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OF SOUTH AFRICA.

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VOL. XI. NOVEMBER, 1910. No. 5.

Proceedings

AT

Ordinary General Meeting, November 19, 1910.

The Ordinary General Meeting of the Society was held in the Lecture Theatre of the South African School of Mines, on Saturday evening, November 19th, Dr. James Moir (President) in the chair. There were also present :—

43 Members : Prof. G. H. Stanley, Messrs. C. B. Saner, W. R. Dowling, R. Allen, K. L. Graham, Tom Johnson, E. J. Laschinger, H. A. White, Prof. J. A. Wilkinson, S. Beaton, W. Beaver, A. J. Bowness, G. L. Burnett, J. Chilton, M. H. Coombe, T. Donaldson, R. Gascoyne, W. W. Hesom, A. B. Inglis, A. E. Irwin, J. H. Johnson, A. J. Johnson, G. A. Lawson, J. Lea, W. D'Arcy Lloyd, H. S. Macgregor, F. A. McCoy, P. T. Morrisby, E. Pam, J. F. Pyles, A. Redfern, E. Roberts, G. A. Robertson, A. H. Scarf, G. H. Smith, S. H. Steels, Ralph Stokes, W. A. C. Tayler, A. Thomas, C. Toombs, C. F. Webb, and E. M. Weston.

20 Associates and Students : Messrs. M. S. Archibald, W. E. Burrows, F. C. Carbis, G. J. V. Clarence, J. Cronin, C. N. Davies, C. L. Dewar, W. J. N. Dunnachie, J. Gibson, O. Harrison, B. W. Holman, A. King, L. T. Leyson, S. H. Olivier, F. J. Pooler, H. B. Powter, T. L. Thorne, W. Waters, F. Wartenweiler and E. J. Wiseman.

18 Visitors, and Fred. Rowland, Secretary.

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

NEW MEMBERS.

Messrs. W. Beaver and E. J. Laschinger were appointed scrutineers, and after their scrutiny of the ballot papers, the President announced that all the candidates for membership had been unanimously elected, as follows :—

CREASEY, WALTER JOHN, P. O. Box 1112, Johannesburg. Cyanider.

GIDDEY, CLAUDE LESLIE, East Rand Proprietary Mines, Ltd., P. O. Box 66, East Rand. Cyanider.

THOMAS, ARTHUR, Durban Roodepoort Deep, Ltd., P. O. Box 110, Roodepoort. Shift Boss.

WELLS, ERIC FRANCIS VESEY, Village Deep, Ltd., P. O. Box 1145, Johannesburg. Mine Captain.

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

As Associates.—

CHESTER, WILLIAM, P. O. Box 463, Germiston. Amalgamator.

FILMER, Jr., HARRY JOSEPH, P. O. Box 1409, Johannesburg. Assistant Amalgamator.

FREEMAN, CHARLES CUTHBERT, Great Fingall Consolidated G. M. Co., Ltd., Day Dawn, Western Australia. Metallurgist.

MCDIVETT, DAVID, Geldenhuis Deep, Ltd., Cleveland. Cyanider.

As Students.—

COWLES, EUGENE POMEROY, Simmer Deep, Ltd., P. O. Box 178, Germiston. Reduction Works Learner.

MCLEAN, CALVIN STOWE, Simmer and Jack Proprietary Mines, Ltd , P. O. Box 192, Germiston. Tube Mill Learner.

GENERAL BUSINESS.

The President : I have a letter of apology from Mr. Stuart Martin, who regrets he is unable to attend this meeting through illness.

I have also to state that our Committee on Mining and Metallurgical terms has extended the date for suggested alterations to the end of this month. In the meantime we have received a letter from the Institution of Mining and Metallurgy stating that it is going to reopen the question. I think the subject may as well be closed at this end, and that Institution, with its greater opportunities, be allowed to complete the matter finally.

A third matter which I have to mention is the address to the Duke of Connaught from the ten local Societies, a photograph of which you may see here. The original has been finished and signed, but unfortunately it is not visible, as it has to be bound in leather and will only be ready at the last moment.

CYANIDE AS AN INSECTICIDE.

Prof. J. A. Wilkinson (*Member of Council*): In order to prevent any misconception which might otherwise possibly arise in connection with an abstract recently inserted in the *Journal*, and mentioned by Mr. Thomas at the last meeting, I beg to state that the procedure adopted by the Council in such matters deprives abstractors of any responsibility in publication. Further, one cannot be held responsible for the experiments and opinion of others. As cyanides are used so largely here, I thought that the note would prove of interest to members. Unfortunately I came in somewhat late at the last meeting and did not know that this matter had been mentioned until the printed copy of the *Journal* came to hand.

CHEESA* STICKS.

Mr. T. Donaldson read the following contribution on behalf of the author:—

Mr. W. Cullen (*Past President*): At the last meeting of the Society I mentioned that Messrs. Brock, of London, had introduced a cheesa stick which seemed to do very well in practice. Since then further experience has been gained with it, and as a result some small modifications were made which are embodied in the samples which are exhibited. A quantitative analysis of the ingredients has not been made yet, but they consist of chlorate of potash, barium nitrate, magnesium and shellac. In order to make the ignition certain and easy, a little touch paper is wound round the working end. The stump, or handle, is composed of about two inches of wood, which enables the stick to be burned right down to the bottom.

