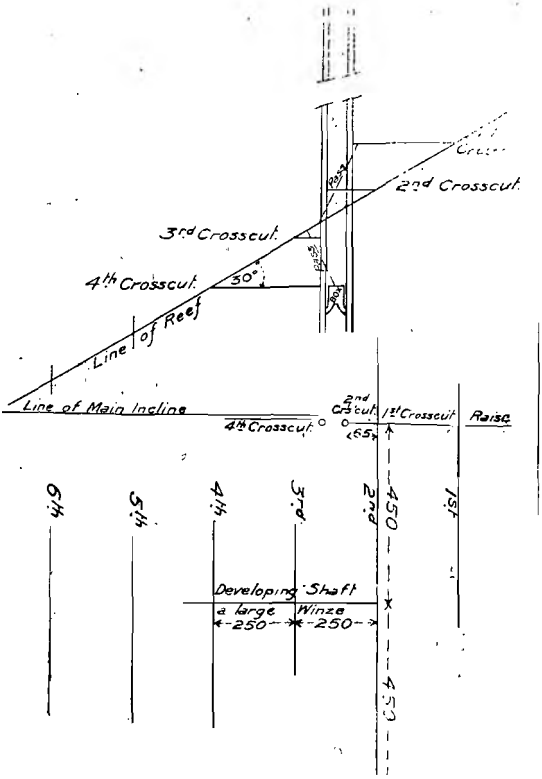

FIG. IV.—SECTIONAL ELEVATION.

FIG. V.—PLAN.

from upper levels would be handled there, and number one sinking, etc., could be finished. If No. 2 shaft gets to the reef first, go below a little and then start Nos. 3, 2, and 1 crosscut; when reef was reached on No. 3, start driving both ways to point of first winzes, and start the development shafts and main incline; carry on sinking at No. 1 shaft to bottom, put in box, cut station, and drive No. 4 crosscut; when connection is made with the fourth drives, and the boxes, etc., finished, turn development rock to this shaft and finish sinking No. 2. It will be seen that cutting stations does not interfere with the development, and it is the same in the main incline, the development will go on without interfering with it, nor will it suffer whilst stations are being cut; the development rock will go up to a station above through the development shafts or winzes; as portions of the shafts or winzes are freed by the hoists being advanced, such portions become available as faces to commence stoping from. In developing from a single shaft, as soon as room for number three crosscut is attained, start it, and when reef is reached drive on it as before and start development shafts or winzes, using one pair of skips to handle the development work, and the other pair for finishing the sinking and cutting stations and boxes and crosscut at number four. If, as it should be, the higher side company had driven their incline down, the higher levels could be started also, the rock coming down to number three box.

Mr. Lane Carter thinks a good electric bell system is sufficient in the majority of cases, but I am still of opinion that telephones are a paying proposition, and I also strongly advocate a speaking tube from banksman to engineman. With regard to rectangular versus round shafts, he brings forward the old stock argument of utilisation of space; this utilisation of space has cost the Rand some thousands of pounds, and is a poor argument, as I hope to prove. In my opinion, the reasons why we keep to the rectangular style of shaft are:—(*a*) Use, it being second nature to use them; it would naturally be the form chosen for sinking on the reef, so as to have as much reef in the shaft as possible, not only to ascertain the value, but also to help to pay expenses. (*b*) The facility for lining with timber, which is generally the most handy thing to be got for the purpose in new countries. (*c*) What I call the mistaken idea that we want so much room for hoisting purposes. (*d*) Being able to turn bends from them and follow the dip of the reef. For the last few years I do not think that any of these reasons should have been allowed to carry any weight as argument against round shafts, as:—(*a*) In all our vertical shafts we have had to go through country. (*b*) Where lining was necessary we have had the choice of other material than wood. (*c*) All the space of a shaft should not be taken up by the hoisting arrangements, it not being necessary and certainly not good policy. (*d*) In some cases bends have not been put in, and in many others where they have, it would have been much better if they had not. I believe, myself, that the question is simply one of cost, and that a round shaft would come out cheaper than a rectangular one. It should be sunk faster, as it has a much larger sumping area and has less periphery to be dressed down. I am glad to see he agrees that there is a chance for endless rope work underground on the inclines, and I am sure that if the system is introduced it will be found good policy to adopt it both on the rise side and dip side of vertical shafts. I agree with him that electricity will be very good for underground work, if we are to make extended use of it above ground, for it is not the best policy to have several different methods of power supply on the same place. With regard to power schemes, I have often wondered why some of our large firms have not taken a colliery, turned the coal into gas, generated electricity at the colliery, and transmitted the power along the Rand.

He does not think there are many mines along the reef that can supply a 200-stamp mill off three

levels; there are quite a few if we can only get rid of some of our old ideas. A 200-stamp mill, milling 7 tons per stamp per 24 hours would, with 20% sorting, need 1,000 tons per shift broken in the mine, and 1,000 ÷ 3 equals 333 tons per level. Say, the property is 5,000 ft. long, of which 4,000 ft. is open, with winzes every 500 ft., giving 16 faces per level; 333 tons ÷ 16 gives, say, 21 tons per face, which on a 4½-ft. stoping width means a little over 1½ fathoms per face; this is for one reef with a 4½-ft. stoping width, and I think the greater part of the mines with 200 stamps have more than 4½-ft. stoping width.

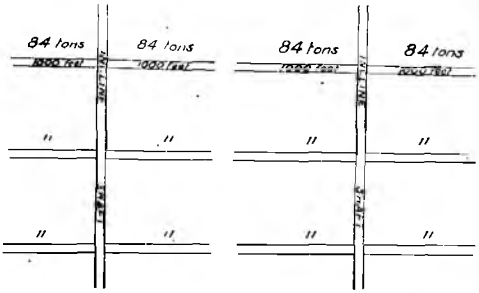

FIG. VI.

Now we have two shafts with the three levels—(see Fig. VI.)—open on each side, making twelve short levels; 1,000 tons have to come off these twelve levels, which equals 84 tons per level, and with, say, nine hours' tramming, there is 9⅓ tons per hour from each, not too much to expect, I think. I cannot see why this could not be done in most mines, or why in the hoisting problem we should pass remarks about having to haul from nine or ten stations. It is known to all of you that things are being done on the Rand now that many people two years ago would have been ready to swear could not be done, and we must always remember that the impossibilities of a few years ago are the common occurrences of to-day. I agree that the contractor should have a direct interest in saving stores, etc.; the blasting and rigging gang would, of course, consist of one of the contractors, "of whom there should be three, if two shifts are working," with help to clear away the gear whilst he himself would be loading up the holes. With regard to packing with sands, many areas could be flushed full without any gas that might be produced having a chance to injure anyone, notably in the outcrop mines, and these are going to prove dangerous to the lower side mines if something of the kind is not done. There are many areas in a mine where acid water would never come into contact with the tailings, and it must not be forgotten that by packing our stopes we bring about better ventilation to deal with any gases given off.

The system of drilling, as shown in Fig. V. of the paper, I have found to answer very well in avoiding breaking too much waste; it will be found that no hole is fast, and it will not cut into roof or floor; it is principally for places where there is no parting at roof or floor, and where there is a false hanging at a short distance above the hanging of the stope. As regards two classes of certificates for miners I do not thing we shall get much benefit from them; for with one certificate we always have the date of issue for a guide. It is a very hard thing to judge miners; I have seen places where old hands, with years of Rand experience, could not make anything like wages, and where men who had not had two months experience made twice as much money. I have also seen clinking good men in certain stopes become failures in other stopes of the same mine with different dip or width of stope. The system I am advocating of putting more machines on a face would be a complete failure, from want of adaptability, if the different machine runners were allowed to go their own way; but by putting a man in charge of the face who has been proved to be a good man under the conditions ruling there, we ought to get better work done. I think it much better to take the man whom we know to be a good man and use his ability to direct ten or twelve machines, instead of leaving him on two or three, and leaving the other machines in the hands of incompetent men.

Mr. Phillips says truly that the output of a stope is limited by the amount of rock that can be taken away from it, but I hold the opinion that more rock than four or six machines ordinarily produce can be taken out of a stope. Some adopt the principle of splitting up the face into sections for shovelling purposes, but work the sections alternately according to where the rock is; I believe this principle can be carried further and the several sections worked together. He says he has no faith in a rigging gang, but likes every man to rig up his own machine; I find that every man is not competent to do so, and in the way I propose we have a much better chance of getting a good man who will see to the machines being rigged up properly. As to being able to keep plenty of rock in stopes I cannot see why big mills should make any difference; the mills must be supplied; so to get extra rock the expense of a few extra machines should be borne for awhile until a surplus is produced; of course, if a manager has not got a labour force large enough he cannot do it: available rock should be treated like development work and kept well ahead of requirements, as it comes in handy in case of breakdowns. I did not know that I advanced any claim as to sorting and packing underground being new, nor did I claim anything new for

flushing tailings into stopes, although whoever starts doing it will be the first here; I believe I mentioned that packing workings by this method started in America in 1887.