In order to ascertain whether the gases were noxious or not, ·4 gm. of the cheesa stick mixture was ignited by electric means in a gas tight bottle of air of 3,300 c.c. The resulting gases had the following composition:—

	Per cent.
Oxygen	19·27
Nitrogen, etc.... ...	78 62
Carbon monoxide ...	·20
Carbon dioxide ...	1·91
Nitrogen, oxides of ...	Nil

As will be seen, they are very low in carbon monoxide. In order to ascertain the probable vitiation of the mine air underground, taking the composition of pure air to be:—

	Per cent.
Oxygen	20·90
Nitrogen	79·06
Carbon dioxide ...	·04

and working with the analysis given above as a basis, the following has been calculated. If one of these cheesa sticks, containing 18 gm. of combustible mixture be completely burned in an enclosed space filled with air and measuring 6 ft. × 6 ft × 8 ft., *i.e.* with a capacity of 288 cub. ft., the air afterwards would have the following composition:—

	Per cent.
Oxygen	20·87
Nitrogen	79·05
Carbon monoxide ...	·0035
Carbon dioxide ..	·07

One of the advantages of this cheesa stick is that it can be burned right to the end. Supposing only one inch of it is necessary to set off a round, the remainder can be kept for further use. If used in this way it comes out quite as cheap, if not cheaper than the spit fuse, and the advantages over the blasting gelatine stick are obvious.

Once ignited, the stick is extremely difficult to put out, and it even withstands the concussion of the underground blast. It burns for 6 minutes 40 seconds to 6 minutes 45 seconds, but of course there are slight variations as between stick and stick. The flame, as you can see, is bluish, and shows up very well underground. There is only one further improvement which has been suggested, and it will be given effect to, *viz.* the waterproofing of the paper wrapping, which would make it better able to withstand the underground conditions.

The President: I am sure we all thank Mr. Cullen for his interesting contribution.

Mr. E. J. Laschinger (*Member of Council*): I was talking to one of our Past-Presidents, Mr. S. H. Pearce, and it occurred to both of us that it might be a very good idea to try the lighting of fuses with an ordinary blow lamp as used by plumbers. If a special lamp were constructed of a convenient size, we thought it would be a good thing to try. We know, of course, that the composition of the gases from such a light would be free of noxious fumes or any really dangerous constituents.

The President: It has a blue flame, has it not?

Mr. E. J. Laschinger: Yes. I bring it up as a suggestion as to whether it would not be advisable to try it, or if it has been tried the results might be ascertained.

Mr. H. A. White (*Member of Council*): I have to point out that that kind of lamp is on the same principle as the Primus stove, which was so strongly condemned by the Society a little while ago.

Mr. E. J. Laschinger: It is very much more convenient and gives a strong heating flame. I am certain from my own experience of lighting

* Cheesa—a phonetic spelling of *tyisa* or *tshisa*, Zulu for hot or burn; cheesa stick literally in this instance firing stick—a substitute for spit fuse, commonly in use in South Africa.

fuses, although I have not tried this lamp myself, that a blow lamp would ignite the fuses as well as any cheesa stick ever invented.

Mr. H. A. White: Personally I have no objection to using a Primus lamp, but this Society has practically condemned the use of them.

Mr. E. J. Laschinger: In the hands of the inexperienced ordinary public.

Mr. H. A. White: Or miners!

Mr. M. H. Coombe (*Member*): I noticed that the moment water was applied to the cheesa stick by Mr. Donaldson it was extinguished. That certainly renders it impracticable for use underground, in wet winzes, shafts or faces generally. It could only be used in a very dry face. Another important factor which takes away from its efficiency is its lighting power. It gives a dull bluish light, which would be very difficult to find fuses by, and in the endeavour to evolve a substitute for the ordinary gelatine cheesa stick this most important point must be borne in mind.

Mr. J. Chilton (*Member*): We tried these new cheesa sticks at the Village Main Reef last Thursday, and there was only one complaint in the whole mine. Otherwise they were a perfect success.

Mr. T. Donaldson (*Member*): Although these cheesa sticks are put out by water, let them dry a little and they are as good as ever again.

Mr. M. H. Coombe: Referring to Mr. Laschinger's suggestion that a modification of the plumber's blue flame would be an efficient lighting medium, I consider it most impracticable. The miner is burdened with sufficient paraphernalia now, and hedged round with regulations and restrictions which sufficiently hamper him in his work without having to carry a plumber's blue flame lamp around with him, to say nothing of tins of paraffin or petrol. In a dry face no better medium can be used for lighting than a "spit fuse." If "nicked" in a proper manner it is absolutely efficient and quick, and when a 3 ft. "spitting fuse" is burnt out when lighting a round of 5 ft. fuses it is a warning to the miner to "get out" whether his fuses are all lit or not, but in wet faces nothing as efficient as the gelatine cheesa stick has as yet been devised, and the members of this Society will have a hard battle to fight before they chase it out of our mines.

Mr. J. H. Johnson (*Member*): I notice the word cheesa stick is used very continuously. Speaking of metallurgical and mining terms I believe the sub committee decided that the word cheesa stick should not be used. It is a Kafir word, and to spell it as we spell it is wrong also.

The President: I believe, Mr. Johnson, the Zulu dictionaries have special ways of spelling the word, viz., *tshisa* and *tyisa*, but we have apparently bilingualized the word for the *Journal*, and the result is not very pretty.

Mr. E. J. Laschinger: I should like to know the objection to the blow lamp. As for increasing miner's paraphernalia, I would very much rather enjoy an innocuous atmosphere with a blow lamp in a confined space than the noxious gases usually prevailing. What we have to do in the elimination of the cheesa stick is to get rid of the noxious gases arising from that stick, which are the most harmful gases the miner has ever to fear. I do not think we can exaggerate the danger these gases mean to the miner's health and the sooner something is adopted which will do away with the dangerous fumes and give off harmless gases so much the better. I think every miner ought to thank those who try to introduce something of that kind to prolong his life. Therefore I do not see any weighty argument in the suggestion that it would be extra paraphernalia. The cheesa stick takes some time to manipulate. A blow lamp might require a few minutes more to start it going, but in the actual operation of lighting the fuses I think it would take no more time and we have the argument in favour of using such a harmless flame.

Mr. T. Donaldson: This cheesa stick I have described has no harmful gases whatever.

Mr. M. H. Coombe (*Member*): I trust the members of this Society recognize the importance of this matter. One of the most important and hazardous details of the miner's work is the lighting of his fuses. The success of his round of holes depends on the correct lighting sequence of them. In addition to this the element of danger is of the greatest. Despite the advance made in the art of fuse manufacture the best brands will sometimes "run," and the miner knowing this "lights up" as quickly as possible that he may get away from the face in a minimum of time. Herein lies the efficiency of, and consequent liking by the miners for, the gelatine cheesa stick. First, it gives out a fierce heat which cannot fail to ignite the powder in the fuse; second, it burns slowly; third, it gives a strong light which enables him to see clearly and quickly what he has to do, and lights him away after he has lit up; fourth, it is easy of ignition; fifth, it is not extinguished by water falling on it; sixth, it is always at hand and simple of preparation.

I have heard of shift bosses going through the mine with a policeman's bulls-eye lantern hung up to their belts, but I fail to imagine the miner with a plumber's blue flame lamp hung around

his neck, and the boy in the rear with a tin of paraffin or petrol.

Mr. E. J. Laschinger (*Member of Council*): I have had some experience of lighting fuses. I have been underground in a very wet shaft and lit 20 fuses one after the other after everyone else had cleared out. I am not speaking from ignorance of the subject. Although I have not had a chance of doing that sort of work this last few years, I should be quite prepared to do it at any time. I think a plumber's blow lamp would be a good thing to try and see whether it is adapted or could be adapted to the purpose. I do not wish to lay down the law that a blow lamp should be used all along the reef but because of what I know of lighting fuses and of what I know of blow lamps, I think the experiment is worth making If the gentlemen who criticise me would try a blow lamp and let us know their experience we should know something more about it.

Mr. A. Redfern (*Member*): I think that for stopes and drives there can be nothing better than the "spitting fuse" for lighting holes. If a large number of holes has to be fired, then a fuse of equal length to those in the holes charged, should be "nicked" every half inch, the "nick" made by cutting out a small wedge shaped portion of the fuse, always reaching the core. To ensure a good light which will not be extinguished by concussion, a torch should be used, as described by Mr. E. M. Weston, or a similar one made, without using a candle, from the wax coated paper in which some firms supply explosives. When firing charges in a wet sinking shaft, or in case of falling water in stope or drive, the miner carries both knife and "cheesa stick" in one hand, takes up and holds the fuse with the other hand, while he nicks it with the knife and lights it from the cheesa stick. Therefore a blow lamp or other device, to efficiently replace the "cheesa stick," must be capable of being held and used in the same hand with the knife.

Mr. G. Hildick Smith (*Member*): I should like to draw your attention to a lamp, recently invented locally, with which experiments are being conducted at present, I believe by the Mines Trials Committee. The lamp contains a mixture of calcium carbide and calcium phosphide, and a gas is given off, which ignites spontaneously. I have personally used this lamp underground for lighting up fuses and it worked quite all right. It was impossible to blow it out. I believe experiments were also satisfactorily conducted with it on the Village Deep.

The President: Does it not get too hot to hold after using it a few minutes?

Mr. G. Hildick Smith: No, it is protected with asbestos packing.

Prof. J. A. Wilkinson (*Member of Council*): I should like to ask Mr. Saner if electric firing has been used here to any large extent. That undoubtedly is a method which would do away with the cheesa or any other kind of firing stick.

Mr. C. B. Saner (*Vice-President*): I was very keen to try electric blasting. Mr. Weiskopf, from the Dynamite Factory, offered to try it underground. On arrival at the mine I informed him we were going to blast 38 holes, but that before going below I would like to see a demonstration of it on the surface. I sketched out the actual plan of the shaft, full size. The mine captain marked out the position of the 38 holes; the fuses were connected up from outside the area of the shaft; the electric battery was pressed down: final result, eleven misfires!!! If Prof. Wilkinson would come down into a shaft bottom in two or more feet of water in a dimly lighted humid atmosphere standing amongst the fuses he would realize the difficulties of electrical firing in this particular case.