With regard to long and short holes, if Mr. Phillips had noticed the diagrams he would have seen that the length of hole varied for different stopes; in the tight stopes the length is 6 ft. He says the average length of hole here is 6 ft. and average burden 2·5 ft. I wish I could believe him, for the average footage drilled per machine was not above 21 ft. and fathoms broken not ·5 on the mines of one of our largest firms here for the year 1907. I am glad there is something we can agree about, for he says that where holes bull-ring he prefers to put a little more burden on and give a little more explosive; so do I. I think it a better paying way than shortening the holes. Mr. Phillips says he has seen old miners since the eighties placing holes in the manner I describe, and that the old miners know all there is to know about placing holes; I am pleased to hear it, particularly as Mr. Weston says only incompetent men so place them. I do not think anyone can claim that all our old miners are good men, even if we restrict the term to the old Rand miners; I might have expressed myself better by saying old mining men in distinction to men of little previous mining experience. I did not use the words "the great majority," but, "a great proportion." As to the cut, there is plenty of time to prove it: I do not think Mr. Phillips has seen the cut as described used for pulling to a slip, if so, it is not the best way of doing it.

I am quite in agreement with him as regards an adequate supply of drills, spares, machines, etc, but there must also be adequate control over same. The system of drill supply I have been used to has been to send drills down twice per shift, once in the middle of the shift and then about the end of drilling time, so as to be ready for the oncoming shift. I have tried working men under bosses who have gone through the mill and had to prove themselves able to earn their own living at the work; I do not care where a man comes from, whether it is from the ranks or from a school, so long as he is capable. In dealing with new hands I do not think we should make hard and fast rules, but treat the men according to the diligence and ability shown. Some men would never make miners, whilst others seem to have a gift for mining.

I think Mr. Phillips made a slip in his example of a bench 10 ft. long by 6 ft. wide. If I drilled the four holes to break the bench, why should I only have 2 ft. 6 in. on? I should have slightly over 3 ft. on the front holes and slightly less on the back holes, and if the front holes bull-ringed I should not lose the bench, as I do not practice firing the four holes at one time. In overhand stoping it is not necessary to use dry holes; let the benches dip or run towards the bottom of the stope and then the holes will dip to carry water; this dip should be more pronounced than shown in Fig. III. of the paper. If a hole breaks badly and cannot be rectified by firing again, get a hammer hole in and clear that way instead of drilling dry to gain the bench again. If Mr. Phillips drilled dry holes on a bench with wet holes in front he must fan the back holes, which Mr. Weston says is not necessary.

I start by being in agreement with Mr. Schmitt with regard to different generations looking at things in a different light. There would be no need for papers and discussions if we all looked at things in the same way. When Mr. Schmitt says no one would sink a shaft larger than was necessary for hoisting the rock, he is wrong, for given that all the compartments provided are wanted, there is quite an amount of room provided in some of the new shafts as clearance, so as not to interfere with ventilation too much, thus recognising that ventilating room is required. I think it is cheaper to sink the shaft a little larger than to provide extra ventilating power; taking the three-compartment shaft as being large enough to supply hoisting room for the tonnage quoted, I should certainly sink a larger shaft, so as to provide for ventilation. From the number of persons, Mr. Schmitt requires to handle this tonnage, I should say he is a great advocate of hammer work; but to enable us to start up some fresh work when money comes our way again, we have got to get down to about half his estimate of coloured. There are objections to mechanical truck haulage on inclines of 45°, but I may say I have used it on inclines of 44°.

Mr. Richardson thinks it wants a stronger argument than the absence of record hoisting performances to show that we have a long way to travel before we are in the front rank. You know the old saying to the effect that some are born great, some acquire greatness, and others have greatness thrust upon them; I think there is a lot of the last about us and that we have got to fancy we have earned it. I do not think we have more than one mine that can show an average of over 400 tons per shift for the hoisting engines; and the large reduction in costs this last 18 months shows that we have had nothing to crow about.

Mr. Richardson gives us some very interesting figures on winding speeds, he also tells us that at Kimberley they have done 363 tons per hour on several occasions, better than many of our shafts

do in a shift. Then for British collieries he tells us the speed at Denaby and Cadeby is 2,289 ft. in 55 seconds; he might have gone further and told us that they hoist 269 short tons per hour with a useful load of only 4·48 tons, the total load being 22·4 tons. Bolsover* hoists 224 tons per hour with a useful load of 2·35 tons, Mamfield 336 tons per hour, Rotherham 323 tons, the useful load being 5·6 tons and the total load 20·6 tons, Baggeridge and Sandwell Park 430 tons per hour, useful load 7·16 tons, total load 19·5 tons, H. C. Frick Coke Co., U.S.A., 300 tons per hour, Hitchester Main 300 tons per hour, useful load 4·92, total load 21·67 tons (these are short tons): then they have a bulk 7/4 of ours for the same tonnage and have to carry it in cars. Looking at the total load relation to useful load at these places, and what it would be for us at the same depths, I think you will admit that we have the best chance. I mentioned the total loads at these places because some may think I have fixed the loads in my examples at too great a figure, but if these collieries had to hoist from 3,000 ft., as in the examples I give in answer to Mr. Phillips, they would have to use a much greater total load than they do at present and greater than the load I use. As shown in my example, it is possible to use the balance rope in such a manner that independent trips could be made. Oval shafts would be preferable to round where we wished to use two sets of skips in one shaft. I do not see why we should not use tipping skips with rope guides, by using wood steadying guides at the dump.

Now let us look into the matter of bends and compound shafts and see if we could not do as well, or better, without them. First, I must thank Mr. Richardson for his courtesy in using the Geldenhuis Deep as an example, as he knew I ought to be familiar with that mine.

In Figs. VII. and VIII. I show the No. 2 shaft

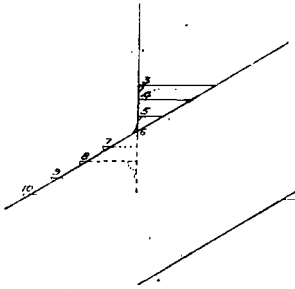

FIG. VII.

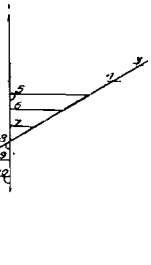

FIG. VIII.

as it is, also how I think it ought to be. In Fig. VII. there are shown crosscuts driven at Nos. 3, 4 and 5 levels, No. 3 being the top level at this shaft. The bend is turned off immediately above No. 6 level, the 7, 8, 9, 10, and so on, being on the incline. The same shaft could have been sunk as shown in dotted lines. Now, suppose the shaft had been sunk, say, 300 to 400 ft. further south to give more ground on top side of shaft, the distance to the reef would be greater by 200 ft. or so, giving five levels on the top side instead of three, then by sinking another 200 ft. or so we should have the 9th and 10th levels by crosscutting, by doing this it would be as in Fig. VIII. Here a crosscut is driven to what would be the 5th level and a rise sent up to open up the 4th and 3rd, the rock from these levels would be sent to the 5th by self-acting endless rope incline, crosscuts would be driven on the 6th and 7th levels on the north side, and on the south side we should have the 8th, 9th and 10th by crosscuts; boxes would be put in at the 5th, 8th and 10th; that is the way I should get out of having shallow vertical shafts, bends, and avoid duplicating machinery, and also find work for the one pair of engines that would be necessary. When it was necessary to develop the lower part of the mine this could be done with a small pair of engines. In cases like this, development would probably have been done much quicker by taking longer to sink the shaft, as more points are opened for attack. It may be all right for people to talk about saving money by using bends, but I have seen a good few mines using bends, and I fail to see any benefits arising from them, especially with the reef dipping 25° and over, the more the reef dips the less need for a bend. If the Geldenhuis Deep had been sunk, as in Fig. VIII., it would be an easier and cheaper mine to handle, despite the crosscuts shown.

Bends may be convenient when nothing has to be done beyond playing at hoisting with them, but try to push things and begins to appear trouble without looking far for it. Taking the illustration of the Geldenhuis Deep and Jumpers Deep, and assuming that in both cases the plant is worked to full capacity, which would be the cheapest way of doing the work—by two engines in the present manner or by having laid out the work as I have shown? As well argue that in a 6,000-ft. vertical shaft it is as cheap per ton to hoist by one lift as by stage winding when working at full capacity. Of course, someone will say it is quite easy to talk of it now, but I must say I have held the same ideas ever since I first knew the Rand. The accidents that Mr. Richardson quotes the number of, were not exactly the accidents to shafts, cages, engines, etc., that I referred to; many of them were of men falling from cages, because it is not the general custom at Home to use gates on the cages as we do here.

* Twenty years ago, a pair of Daglish's engines made a record run of 246 short tons per hour from 1,125 ft., at Bolsover.