Mr. M. H. Coombe: Electric firing is efficient under certain conditions. The prime condition is that each hole must be free to carry its own burden. In other words, there must be no "easers." In the Kimberley open face workings battery firing was largely used and with success, but only on large benches where each hole had a free "burden." Electric firing can be used in shaft work, and indeed in any class of mine work, but it would be necessary to fire in "rounds," and as this is a loss of time, and a loss of time is a loss of footage, present day practice on the Rand cannot allow of electric firing. Time fuses must be used, as often the results of twelve to fifteen holes depend on the successful breaking of three or four in front of them, and these must be timed to go in front of the back holes.

Mr. W. Waters (*Associate*): With regard to Mr. Saner's remarks about misfires with electric blasting, I should like to mention that I was present with Mr. Weiskopf when the experiments were made. The misfires were not the fault of electric blasting, but must be attributed to long storage of the fuses at the Dynamite Factory. Careful examination of the fuse heads proved that the flashing mixture around the platinum bridges had become affected by dampness. I have tried electric blasting with success on various mines of the Rand, both in stopes and drives, and this method of firing holes can be made effective and efficient if miners will give it the necessary time. On the Rand every miner is in a great hurry to blast his holes and get to the surface as soon as possible. If electric blasting

were insisted upon by the Government and by mine managers, there should be no difficulty in making it successful in every way. We have electric fuses of two kinds: (1) instantaneous (2) with delay action. It is possible to fire a whole face at one time, and to arrange for the shots to go off in the required rotation. I shall be pleased at any time to prove to Mr. Saner that electric blasting can be made a success of, both above and below ground.

Mr. G. Hildick Smith: I was engaged in sinking shafts in a colliery district at home where all the blasting was done by electricity, using high tension detonators fired from the mine power service, a round of sump holes being first fired simultaneously; if it was found that these had fired all right, the remaining holes making up a complete round were finally fired simultaneously also. Electric firing by means of a magneto-exploder was also successfully resorted to in driving the cross measure drifts.

THE SHRINKAGE METHOD OF STOPING AT THE FERREIRA GOLD MINE.

By G. Hildick Smith, B.Sc., F.G.S. (Member).

Generally speaking, overhand methods of stoping on the Rand are the exception rather than the general rule, owing chiefly to the average dip of the reef—which may be taken at about 30°—being too flat to admit of the adoption of these methods, and it is for this reason that the "back filling" and "shrinkage" methods of overhand stoping are perhaps not so clearly understood on this field as they should be.

The author is fortunate enough to be in charge of that section of the Ferreira Gold Mine in which a block of ground, dipping at an angle of about 70°, is being worked out by means of what may perhaps be said to be the first application of the "shrinkage" method of overhand stoping on the Rand, not taking into account, of course, the many examples of the so-called "back" stoping, which are rendered necessary on so many mines due chiefly to dykes and faults. In view of this fact the author has gone to a certain extent into the question of "shrinkage" stoping as practised in various parts of the world, and also into the "back filling" method which is closely allied to the "shrinkage" method and about which a few remarks are included.

The following paper has been written in the hope that it may perhaps be the means of bringing these methods, where practicable, into a more general use in this country, chiefly on account of the increased safety afforded in working as compared with the, at present, too common and dangerous underhand methods employed on the highly inclined areas of the reef.

The subject has been divided into the following heads:—

(1) "Shrinkage" and "back filling" methods of overhand stoping as practised in various parts of the world.
(2) A detailed description of the "shrinkage' method as applied at the Ferreira Gold Mining Co., Ltd.
(3) Advantages of the "shrinkage" method.
(4) Disadvantages of the "shrinkage" method
(5) Conclusions and possible further applicability on the Rand and in South Africa generally.
(6) Bibliography.

(1) *Shrinkage and Back Filling Methods of Overhand Stoping.*—By the "shrinkage" and back filling methods of overhand stoping are understood those methods of breaking ground by means of which ore bodies having a dip of about 50° or over are worked out from the various levels upwards as opposed to the various underhand methods, which necessitate the working out being done from the levels downwards.

(*a*) *The "Back Filling" Method of Overhand Stoping.*—This method differs from the "shrinkage" method inasmuch as the stoped areas as they are formed are filled with waste rock, debris, etc., instead of being filled with part of the ore as it is broken. This system, as already mentioned, is chiefly applicable to highly inclined ore bodies, and hence is the system of stoping *par excellence* for sand filling; as by the adoption of the method of timbering the chutes described in detail under head 2, either by filling directly off the drive pillars, or where no drive pillars are left, off stulls, practically all the reef can be taken out. When the mine is worked out, the result is that the whole of the worked out portions can be closely filled with sand or waste, eliminating all danger to the surface, etc., due to caving ground or what not. This in theory, therefore, is the ideal method of working out a highly inclined ore body and is employed, *e.g.*, in the Butte District, Montana, and in Western Australia, etc.

(*b*) In the "Shrinkage" method of overhand stoping the ore broken is used temporarily for the filling, and therefore during what may be termed the first working, only about 33% of the actual ground broken can be directly run off for further treatment. The remaining 67% has to be left in the stopes as filling and remains there until the block being stoped is worked out, when the whole can be run off through the boxes and trammed to the shaft. This method is employed at the Alaska Treadwell Mine, Alaska; Colorado

etc., and it is this method that has been employed at the Ferreira Gold Mine, described in greater detail under head 2.

To summarise therefore we have the "back filling" method, in which all the broken ore is directly available for treatment as required, suitable for moderately wide deposits requiring a minimum of timber and a minimum amount of ore to be left as safety pillars, etc., and also sufficient suitable waste material to fill up all stoped out areas.

The "shrinkage" method is employed where insufficient waste for filling is available, and it requires only a minimum of timber and a minimum of ore to be left as safety pillars, etc.; it is suitable for any width of ore body, but it necessitates a greater expense for actual mining than the "back filling" method, other things being equal, on account of the loss of interest on the money expended to break the ground until it is finally run off.

(2) *A Detailed Description of the Shrinkage Method as Applied at the Ferreira Gold Mine.*—A block of ground A, Fig. I., having been left standing between the No. 1 and No. 2 shafts, and also west of No. 1 shaft, extending from the 100 ft. to 300 ft. level, giving on the dip of the reef about a 200 ft. back and 240 ft. along the strike had to be worked out. Owing to the length of the back and the trouble experienced in the old underhand stopes working on other portions of the reef having the same dip at this mine, and also chiefly to the increased safety obtainable, it was decided to work out this block on the "shrinkage" system. By referring to Fig. I. the general condition of affairs before work was commenced on the block can be seen. The No. 1 shaft being here above the reef, it was necessary to leave a shaft pillar *b*, 40 ft. on each side of the shaft. The two remaining portions of the block A, could therefore be worked when required.

Work was commenced on the eastern portion *a*, as follows :—

The distance along the 300 ft. drive between the shaft pillar line and the fault *c* was roughly bisected at *d* and a rise commenced to connect to a winze *e* sunk down from the 100 ft. level for about 50 ft. The rise was put up by means of Waugh drills, and at the same time the box holes about 30 ft. apart along the drive were also put up to a height of 15 ft. As will be seen from Fig. I. the ground below the block A had previously been stoped away as far as possible by the underhand method causing the 300 ft. level itself to be carried on stulls (as shown in Fig. VI.). The hanging being good, this old stope was still standing open as far as the 400 ft. level. On this account it was thought advisable to leave drive pillars above the 300 ft. level instead of, as might have been possible, taking all ground out from immediately above the level and replacing it by a row of stulls with box holes at convenient intervals. A stope drive was carried east and west from a point 15 ft. up the raise to hole the box holes and form the drive pillars. Starting with the idea that we are going to stope and fill with broken ore, it will be seen that it is necessary to have some definite and safe way of getting to the face as it proceeds upwards, and also to have an easy method of handling any broken ore in excess of that required for filling. It was therefore decided to carry up three pass ways in all, these to be put in and reared up as stoping proceeded, from the box holes Nos. 2, 4, and 7, Fig. I., and in the remaining box holes, Nos. 1, 3, 5, 6, 8, to put in ordinary boxes of the type shown in Fig. II.

For this kind of box all that is required is that the box holes should be squared up, by "pop

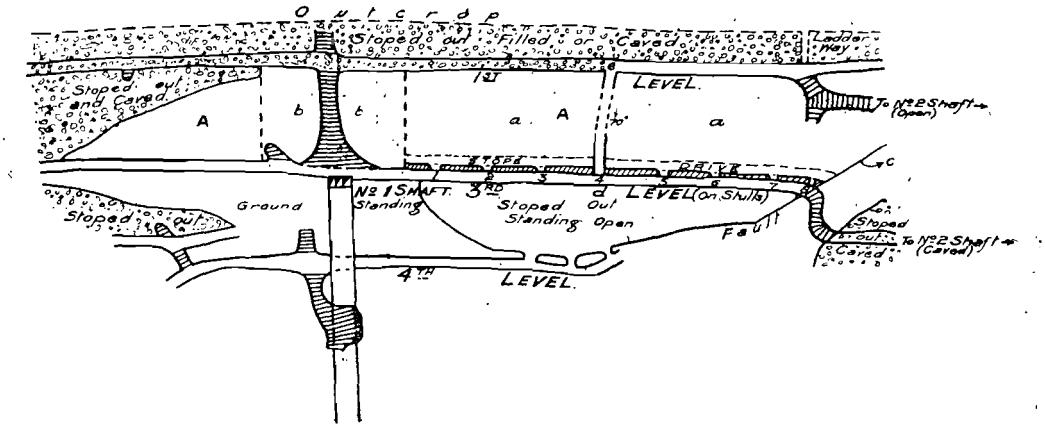


FIG. I.—(Plan.)