The sinking of two shafts close together would not make ventilation much worse than now, for what is done at present to stop short circuiting between shafts? What difference is there between ventilating 2,000 ft. outwards from our shafts as at present, and ventilating 2,000 ft. outwards from the two shafts close together? In the proposed long inclines I gave alternate methods, but if stopes were packed, there would be no broken roof, even with what may be called a bad roof, as it would come down bodily; the most broken part of a caving is around the edges of the area, and I have seen miles of road open in caved areas where it could not be told by looking at the roof that there had been any caving. I believe Mr. Richardson and I will have to agree to disagree about subsidence. I have seen any amount of subsidence of surface over packed workings, never mind unpacked workings. When in Belgium, I went to visit a pit using the flushing method for packing; before I saw the pit I knew I was close to one, for every house in the village was cracked, and on making enquiries I found the company kept a number of bricklayers doing nothing but going through the village repairing houses. We shall know more about it by-and-bye, for the worked-out area keeps growing, and so does the caving in some of the mines. It is just as easy to keep rock in stopes when feeding large mills as small ones, provided you have the labour force. I hope Mr. Richardson never thought I seriously proposed to stope without supporting the roof; where was I going to put the tailings I used up so much ink in writing about? I made a slight mistake in the diagram of the new cut, as all the holes are water holes parallel with each other; both top and bottom holes are charged and lit together, the fuses being cut for the top hole to go first.

With regard to holidays, I have seen men who did not like taking a short holiday yearly, preferring to allow their leave to accumulate till they could take the long one. In the case of a man going to visit other fields I should, of course, expect the company to offer the encouragement financially, as it would be to their benefit to do so, at least I think so.

With reference to subsidences I hear that Johannesburg has again been visited by supposed earthquake shocks. Now it has been said that it is due to caving taking place in some of our mines, so the mines in the affected area are examined and no caving can be discovered; then some say if that is not so, then it must be earthquake shocks that are felt. It is quite possible, and probable, that there has been no earthquake shock, and yet it is our mines that have been the cause of the shock. With strong roofs, like the roof of our mines, large areas can be opened out without any signs of taking weight for some years, but eventually the roof weight becomes so great that the lower portion begins to rip away from the higher portion and starts "talking," it is this "talking" that is heard and felt; the movement might be so small as to escape notice beyond observing that the loose pieces that hang about some pillars are a little looser. It must be remembered that although the distance moved is so small, yet the weight is so tremendous, it may be a million tons, that we get a shock. I have known of railway trains going over rock outcrops causing vibration in houses half a mile away, and in the case of the above shocks the probable mine should be found by tracing the line of vibration.

This reply is, no doubt, rather loose, but if any member would like to discuss any of the points that may still be undecided, I am quite willing to do anything I can to help him.

The President: We must congratulate Mr. Johnson very much on his reply. It has been a most exhaustive one. He has replied very thoroughly to all his critics, and has raised more controversial points, and he kindly leaves an opening for his critics to make further remarks. That is very good of him, I am sure. His reply has become as valuable as his original paper. He has amplified many matters on which there may have been a good deal of doubt existing, and I think the hearty thanks of our Society are due to him for the excellent paper which he has placed before us.

THE ADAIR-USHER PROCESS.

(Read at May Meeting, 1908.)

By Alfred Adair (Member.)

DISCUSSION.

Mr. A. H. Hartley *(Member)*: For current slimes on the Rand I think it is pretty well proven, by practical every-day results, that the use of umber does not increase the extraction of gold. I think it will be conceded that all the gold in current slimes that can be economically dissolved is dissolved. The successful treatment of slimes on these fields is therefore a question of washing, or separating the gold bearing liquor from the slimes, the recovery of gold from the solution, and of costs. For rich slimes, from, say, 3 dwt. and upwards, there is a great deal to be said for the Adair Usher process and for filter pressing.

For persons immediately interested in current local slimes I think filter presses at present may

be considered much too expensive, both as to original cost, costs for repairs, and ordinary working costs. They are often the direct and irremediable cause of high residues, if sands, however small in quantity, should escape to the slimes plant, or when frames and cloths become worn or broken.

Comparing the Adair-Usher with the decantation process, one can assume that all the gold is dissolved in both cases. (As a matter of fact, days of agitation in contact with cyanide solution with or without aeration will not dissolve the last 2 or 3 grains of gold held in the residues of the average Rand slimes.) Treating, say, a 2 dwt. slimes by the decantation process with two washes of $3\frac{1}{2}$ tons of solution to 1 of slimes, and having a 1 to 1 settlement after each of the two decantations, the residue would contain 0·163 dwt. dissolved gold, equal to 8·15d. per ton. Supposing by using 2 tons of wash to 1 ton of gold bearing solution the Adair-Usher process would completely displace the gold solution from the slimes, and taking the cost for precipitation of gold in zinc boxes at $1\frac{1}{2}$d. per ton of solution, this would come to - $7 \times 1\frac{1}{2}$ or 10·5d. per ton.
As against - $3\frac{1}{2} \times 1\frac{1}{2}$ or 5·25 ,,

Difference	-	5·25	per ton.
Add royalty	-	2·00	,,
		7·25d.	per ton.

To this might be added, possibly, extra cost for pumping. There would certainly be the cost of pumping about $3\frac{1}{2}$ tons of liquid to every ton of slimes treated, in addition to decantation costs. In any case, the extra costs as already noted leave a very small margin in favour of the extra recovery.

These observations, I admit, are somewhat superficial, but as most papers read at our meetings are written with the professed intention of causing discussion with the object of ultimate enlightenment, my remarks on Mr. Adair's paper may lead to the desired result. In the end it is the most effective and most economical method of gold recovery, irrespective of criticism, which appeals to the mining industry in general.

Mr. C. E. Rusden *(Member)*: This is scarcely what may be termed a discussion on the above process, but more in compliance with our late President's request to place before the members some data obtained from the working of the process on a large scale. Therefore I have pleasure in giving you the following from the Geldenhuis Estate & G. M. Co. slimes works.

These figures are for the purpose of comparing the extractions, working costs, etc., taken over the twelve months decantation process directly preceding the installation of the "Adair-Usher" process, and the twelve months following.

During the first period under review 48,064 tons of slimes were treated, containing 4,953·38 oz. fine gold, equal to 2·061 dwt. per ton; out of this amount 4,163·16 oz. fine gold were recovered, equal to 1·732 dwt. per ton, or 84·04% actual extraction. During the second twelve months 59,800 tons of slimes were treated, containing 5,698·05 oz. fine gold, equal to 1·906 dwt. per ton, of which 5,322·295 oz. fine gold were recovered, equal to 1·78 dwt. per ton, or 93·40% actual extraction. This shows a difference of 9·36% distinctly in favour of the Adair-Usher process, and I am sure you will all agree with me that it is a very considerable item when one takes it over the huge tonnage of slimes daily handled on these fields; but what this increased extraction meant to the Geldenhuis Estate was a reccovery of 533·38 oz. fine gold (in value about £2,246) more than we could possibly have expected from the decantation process.

Working costs are also in favour of the Adair-Usher process, as a decrease of 2d. per ton has been effected since its installation as compared with the preceding twelve months. This saving would have been still greater had the costs of installing the process been charged to capital expenditure instead of being included in the working costs. So by these two items alone the sum of £2,744 has been added to the profits of the Geldenhuis Estate during the first year's working of the process.

Increased tank capacity is another advantageous feature of the process, as one can reckon on, at least, a 30% to 33% increase, and there is not the least doubt but that a large number of slimes plants along the reef would derive a great benefit from this source, even if it only gave them a 10% to 15% increase.

A considerable saving of time is effected in the washing, as from 12 to 18 hours is saved per charge. This advantage being especially appreciable during the winter months.

Regarding umber, I have little to say. We experimented with this oxidising agent for a short time, but found we obtained just as good results without using it.

In conclusion, I might say that I think it is due to the Adair-Usher process that these advantages which we have obtained on the Geldenhuis Estate should be brought before the members of this Society, more especially as one of the members, in his discussions on Mr. Adair's paper, seemed to have overlooked the good points and brought to your notice imaginary "defects," "drawbacks" and "objections" before giving it a fair and extended trial.

Mr. Geo. Melvill (*Member*): Being personally connected with the original experiments on a large scale at the Crown Reef Mine, a few notes on the working results of the Adair-Usher process will, no doubt, be of interest.

The first experiments were carried out in a tank 32 ft. diameter, 14 ft. deep, with four revolving arms at the bottom of the tank, connected on to a 9-in. down pipe. These arms were made of 3 in. pipes with 50 ½ in. holes in each pipe. Each charge consisted of about 100 tons. The results obtained were excellent, but the mechanical part of the business was defective, and consequently one could never rely on the charge being completed. This apparatus gave better results than the stationary pipes at present in use, as the solution was more evenly distributed, but the cost of maintenance was too great.

The plant at the Crown Reef mine consists of :—

5 Collecting vats	30 ft. × 10 ft. 0 in.
5 Treatment vats	32 ft. × 14 ft. 0 in.
3 Treatment vats	50 ft. × 13 ft. 0 in.
2 Filters	20 ft. × 5 ft. 6 ft.
2 Filters	24 ft. × 8 ft. 0 in.
1 Solution sump	40 ft. × 11 ft. 0 in.
3 Precipitation boxes	30 ft. × 6 ft. 0 in. × 3 ft. deep.

With this plant I am treating 8,000 tons of slimes per month, and it is capable of doing still more.

The Usher apparatus in the treatment tanks consists of a 10-in. down pipe at the centre of the tank, and projecting 6 ft. above the top of the tank, and sufficient 1 in. pipes radiating from the distributor at the bottom end of the pipe, so that the distance between the pipes at the periphery is not more than 2 ft. In each pipe ⅛ in. holes are drilled at an angle of 30° to the bottom of the tank, with centres varying from 12 in. at the centre of the tank to 3 in. at the periphery. The radiating pipes are kept about 2 in. from the bottom of the tank.

Treatment.—About 100 tons of slimes are collected in each charge, and when settled the charge is transferred to one of the treatment tanks, with solution (·02%) from the next tank to be discharged, care being taken that solution is being introduced in the centre pipe from the time the transfer begins, until the treatment is stopped. After the transfer is completed the charge is circulated for two hours or more if necessary, and washing is continued with precipitated solution until the assay value of the clear solution leaving the tank is about ·20 dwt. or less, when the charge is allowed to settle. The settled solution is then used for transferring another charge. The residue samples are taken by means of a long tin tube (2 in. diameter) with a flap valve at the bottom end. The rate of washing is regulated by the settlement of the suspended slimes and the box and filter capacity, always bearing in mind that the faster the washing the sooner will the treatment be completed.

With 100-ton charges in the smaller size tank (32 ft. × 14 ft.) washing is at the rate of 10 to 15 tons per hour, being completed in about 48 hours, and settlement about 36 hours.

It is very important that every hole in the radiating pipes is clear before transferring a charge. With the small size tanks the holes can be cleared through the centre pipe, but with a 50-ft. tank each pipe has to be flushed out independently, with a strong pressure of water through a 1-in. hose, from the periphery end of the pipe. This takes a little time, but it is time well spent.

We have been using the process on the entire slimes plant for the past twelve months, and the results have certainly been very satisfactory—more actually than theoretically.

The chief benefits derived by the process are :—

(1) Lower residues.

(2) Less dissolved gold retained in residues.

(3) Saving in power and consequently maintenance of pumps.

(4) Reduction in time of treatment, which means increase in capacity of plant.

(5) Less work for shiftmen.

From a charge of 2·2 dwt. the residues are ·3 dwt. as against ·45 dwt. before starting the process; but I consider the greatest saving is in the gold in solution in residues sent to slimes dam.

With the decantation process the final drainings assayed from 10 gr. to 16 gr., and with the Adair-Usher process that solution assays from 2 to 4 gr.. or a saving of 10 gr. per ton of solution. Now for every 100 tons of dry slimes discharged there are about 90 tons of solution, therefore for 8,000 tons of slimes (our monthly tonnage) there are 7,200 tons of solution, which means a saving of 150 oz. fine gold per month.

The pumps run only about two-thirds the time they used to, and consequently the spindles and fans do not require renewing so often, which is a great consideration.

Before installing the process the plant was too small to cope with the tonnage, but now, with one treatment tank less, there is plenty of time for treatment. The charge is completely treated in 72 hours whereas it used to take 100 hours.

Mr. White does not seem very favourably impressed with the process. He has evidently not had much practical experience with it, as there is no doubt the Adair-Usher process is a

decided improvement on the old decantation method.

I would like to point out to the author that there is no supposition whatever on my part regarding the decrease of alkalinity of solution through the use of umber. It is an absolute fact, and if he will come out to the Crown Reef I shall be only too pleased to prove it to him. I did not mention in my remarks that umber was acid, and I agree with him that there is no free acid in it, but I have proved beyond doubt, on a large and small scale, that when it is mixed with an alkaline solution, a loss in alkalinity does occur. I believe Mr. Alexander, cyanide manager, Robinson Gold Mining Co., will agree with me, as he used umber on a large scale, but had to stop it to save his lime bill.

When the author is convinced that a loss of alkalinity does occur, through the use of umber, we shall look forward to an explanation.

REMINISCENCES OF THE EARLY RAND.

(*Read at August Meeting, 1908*).

By M. H. Coombe (Member).

DISCUSSION.

The President: I would like to say that I have brought up the old photograph of Johannesburg, as I promised to do at the last meeting. It will be passed round the room, and I will ask you to be careful with it for it is very old and very tender, having been very much mishandled during the war.

The meeting then closed.

Contributions and Correspondence.

SOME FEATURES OF SILVER ORE TREATMENT IN MEXICO.

(*Read at January and March Meetings, 1908.*)

By W. A. Caldecott, B.A., F.C.S. (Past President).

The following comments upon the above paper have been received from Mr. Bernard MacDonald, President of the Guanajuato Mines Selection Company :—

(1) The only innovation on the old patio process methods was the substitution of a mechanical mixer for the tramping down by horses and mules on the Torta. This was done at the Loreto patio plant of the Real del Monte, which was torn out and a cyanide plant substituted after the Boston Company purchased the property. Fine grinding in preparation of the pulp for treatment by the patio process was recognised as being essential to high extraction. The Mexicans had a saying that this was necessary because "quicksilver has no teeth." By the way, we all have found out that the cyanide solution has no teeth either. Aside from the mechanical features mentioned, there has been no recent change in the patio treatment, the chemistry remaining practically the same, and this is likely to be the case with cyanidation, *i.e.*, the important improvements will be by mechanical means.

(2) The patio process is now only worked at Pachuca, and a few isolated camps, but is being generally abandoned everywhere. Some of the old conservative companies who have been carrying an expensive patio plant on their books as an asset, do not like to destroy that asset and add the cost of a new cyanide plant; hence, for these reasons they are continuing to use the patio process, but, doubtless, when the price of silver advances, they will erect cyanide plants.

(3) The percentage of concentrates in the ores treated by the Guanajuato mills is not uniform, differing both in weight and value in each mill. In the Pastita mill, which is the only one in Guanajuato milling ore exclusively from the Veta Madre vein in depth, the concentrates contain about 50% of the values in about 2·5% of the weight of the crude ore. The Guanajuato Reduction and Mines Co. treat mostly dump ores, which were sorted out by the old miners, and the concentrate from these show lower values than from the ore taken direct from the vein. Other mines in the district vary from each other to a greater or less extent.

(4) The concentrates can be treated successfully by cyanidation if ground fine, but that would necessitate a small special plant, and the smelters are now giving favourable treatment charge for the Guanajuato ores, and this is preventing the companies from erecting special plants for the treatment of the concentrates. I wrote an article showing the results of my laboratory tests on the concentrates by cyanidation. The article appeared in the *Engineering and Mining Journal* of December 23rd and 30th, 1905. This article showed the results obtained then, and these have been verified by numerous subsequent tests.

(5) Antimony and arsenic are practically absent from the Guanajuato ores, and are considered negligible factors in the scheme of treatment.

Mr. E. M. Hamilton writes from London as follows :—

Owing to special circumstances I have only just now been able to study the June number of our

Journal for the current year. I note therein that Mr. S. H. Pearce on discussing Mr. W. A. Caldecott's paper, "Some Features of Silver Ore Treatment in Mexico," attributes to me the idea of using mercuric chloride to aid in the solution of silver from its ores during cyanidation. It is true that I was the first, as far as I am aware, to make use of it on the plant to which he refers, but I derived the idea originally from Mr. H. J. Durant, whom I found using it for this purpose at Minas Prietas, in the State of Sonora, Mexico, eight or nine years ago. In the latter instance I may state that after repeated and careful investigation I decided that there was nothing gained by its use, which was accordingly abandoned.

Queries and Replies.

Concrete Piles in Mills.—I understand concrete piles are being used in mills, on the Rand, in place of pine. Could any member supply me with the component parts, proportions, etc., adopted, and also inform me whether they are proving a success?—L.H.L. (Rhodesia).

Reply.—At the Witwatersrand Deep there are ten stamps arranged with concrete blocks under the mortar boxes. The latter are bolted to 9 in. anvils, secured to the concrete, in which the bolts run in accessible grooves. The composition of the concrete is five parts of stone, three parts of sand, and one part of Portland cement, except in the top 3 ft., in which the stone is reduced to four parts. The concrete piles answer very well and give less vibration; the above-mentioned have been in use about four years. I understand that the whole of the Village Deep mill is erected on concrete piles, whilst in the Knights Central mill, now in course of construction, the anvil block is dispensed with, a mortar box with a thicker bottom being used instead.—A.S.