FIG. II.

holes" if necessary, to allow for a good head and hitch for each of the two back legs *bb*, Fig. II. These legs should be about 12 in. diameter, and well put in. To these are bolted two strong lagging pieces *aa* to form the front legs of the the box, across which is bolted another lagging piece *c* as a bearer for the bottom planks *d*. The box is closed in at the sides and across the back legs for a sufficient distance down from the hanging with $2\frac{1}{2}$ in. planks, and $\frac{1}{2}$ in. × $2\frac{1}{2}$ in. iron straps *ee* are bolted on the front legs to hold the front boards, which can be taken out when required to run ore from the box. A box of this description can be put in in one shift by a white man and five boys at a cost of 35s. for labour and 37s. for timber, bolts, etc., or a total cost of 72s. Bearers for the setts used to form the ladderways and ore passes in the box holes Nos. 2, 4 and 7, Fig. I. were then put in. The bearers were of 9 in. × 6 in. timber and were three in number, the outside two to carry the end pieces, and the centre one to carry the dividing piece of the first sett. The setts were

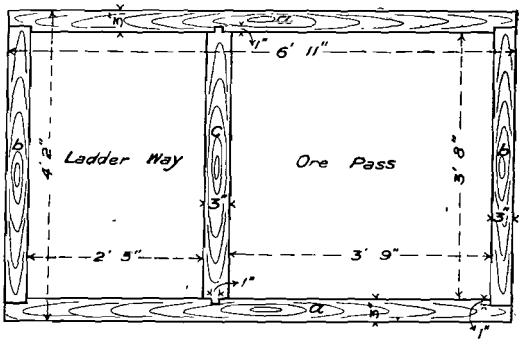


FIG. III.

cut from 3 in. × 9 in. gum planks, and were put together as shown in Fig. III., *aa* being the wall plates, *bb* end pieces, and *c* the divider, between the ladderway and ore-pass. It is important that the setts should be made of gum planks or some equally hard wood, in preference to deals, which are too soft and would be broken up before stoping was finished. Each complete sett costs 17s.

for timber, and one carpenter could cut sufficient setts to keep the work going underground by cutting out 10 setts per day, giving a cost for labour on the surface of 2s. per sett. One white man and 5 boys, on single shift, put in on an average 12 setts per shift underground, which were sufficient to keep pace with the stope, (when stoping was being done on double shift) giving a cost for labour underground of 3s. per sett or a total cost per sett, when in position, of 22s. making a cost per foot for each chute of 29s. 4d. This gives a cost for timbering of 1s. per ton broken for the particular block of ground under consideration. Another important step in the building up of the setts is the proper placing of the three bearers already mentioned. These should be placed as nearly as possible at right angles to the dip of the reef—and hitched in the foot and hanging—so that wh n the setts are built up from them, little or no curve is required at the start to bring the setts into the line necessary to carry them straight up through the stope. This is shown in *a*, Fig. IV., not as at *b*, Fig. IV., which necessitates a curve, an easy thing to make if only a slight curve is required, but if

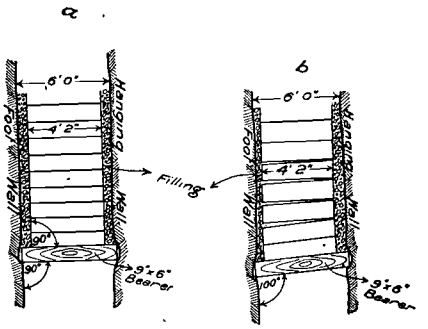


Fig. IV.

too sharp a curve is necessary with only an inch joggle, the dividing pieces will be out of their joggles at either their foot or hanging wall ends, depending on which way the curve is necessary and a wide gap will consequently be left between the setts.

For the sizes of the setts used the bearers were placed 3 ft. 7 in. apart for the ore-pass and 2 ft. 3 in. apart for the ladderway. The bearers having been carefully fixed into position, the first sett can be laid directly on them and secured in

Fig. IV*a*.

any position required by filling in and wedging up around the wall plates and end pieces; no bolts, nails, etc., being used throughout, each sett simply lying directly on the one below, the whole being packed in and wedged tight, as may be seen from Fig. IV*a*., p. 196. The setts should be carried up to within about 6 ft. of the stope face. Due to the fact that as the stope face proceeds upwards any broken ore shovelled into the ore-pass would have some distance to fall and would therefore smash up the type of box shown in Fig. II., it was necessary to put in a different kind of box at the bottom of each ore pass. The ore-pass boxes were made as shown in Fig. V.

The front legs *aa* were of 9 in. × 6 in. timber morticed and bolted on to the bearers *bb* at the top, and at their footwall ends placed on a 9 in. × 6 in. baulk packed into the stull filling. A cross bearer *c* was placed between the front legs and held in position by morticing and the bolt *d*; this bearer carries the front ends of the bottom planks of the box, the back ends of which are supported by a cross lagging piece *e* carried on holding-down bolts. The planks forming the bottom of the box were finally covered by ½ in. iron plate *f* cut to fit behind the front legs to prevent it slipping forward. The sides were closed in with gum planks and the front fitted with a ½ in. iron door *g*