Obituary.

The death is announced with much regret of Dr. William Ironside Moir, M.B., Ch.B., D.P.H. (Aberdeen), who died suddenly on September 28th at the Johannesburg Hospital. Dr. Moir, who was a brother of Dr. Jas. Moir, Vice-President of the Society, joined the Society in May, 1904.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

Action of Hydrogen Sulphide on Alkaline Solutions of Zinc.—"Zinc sulphide or zinc hydrosulphide precipitated from alkaline solutions by means of hydrogen sulphide is dissolved again if the action of the gas is continued for some time. Hence, in the separation of manganese from zinc by adding an excess of alkali hydroxide, filtering off the manganous hydroxide, and precipitating the zinc in the filtrate by means of hydrogen sulphide, the whole of the zinc sulphide may be redissolved in 15 to 20 minutes, by a rapid current of the gas, and may conceivably be overlooked."—L. W. McCay, *Journal of the American Chemical Society*, 1908, *30*, 376—378. —*Journal of the Society of Chemical Industry*, April 15, 1908, p. 334. (A. W.)

Sulphides and Double Sulphides. — "The mechanism of the technical preparation of cinnabar by the wet process is explained, as follows: When black mercuric sulphide is digested with an alkali sulphide or polysulphide solution, a certain quantity of the double sulphide, $5HgS,K_2S,5H_2O$, is formed. As the temperature gradually rises, this is partially decomposed with separation of red crystalline mercuric sulphide. On now cooling, a further quantity of double sulphide is formed from the amorphous mercuric sulphide, and this again decomposes as the temperature rises. By treating excess of black mercuric sulphide with excess of a dilute solution of potassium sulphide, and allowing to stand in a closed vessel at the ordinary temperature, the variations in temperature during day and night suffice to gradually effect the solution of the whole of the mercuric sulphide and its separation in the crystalline form by means of the reactions just described. It is generally stated that stannous sulphide is soluble in alkali sulphides, and this is utilised for its separation from insoluble sulphides. Actually both the solution and the residue contain tin. In order to avoid error in such separations, it is best to use ammonium polysulphide instead of ammonium sulphide."—A. Ditte, *Ann. Chim. Phys.*, 1907, [8], *12*, 229—277. *Chem. Zentr.*, 1908, *1*, 799—804.—*Journal of the Society of Chemical Industry*, April, 15, 1908, p. 334. (A. W.)

Determination of Tungstic Acid. — "The purpose of this paper is to describe a method devised by the authors, by means of which all evaporations, etc., may be dispensed with, and the time thus occupied saved. This is accomplished by a titration of the amount of a standard ethylamine solution required to dissolve the tungstic acid.

About 1 gm. of the ore, if it is very low grade, 0·7 gm. if it is medium, and about 0·3 gm. if it contains over 40% tungstic acid, is very finely pulverised, and decomposed with 50 c.c. hydrochloric acid and 15 c.c. nitric acid in a 300 c.c. Erlenmeyer flask. In order to decompose a high grade ore completely, it must be distributed evenly over the bottom of the flask and must not be stirred. When the ore is decomposed completely 50 c.c. of hot water is added, and the tungstic acid is allowed to settle a few moments. The liquid is closely decanted through a good filter.

The tungstic acid is then washed by decantation four times with a 5% salt solution, using about 40 c.c. of this hot for each washing. If the precipitate is

closely decanted each time, all free acid will be washed out, as will be shown by letting the last washings drop into a beaker containing water coloured by means of phenolphthalein and a drop of ethylamine solution, to be described later.

The filter should be rinsed between decantations by a stream of hot salt solution from a wash bottle. Salt solution instead of water is used for the washing because it prevents the tungstic acid from assuming the colloidal state and passing through the filter. When the last washing is finished, the filter is removed from the funnel and placed in the original flask containing the bulk of the tungstic acid.

An excess of standard ethylamine is then run into the flask from a burette. In running in the ethylamine the first few cubic centimetres should be allowed to run down the sides of the flask slowly to dissolve any adhering tungstic acid. Any yellow particles on the rim are wiped off with a small piece of filter paper and added to the flask. The filter paper is then broken up into bits with a stirring rod, the rod rinsed, and the flask closed with a rubber stopper. This is done to prevent evaporation of ethylamine which might be great enough to vitiate the results of the analysis. On no account is the flask to be warmed.

When all the tungstic acid is dissolved, which is recognised by the disappearance of all yellow particles, a drop or so of a neutral phenolphthalein solution is added and the excess ethylamine titrated back with oxalic acid solution. The disappearance of the pink colour is not sharp, so it is better to add a drop or two of oxalic acid in excess, so as to be sure the colour is gone, and then bring it back with a drop or so of ethylamine. In this way a sharp end point is obtained. From the known relation of the oxalic acid solution to the ethylamine solution, calculate the number of cubic centimetres of ethylamine used up in dissolving the tungstic acid. From this and the ethylamine standard, the percentage of tungstic acid in the ore is readily calculated.

Pure ethylamine in a 33⅓% solution is on the market (Kahlbaum's preparation is very satisfactory). Weigh out 6 gm. of this and make up to 1 litre.

Oxalic Acid Solution.—Weigh out 2·8 gm. of pure oxalic acid and make up to 1 litre.

Standardisation.—Find the value of 1 c.c. of oxalic acid solution in terms of cubic centimetres of ethylamine solution, using phenolphtalein as an indicator. Run into a beaker 20 c.c. oxalic acid solution. Make acid with sulphuric acid, and titrate hot with a $KMnO_4$ solution of known iron standard."—J. B. Ekeley and G. D. Kendall, Jr.—*Mining World*, May 9, 1908, p. 760. (H. A. W.)

Analysis of Lead Slag.—"In sampling lead slag, the sample should always be chilled. The chilling renders the slag brittle, making it much easier to grind and dissolve. It is chilled either by pouring the molten slag into water, or by taking the sample on an iron rod and plunging it into water.

In lead smelters the slag is assayed daily for iron, lime, silica and lead. It is assayed at intervals for baryta and magnesia, if these metals are known to be present.

Lead.—Make up a lead flux composed of one part borax, two parts potassium carbonate, and three parts sodium bicarbonate. Weigh 10 grams of slag into a 10-gram crucible containing 20 grams of lead flux and 5 grams of argols. Add 1 gram of silver to act as a collector of lead.

Fuse in the muffle, beginning at a low red heat. Keep at this heat until fused, which will take about 20 minutes; then raise the heat for 10 minutes, and pour. Weigh and subtract 1 gram from the weight on account of the silver added. Each 100 milligrams of the remainder will represent 1% lead.

Silica, Lime and Iron.—While the lead is fusing, the silica, lime and iron determinations may be carried forward.

Weigh 1 gram of slag into a No 2 beaker, and add about 25 c.c. of water, and stir. While the slag is in suspension add 10 c.c. of hydrochloric acid; cover the beaker with a watch glass, and place on the hot plate.

Weigh 1 gram of slag into a small evaporating dish, and add about 1 c.c. of water, and stir with a short blunt rod. Add 2 or 3 c.c. of hydrochloric acid; place on a sand bath, and stir constantly while dissolving. As the assay goes to dryness, work the silica up on to the sides of the dish, so that when the assay is dry there will be none in the centre. Allow the assay to dry, and scrape the silica down into the centre of the dish, breaking up all large pieces.

Cool and add a few cubic centimeters of water and about 5 c.c. of hydrochloric acid; cover with a watch glass, and boil. Filter and wash, first with hot water, then wash out the dish with hot dilute hydrochloric acid and pour around the edge of the filter, then wash with hot water again.

Place the precipitate in annealing cup, and burn in the muffle, and weigh as silica. This operation takes about 30 minutes. When it is finished the lead will be ready to pour.

To filtrate add a few drops of nitric acid and boil. Remove from the hot plate, and add ammonia until the iron begins to precipitate, then add a hot solution of oxalic acid until the iron precipitate is dissolved. Repeat this operation three times, finally adding an excess of oxalic acid.

Allow the precipitate to settle in a warm place, and filter. Wash five times with boiling water to remove all traces of oxalic acid. Place the precipitate in an annealing cup, burn in the muffle, and weigh as lime.

Another method in common use is to dissolve the precipitated calcium oxalate in dilute sulphuric acid. Boil and titrate with a standard solution of potassium permanganate.

The slag in the No. 2 beaker is now dissolved and is ready for treatment. Add a few drops of stannous chloride until any yellow color present disappears; then about 25 c.c. of a saturated solution of mercuric chloride, and titrate with a standard solution of potassium bichromate.

If barium is to be determined in the insoluble residue, burn and weigh in a platinum crucible. Moisten with water and add about 1 c.c. of sulphuric acid and 2 or 3 c.c. of hydrofluoric acid, and evaporate to dryness. Ignite and weigh as barium sulphate. The weight of the barium sulphate subtracted from the original weight of the insoluble residue will be the silica.