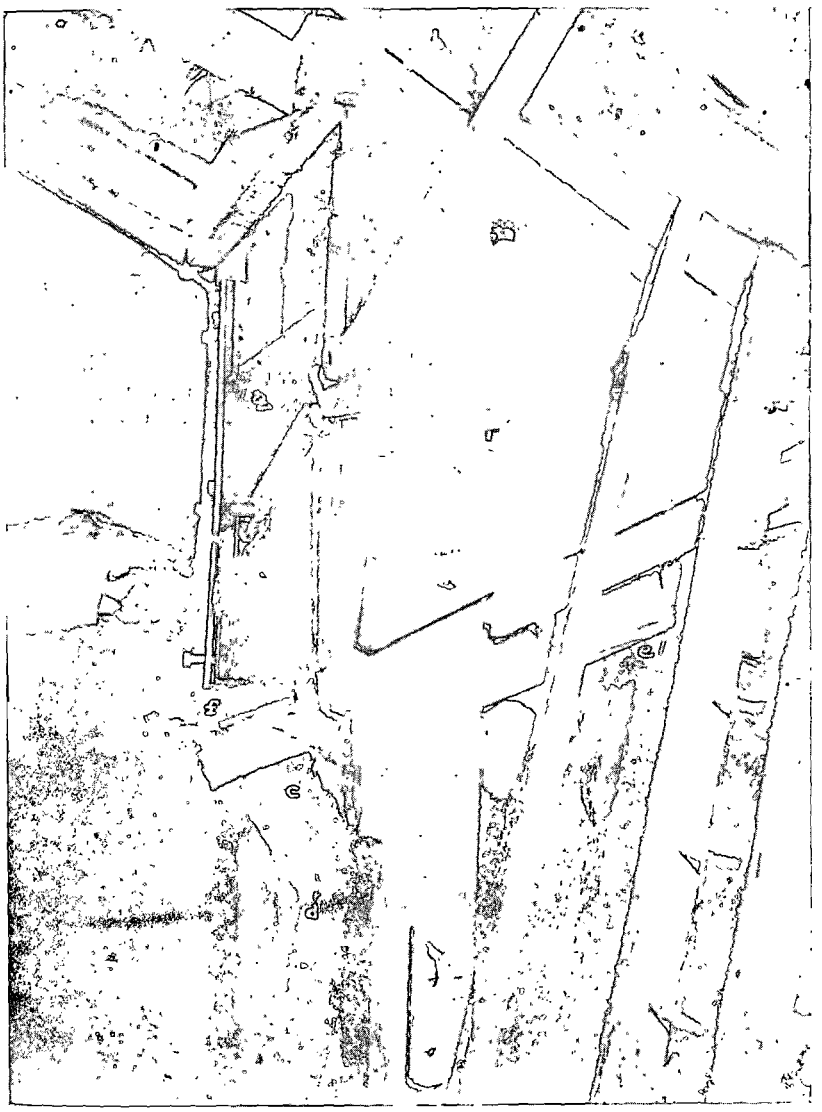


FIG. V.

moving in suitable slides and operated by means of the lever *h*.

A box of this kind can be put in in $1\frac{1}{2}$ shifts by a white man and five boys at a cost of 53s. for labour and 85s. for timber, plates, etc., or a total cost of £6 18s. All air pipes were at first carried up the ladder-ways (as shown in Fig. V.), and as stoping proceeded fresh lengths of piping were added, all taps being kept below the level of the top sett.

The ladders were made of $2\frac{1}{2}$ in. × $4\frac{1}{2}$ in. planks, placed 10 in. apart, with $\frac{3}{4}$ in. diameter rungs, having a 10 in. tread. Ladders of this description (as seen in Fig. V.) would cost, when in position, about 1s. per ft.

A general idea of the condition of affairs after the stope had been got into good order, may be gathered by reference to Fig. VI. It was found best to have a box at the extreme east and west boundaries of the stope. On the west, therefore, the pillar, as originally cut, was stoped away by hand labour and replaced by stulls and the box put in (as shown in Fig. VI.) This allowed the broken ore to be run off more easily, and hence the cutting of the shaft pillar was facilitated, the last bench being always carried out dead in line with the preceding benches. This gave some trouble at first as the face was carried too steeply and the broken ore became blocked up at the points of the stope, the portions of the face east and west of the chutes were eventually carried at a flatter angle than the rest of the face —the best angle for which was found to be about 35° (as shown in Fig. VI.)—and no further difficulty in this way was experienced.

The main reef and leader run together giving a total width of reef about 12 ft. The main reef portion being of poor value, the leader bands only were stoped on to the so-called interbedded dyke, which is here slightly decomposed giving a narrow soapy band and making an excellent footwall to break to. The hanging also is very regular and forms a good breaking plane, the average stope width being about 6 ft. All stoping was done on contract, double shift, at a price of 40s. per fathom, and the average results obtained were as follows :—

Fathoms per machine per shift = 0·582.
Tons per machine per shift = 10·48.
Cost of explosives per fathom = 12s. 1d.
Cost of explosives per measured ton = 8·06d.
Pounds of explosives used per ton = 0·57.

Giving a total cost per measured ton broken of 3s. 1d.: this includes contractor's earnings, cost of upkeep of machines, drill sharpening, etc., but exclusive of timbering.