If a platinum crucible is not to be had, burn the insoluble residue in a porcelain crucible. After weighing, fuse with 2 or 3 grams of sodium bicarbonate. Place a scorifier in the muffle upside down and put the porcelain crucible on it. This will keep the bottom of the crucible clean. Allow fusion to remain in the muffle for 30 minutes to cool and dissolve the fusion in hot water, and filter and wash until the washing show no precipitate with barium chloride.

Dissolve the barium carbonate on the filter with dilute hydrochloric acid, and wash until the washings show no precipitate with sulphuric acid. Boil the filtrate and precipitate with a hot solution of sul-

phuric acid. Allow to settle in a warm place, filter, wash, burn, and weigh as barium sulphate.

Magnesia is determined in the filtrate from the calcium oxalate precipitate. Boil the solution, and precipitate the iron with ammonia, boil, and filter. Wash with hot water, and invert the funnel, and wash the ferric hydrate precipitate back into the beaker. Dissolve in hydrochloric acid and again precipitate with ammonia. Boil and filter through the same filter and wash with hot water.

Cool the filtrate to as low a temperature as possible, and precipitate the magnesia with sodium-ammonium phosphate. It may be necessary to stir quite violently. When the magnesia begins to precipitate, set the beaker aside and allow to stand over night.

Filter and wash with hot water containing ammonia. Burn in the muffle, and weigh as magnesium pyrophosphate. The weight of the precipitate multiplied by 0·36036 will give the weight of magnesium oxide."—EVANS W. BUSKETT.—*Mining World*, May 9, 1908, p. 754. (H. A. W.)

DETERMINATION OF NICKEL.—"For the separation and determination of nickel in the presence of any proportions of cobalt, iron or manganese, the following method is presented in *Comptes Rendus*, vol. 145, 1907, p. 1,334.

Treat the ore with acids until all metallic parts are in solution, precipitate the alkaline earths by adding ammonium sulphate, filter, and concentrate the filtrate. Add to it a large excess of saturated solution of ammonium molybdate, then ammonium chloride, and heat for several minutes at 80 or 90° C. Cool, preferably with ice, stirring all the time. The precipitate contains all the nickel as nickel-ammonium molybdate, and nearly all the iron; all the manganese and cobalt, and a trace of iron, remains in solution. Filter, wash with saturated ammonium chloride solution, and then put the filter and precipitate into a beaker. Treat with boiling water, add ammonium chloride, then ammonia, and filter. All the iron is precipitated while nickel remains in solution.

The nickel can then be determined electrolytically, after precipitating it with an alkali and redissolving in sulphuric acid." — *Engineering and Mining Journal*, May 2, 1908, p. 910. (G. H. S.)

THE SOLUBILITY OF POTASSIUM CHLOROPLATINATE.—In a series of observations carried out by the authors the values found were considerably lower than those of previous investigators. Those found now along with the values given by Bunsen are shown below:—

Temperature °C.	Authors.	Bunsen.
0	0·4784	0·74
10	0·5992	0·90
20	0·7742	1·12
30	1·000	1·41
40	1·355	1·76
50	1·865	2·17
60	2·444	2·64
70	3·055	3·19
80	3·711	3·79
90	4·360	4·45
100	5·03	5·18

The author's values give a continuous curve. The solubility in methyl and ethyl alcohols was also determined, and it was found that the solubility of the salt in alcohol water solutions varies in a regular manner with the amount of alcohol present, decreasing gradually as the percentage of alcohol increases. In potassium chloride solutions the solubility of the chloroplatinate decreases with the increase in concentration of the potassium chloride until a concentration of 1 gm. molecule per litre is reached. Beyond this point the increase in concentration of the potassium chloride has practically no effect. The solubility in NaCl solutions increases rapidly until a concentration of ·05 gm. molecules per litre is reached. For more concentrated solutions the increase in solubility is small and almost proportional to the increase in concentration of the sodium chloride.—G. S. JAMIESON, L. H. LEVY and H. L. WELLS.—*Journal of the American Chemical Society*, xxx., 5, May, 1908, p. 747. (J. A. W.)

METALLURGY.

BOILING POINTS OF METALS.—"A paper by Dr. Oliver P. Watts dealt with 'the metals in order of their boiling points as arranged from Moissan's experiments in the distillation of metals and alloys.

Dr. Watts gives a concise summary of the various determinations made by Moissan, and finally presents his attempt to arrange the entire series of metals treated in order of their boiling points. This is a difficult task to accomplish satisfactorily, and probably an impossible one to carry out with complete accuracy. In several cases the difference between the weights of two metals distilled is within the limit of experimental error. The specific heats and unknown latent heats of vapourisation may also cause errors in the tabulation, as the writer could not make allowance for these factors. The only known points in the series are the boiling points of zinc, 940° C., and copper, 2,100° C., as determined by Féry. The boiling point of tungsten was then assumed to be 3,700° C., and the other metals were arranged between these limits.

The number standing opposite each metal is not claimed to be its boiling point, but only as an indication of its relative position in the series which begins with zinc at 940° C., and ends with tungsten at some unknown temperature above the melting point (3,200° C.).

	°C.		°C.
Zinc	940	Titanium ...	2,700
Cadmium ...	1,025	Rhodium ...	2,750
Lead	1,250	Ruthenium ...	2,780
Silver	1,850	Gold	2,800
Copper	2,100	Palladium ...	2,820
Tin	2,170	Iridium... ...	2,850
Manganese ..	2,200	Osmium ...	2,950
Nickel	2,450	Uranium ...	3,100
Chromium ...	2,500	Molybdenum ...	3,350
Iron	2,600	Tungsten ...	3,700
Platinum ...	2,650		

—O. P. WATTS.—*Electro-Chemical and Metallurgical Industry*, November, 1907. (J. A. W.)

SLIMES FILTER.—"A number of flat filter boxes, or trays, are arranged in a circle around a reaction-wheel which forms a central rotating distributor. The pulp to be filtered is fed into a hopper at the top of the wheel, which, as it rotates fills tray after tray with pulp. Vacuum is applied to the interior of the trays and draws off the liquid, and when a sufficiently thick cake has formed, the supply of pulp is stopped and wash-liquor is delivered on to the surface of the cake from another arm of the central rotating distributor. When washing is finished, the supply of wash-water is stopped, and the cake is drained dry by the vacuum, after which the vacuum is shut off and the filter tray reversed. Compressed air is then admitted and the cake of residue is blown off the filter into a receptacle placed below. All these

operations follow one another automatically, the duration of each being regulated by the speed at which the distributor rotates. The shutting and opening of the vacuum and compressed air valves and the reversing of the filter trays are performed by levers attached to tappets which are operated by arms projecting from the central distributor and coming into action as it rotates."—H. D. FITZPATRICK, Glasgow. From H. M. LESLIE, Marrikuppam, India, Eng. Pat., 13,569, June 12, 1907.—*Journal of the Society of Chemical Industry*, March 16, 1908, p. 214. (W. A. C.)

CHLORIDE OF BARIUM FOR HARDENING.—"While the lead bath can be used for heating high-speed tools, and may be very useful in the case of small ones, it does not prevent entirely the trouble of oxidation. None of the other metals available for metallic baths are any more suitable.

A partial solution of the problem was found when barium chloride was tried. The chloride melts at a temperature somewhat below that best adapted to hardening high-speed steel, and can readily be raised to the proper temperature in a suitably designed furnace. A piece of steel when heated in a bath of barium chloride and then withdrawn is coated with a thin film which effectually preserves it from contact with the air until it is plunged into the cooling bath or cooled in the air blast.

The process has recently been taken up and developed by the Firth-Sterling Steel Co., U.S.A.

As now used it is exceedingly simple. Commercial barium chloride, which sells for about 1½d. a pound, is melted in a graphite crucible and brought up to a temperature of about 2,150° F. in a gas furnace. A small proportion of sodium carbonate (soda ash) is added in order to prevent fuming, and the bath is ready for use."—*Engineering Review*, June, 1908, p. 387. (A. R.)

MINING.

RISE OF EARTH TEMPERATURE.—"M. Durnerin has recently communicated to the Société de l'Industrie Minérale the result of his observations on temperature taken during some deep boring operations in Meurthe-et-Moselle. The principal difficulties in the observation of the earth's temperature are currents of underground water, the admission of outside water into the borehole, and the heat produced by oxidation which goes on when the hole passes through carboniferous or pyritiferous strata. He found that 1° of temperature was gained every 53 metres in Triassic sandstones, and every 16·5 to 20 metres in carboniferous conglomerates having a shaly matrix. These results confirm the previous knowledge that the rise in temperature is more rapid in the older strata. It is probable that in the future development of the deep coal measures of Lorraine, relatively high temperatures, necessitating, perhaps, the use of artificial cooling apparatus, will be encountered."—*Science and Art of Mining*, June 27, 1908, p. 529. (A. R.)

SAFETY APPLIANCES IN WINDING.—"How to prevent damage and loss of life in coal-winding accidents occupied a good deal of attention at a recent meeting of the Manchester Geological and Mining Society. Mr. Leonard R. Fletcher (Messrs. Fletcher, Burrows and Co.) described the patent keps in use under the cages at Chanters Pit, Atherton collieries. The chief advantages claimed for these keps are, he said, (1) allowing the withdrawal of the keps from under the cage with the full load of the cage upon them, so that the engineman has no need to reverse his engines for banking purposes; (2) consequent saving of labour to the engineman in manipulating the winding engines; (3) consequent saving of time and steam; and (4) reduction of wear and tear on the winding appliances. 'After three years' experience with a set of these keps at this pit,' Mr. Fletcher added, 'I can confidently say that these claims are justified in actual practice. Since these patent keps have been put in at Chanters No. 1 Pit three-deck cages have been substituted for the two-deck cages formerly in use, and the advantage gained by the use of these keps has been most pronounced.' Mr. T. H. Wordsworth, of the New Moss colliery, followed with a description of Lee's patent safety appliance for cages. A model of the appliance was exhibited. The purpose of the inventor (one of the workers at the colliery) is to hold the cage in the shaft if the winding rope breaks where wire conductors are used. Although, Mr. Wordsworth said, it has been argued that safety appliances of this description were better left unapplied, he thought it would be quite worth the expense of applying a catch of this sort if it only operated once in a lifetime, as the upkeep was infinitesimal.—Mr. Joseph Hindley and Mr. John Stoney afterwards exhibited another patent safety appliance for use in winding. In this case it was claimed that the appliance would hold an ascending cage in the place where it happened to be when the rope broke. If the cage were descending at a rapid rate at the time of the rope breaking the arrest of the falling cage would be gradual."—*Science and Art of Mining*, June 27, 1908, p. 530. (A. R.)

USE OF EXPLOSIVES. — "Mr. A. H. Merrin, M.C.E., Chief Inspector of Mines, in his report to the Victorian Mines Department, gives the following to secure effective results with nitro glycerine compounds, preventing poisonous fumes and missfires :—

1. Use an efficient cap. Select for the particular kind of explosive, the cap (detonator) with the proper charge of fulminate—*i e.*, the explosive compound contained in the cap. The grade of cap to be on the strong rather than on the weak side. The longest charge requires the strongest cap; a weak cap will leave hole unbottomed. The cap to be freed from sawdust by gentle tapping. Used under water, the upper end of cap where it joins the fuse to be made water-tight (grease, pitch, clay, etc.). The cap deteriorates by exposure to damp air.

2. Select the right fuse for the kind of work. The fuse to be cut clean across and not slanting. The powder not to be shaken out of end of fuse. Fuse to be inserted in cap until it reaches the fulminate. The upper part of cap to be crimped with a broad-faced tool and not grooved so as to choke the fire in the fuse. With fuse too ragged or too large to enter cap, the end to be swaged to proper size by broad crimper. The fuse should not be kept in a damp place.

3. Do not bury cap in primer, so as to cause the burning of latter. The cap to be pushed into primer, the fuse not touching the explosive, and securely tied in that position. The primer to be pushed home with a wooden rammer into contact with charge.

4. Allow no break in the contact of plugs, due to presence of borings, or to careless charging. The plugs to be pressed home firmly with wooden rammer to fill up spaces round charge and so completely fill the hole the proper depth.

5. Tamp the charge sufficiently with clay, dry sand, or borings. Tamping not to contain any sharp particles that would damage the fuse.

6. Do not charge the hole with plugs damaged by storage in a damp atmosphere. The explosive to be stored in a dry and cool place.

7. Do not charge the hole with frozen or partially frozen plugs. The plugs to be soft and plastic. A stronger cap to be used under cold conditions.

8. Do not overcharge the hole."—*The Australian Mining Standard*, July 15, 1908, p. 64. (W. A. C.)

DUST IN COAL-MINING.—"In connection with the important matter of dust in coal mines, says *Engineering*, Mr. B. F. Jones, of the Westmoreland Coal Co., Irwin, Pa., U.S.A., has recently supplied figures of tests made at their mines to *Mines and Minerals*. Compressed air machines were used, and the results of cutting with the hammer-type drill, with chain-machine, and by hand were compared. All the cuttings were passed over a No. 16 screen having 256 meshes per sq. in., and then over a No. 40 screen of 1,600 meshes per sq. in. Reduced to the same basis the hammer-type drill produced cuttings to the amount of 10·95% of the coal mined, the chain-cutter 5·86%, while hand-mining produced cuttings to the amount of 14·45% of the coal mined. The cuttings and dust that passed through a No. 16 screen amounted to 2·4% of the coal mined in the case of the hammer drill, 1·06% for the chain machine, and 1·11% in hand-mining. The dust passing through the No. 40 screen amounted for the hammer-drill, to 1·25% of the coal mined, 0·494% for the chain machine, and 0·408% by hand-mining. According to these results, therefore, the pick chain-cutting machine makes practically as little dust as hand-working."—*Indian Engineering*, July 4, 1908, p. 13. (A. R.)

MISCELLANEOUS.

MOISTURE, AND THE STRENGTH OF WOOD.—"Very little being known about the influence of moisture on the strength of wood, the United States Forest Service made some time ago a thorough study of the question. The results of its investigations, observes the *Engineering Times*, are interesting and instructive. It has been found that the relation of moisture to strength follows a definite law. The strength of all kinds of wood increases rapidly with proper drying, the amount of increase depending on the species and the degree of dryness. Thus the strength of a piece of unseasoned red spruce may be increased over 400% by a thorough drying at the temperature of boiling water. But the strength decreases again as the wood reabsorbs moisture. Air-dried wood protected from the weather, and containing 12% of moisture is, according to species, 1·7 to 2·4 times stronger than when green. Drying also increases the stiffness of wood. These conclusions have been drawn from pieces of small cross-section, not exceeding 4 in. × 4 in. Large timber requires years of drying before the moisture is reduced to the point at which the strength begins to increase. It has been found that, under normal conditions, wood fibre will absorb a definite amount of moisture. Additional water only fills the pores. It has also been found that the water which simply fills the pores has no effect on the strength. The fibre saturation point is :—For long leaf pine 25, red spruce 31, chestnut 25, red gum 25, red fir 23, white ash 20·5 ; Norway pine 30%, estimated on the dry weight of the wood. Timber that has been dried and resoaked is "slightly" weaker than when green." — *Indian Engineering*, March 14, 1908, p. 175. (A. R.)

Reviews and New Books.

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

THE ELECTRIC FURNACE, ITS EVOLUTION, THEORY AND PRACTICE. By ALFRED STANSFIELD, D.Sc., A.R.S.M., Professor of Metallurgy in the McGill University, Montreal. 53 illustrations. Price, 2 dolls. (Toronto, Canada: *The Canadian Engineer*.)

"Although a great deal has been written recently about the different types of electric furnaces and their industrial application, the information available is scattered through technical periodicals published all over the world, and no book giving the gradual development, from the first experimental furnace constructed by the late Sir William Siemens in 1878 to the present modern furnace now employed on an industrial scale, has been available, either for the student or for the technical man interested in the subject. It is particularly fitting that the task of summarising all the most recent information, and the discussion of the future possibilities of the eletric furnace, should have been undertaken by the Professor of Metallurgy of McGill University, as Canada, with its vast undeveloped waterfalls, has very special reasons for being exceptionally interested in the economic application of electrical energy to industry. The book, after a short history of the subject in the first three chapters, gives a very clear description of the various types of furnaces at present in use, and discusses the efficiency of the different types, and of electric furnaces generally, as compared with other furnaces. The design, construction, and operation of the furnaces are described, and one chapter is devoted to the production of iron and steel. In this chapter there is a very good general description of the principal furnaces actually in use for the production of steel, together with the most recent results obtained with experimental furnaces in Canada and elsewhere. Chapter VI. deals with the utilisation of the electric furnace for the production of alloys and general metallurgical work, as well as the various other purposes for which it has been employed on an industrial scale; and the final chapter is devoted to the discussion of the probable industrial development of the electric furnace in the near future. The book, which, though it contains only 195 pages, is well illustrated, should be very useful to the technical man or student, who, without having the time to go deeply into the subject, wishes to obtain a general knowledge of the present position of the electric furnace and of the extent to which it is being employed in our various industries. It is a very clear, concise summary and digest of what has been done, and what is at present being done, on a commercial scale with various types of electric furnaces."—*The Mining Journal*, July 25, 1908, p. 117. (W. A. C.)

ELECTRO - METALLURGY. By John B. C. KERSHAW. Price, 6s. (London: A. Constable & Co., Limited.)

"The author's aim in this book has been to give a brief and clear account of the industrial developments of electro-metallurgy in language that can be understood by those whose acquaintance with either chemical or electrical science may be but slight, and there can be no doubt that he has been eminently successful. We have very great pleasure in cordially

recommending Mr. Kershaw's work to all those who wish to be *au courant* with the general scope of electro-metallurgical practice, and especially with its latest applications. The book presents in a compact form the essential features of a most important department of technology, and will be found to be a most valuable addition to the library of all who are interested in any way in applied science. The book deals with the electrolytic refining of gold, silver, and copper, and with the refining and extraction of lead, nickel, and zinc; with the electrolytic tin stripping processes; with the electric smelting of aluminium; and, as a welcome addition to existing literature, with the production of calcium carbide and cyanamide, carborundum, graphite, quartz glass, ferro-alloys, and iron and steel, by the application of the electric current. The arrangement of the book is all that could be desired, and the illustrations are excellent."—*The Mining Journal*, July 25, 1908, p. 117. (W. A. C.)

REPORT BOOK FOR MINING ENGINEERS. By A. G. CHARLTON. Second edition, revised and enlarged. Price, 7s. 6d. net. (Published by Whitehead, Morris & Co.)

This useful pocket-book, the outcome of 27 years' experience in mine inspection and mine management in various parts of the world, has been compiled with the object of supplying a systematic method upon which to base reports and preserve a record in a condensed form for speedy reference. The book is arranged in skeleton form, blank spaces being left under comprehensive headings and against numerous searching questions to be filled in in accordance with the results of investigations. The information thus asked for covers the field of mining and reduction in such a thorough manner that nothing of any importance can well be overlooked. There are many useful tables and data included, and altogether the work will be found of great value to any mining engineer, who may be called upon to furnish a report on a property. (A. R.)

PEAT AND LIGNITE, THEIR MANUFACTURE AND USES IN EUROPE. By E. NYSTROM. M.E. Canada, Department of Mines, E. Haanel, Director.

This is the latest adition to the excellent series of monographs on special subjects, which is being published by the Canadian Mines Department under the energetic administration of its Director, Dr. Haanel, and readers may be reminded of the reviews which have already appeared in these columns of two published a year or two ago on mica and asbestos. The present volume attains to the high standard of its predecessors, and is practically an exhaustive treatise on the subject with which it deals. The illustrations are especially to be commended, and add considerably to the value of the book. Mr. Nystrom is to be highly congratulated on having so faithfully fulfilled his mission, and the book will be invaluable to those who are engaged in the manufacture and utilisation of this fuel. (J. A. W.)

DESIGN AND EQUIPMENT OF SMALL CHEMICAL LABORATORIES. By R. K. MEADE. 10s. nett. (Published by Chem. Engineering Publishing Co., Chicago.)

The scope of this work is amply and succinctly covered by the title. To the chemist anxious to bring his laboratory up to date, the work is valuable in that modern methods and apparatus alone are dealt with. The fixing up, arrangement and selection of working apparatus for a small scale working laboratory are clearly detailed, and it is needless to say, the assay department is not forgotten. One notes with pleasure the latest designs in crushers, fine grinders, mechanically operated agate pestle and mortar, smelting and muffle furnaces, and cupel making machines. His opening sentence might well be followed with advantage by all engaged in chemical work on these fields. It reads—"Where space permits, have a separate table for carbon combustions, for ignitions, extractions, evaporations, for titrations, precipitation with H_2S, distillations and for electro-chemical analysis, filtrations and various and sundry special tests." A well-written, readable and clearly illustrated book, worthy the perusal of all chemists and assayers. (A. McA. J.)

Selected Transvaal Patent Applications.

RELATING TO CHEMISTRY, METALLURGY AND MINING.

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

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

(P.) 333/08. William Griffith Williams. Improvements in means for heating, melting or volatilising. 18.8.08.

(P.) 334/08. George Smith. Improvements in apparatus for disposing of sand or such like material on mine dumps or other depositing sites. 18.8.08.

(P.) 335/08. Henry Makins Tait. Improvements in safety catch devices. 18.8.08.

(P.) 336/08. Robert Hutchinson Anderson. Improvements in rock drills. 20.8.08.

(C.) 337/08. Paul Sabatier. Improvements in and relating to manufacture of methane or of mixtures of methane and hydrogen. 21.8.08.

(C.) 339/08. Charles Christiansen. Improvements relating to hammer rock drills. 21.8.08.

(P.) 340/08. Allan McGregor Ritchie (1), David Cunningham McIlleron (2). An improved pole or post used for fencing or the like. 21.8.08.

(C.) 341/08. Samuel Victor Man (1), Adolphus Pillman (2). Improvements in means for purifying air. 21.8.08.

(P.) 343/08. Robert Hunter (1), Robert Leslie Polson (2). Improvements in acetylene gas generators and lamps. 24.8.08.

(C.) 344/08. Henry Samuel Potter (1), Frederick Dwight Johnson (2). Improvements in or relating to fluid pressure operated rock drilling apparatus and the like. 24.8.08.

(P.) 345/08. Charles Ward Hammerton (1), Thomas Hugh McDonald (2), Frank Porson Neve (3). Improvements in and relating to electrical signalling systems for mines. 26.8.08.

(P.) 346/08. Charles Hansen. Improvements in safety brake arrangements for mine cages and other hoisting apparatus. 27.8.08.

(P.) 347/08. James Grant Gibson. Improvements in linings for tube mills. 27.8.08.

(P.) 349/08. James Snodgrass. Improvements in the purification of water. 29.8.08.

(C.) 350/08. Walter Neal. Improvements in pebble retaining outlet for tube mills. 31.8.08.

(P.) 351/08. Herbert Lewis Lezard. Improvements in spanners. 1.9.08.

(P.) 352/08. Frederick Thomas Watt. Improvements in machines for crushing or disintegrating ore and other substances. 1.9.08.

(P.) 353/08. Carl Frederick Siegert. Improvements relating to the treatment of auriferous black sand concentrates and the like for recovery of the precious metals. 1.9.08.

(P.) 354/08. John Holgate. Improvements in reciprocating engines. 2.9.08.

(P.) 355/08. William Joseph Holmes. Improved portable ground surface pulveriser and detachable teeth for same. 2.9.08.

(C.) 356/08. Frederick Adamson. Improved diamond washer. 3.9.08.

(P.) 358/08. Frederick Victor William Swanton (1), Thomas Arnott Fletcher (2). Improved apparatus to produce power by the combined action of certain fluids of dissimilar elasticity and weight upon same. 3.9.08.

(P.) 360/08. John Francis Weedon. A new or improved reinforced concrete sleeper for railways and tramways. 7.9.08.

(P.) 361/08. James Catford Koller. Improvements in the extraction of metals from their ores and apparatus therefor. 8.9.08.

(P.) 362/08. William Thomas. Improvements in means for fixing tappets to the stems of battery stamps, also applicable for fixing collars or bosses to shafts and the like. 8.9.08.

(P.) 363/08. Henry Alford Walker (1), George Walter Compton (2). Improvements in the manufacture of earthenware pipes. 8.9.08.

(P.) 364/08. William Arthur Caldecott. Improvements in or relating to means for separating liquid from crushed ore products. 9.9.08.

(P.) 365/08. James Ralph. Improvements in rollers for supporting haulage ropes and the like. 9.9.08.

(P.) 366/08. Anastisis Maronsis. Improved liquid elevating device. 9.9.08.

(P.) 367/08. Louis Thomas Dechow. Improvements in slime filtering devices. 9.9.08.

(P.) 368/08. Herman Georg Spengel. Improvements in rock boring machines. 9.9.08.

(P.) 369/08. Ernest Albert Brinkmann. Tube mill liner. 10.9.08.

(C.) 370/08. Matthew Yarrow. Improvements in joints of pipes or mains. 11.9.08.

(C.) 371/08. Albert Hibbs Taylor. Improvements in hammer drills. 11.9.08.

(C.) 372/08. Albert Hibbs Taylor. Improvements in hammer drills. 11.9.08.

(C.) 373/08. Albert Hibbs Taylor. Improvements in hammer drills. 11.9.08.

(C.) 374/08. William Prellwitz. Improvements in unloaders for fluid compressors. 11.9.08.

(C.) 375/08. William Prellwitz. Improvements in unloaders for compound fluid compressors. 11.9.08.

(C.) 376/08. William Prellwitz. Improvements in brace attachments for fluid operated tools. 11.9.08.

(P.) 377/08. Alexander Purser. A new and improved machine for drilling rock and the like. 11.9.08.

(P.) 378/08. James Jameson. Improvements in means for connecting trucks or other vehicles to a hauling rope. 12.9.08.

(P.) 379/08. Alfred Roach. Improvements in smelting, assay, and like furnaces. 16.9.08.

(P.) 380/08. Arthur William Davis. Device for concentrating and recovering metals from alluvial wash and the like. 16.9.08.

(C.) 381/08. Charles Tingley Carnahan (1), Jeremiah Murphy (2). Water attachments for pneumatic hammers. 18.9.08.

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