Usually four holes were drilled to a bench, in which case the two top holes were drilled so as to just carry water, and the front holes were drilled as nearly parallel to the face of the bench as possible. All holes were drilled nearest together at the collar and leading outwards in order to reach the foot and hanging at the back ends as nearly as possible. Sometimes a bench could be carried

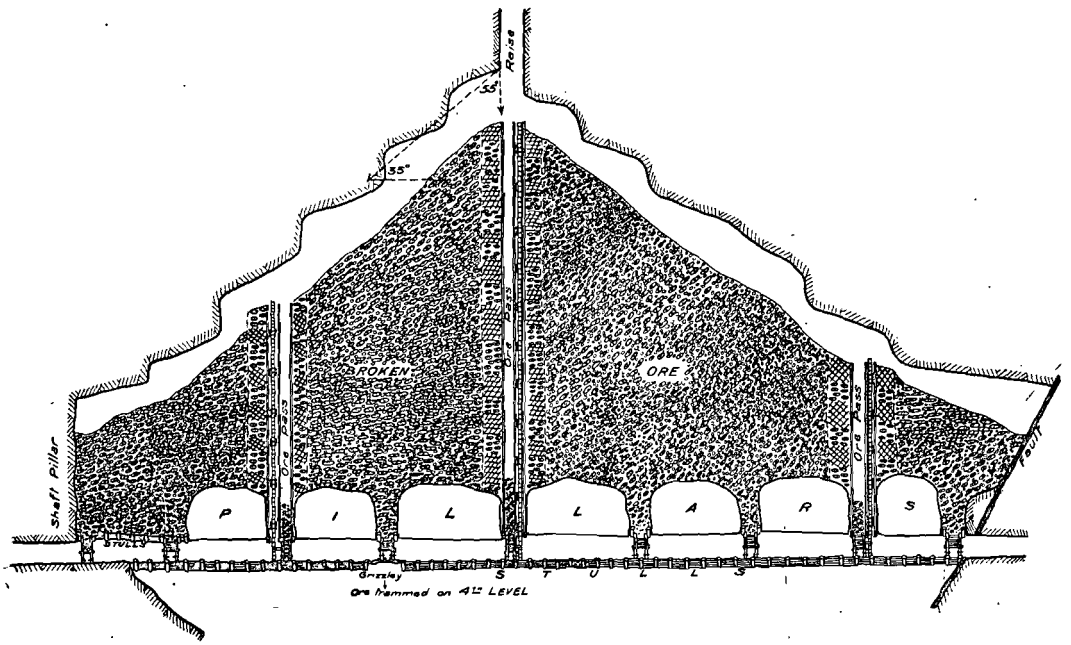


FIG. VI.—(Section.)

with three holes, in which case the two back holes were drilled as usual and the third in front in the centre of the bench. There are one or two points in the handling of the machines which may be of interest. It often happens that the filling is not sufficiently near the face to allow of the bar being rigged vertically from it, in which case either (1) woodblocks were piled up from the filling a sufficient height to allow the bar to be rigged vertically; or (2) the bar is rigged across the stope from the hanging to the footwall, and in order to do this a temporary staging if necessary, may be fixed by either :—

(*a*) Placing two lagging pieces hitched in the hanging and driven down on the footwall, level with each other and a few feet apart, across which planks are placed, forming a level platform. As most of the weight on these is on the footwall side the tendency is for the lagging to be tightened in place ; or (*b*) the staging is formed by placing planks *aa*, Fig. VII., across two lagging pieces *bb*, or across one lagging and back on to the filling. These in turn are supported by means of carrying bolts *cc* placed as shown in Fig. VII. in

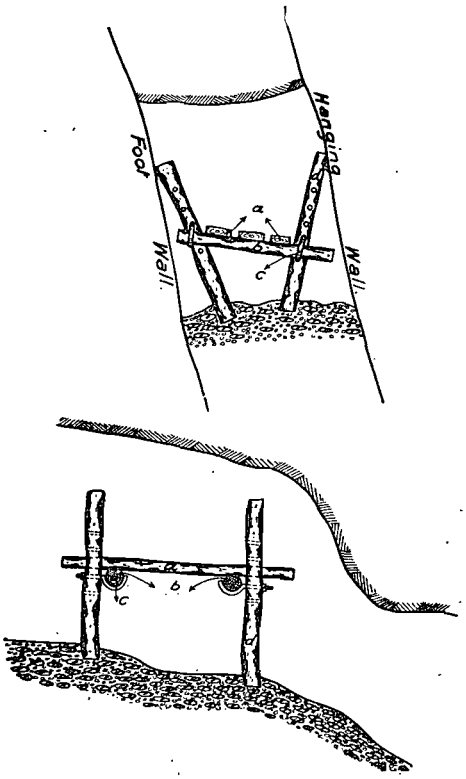


FIG. VII.

holes drilled at convenient heights in the lagging pieces *dd* which are placed vertically on the filling.

Advantages of the Shrinkage Method.—The chief advantage of this method of stoping is its simplicity and safety. No trouble is occasioned by bad hanging, as the only portion of the hanging which can possibly fall away is the small area between the filling and the face, which can always be easily examined. The face itself may be liable to fall away when it would have to be temporarily supported by props off the filling. Air pipes give no trouble as they are carried directly up the ladderway and are easily accessible for repairs, etc. Any width of stope can be carried as required by the reef widening or narrowing down, and no trouble is directly experienced if holes are drilled into the hanging thereby breaking it in places. A good current of air is kept along the face, the chutes acting as upcasts and owing to their small area and the small area between the filling and the face, excellent ventilation is obtained. Comparatively little shovelling is required and the broken ore can be run off where necessary from any of the boxes. Prospecting crosscuts or holes can be put into the hanging or footwall at any point required with great ease without causing future trouble.

(4) *Disadvantages of the Shrinkage Method.*—Only about 33% of the ore broken is immediately available, the remaining 67% having to be left until the whole block of ground is stoped out. This is not of so much consequence in cases where the whole mine is worked by this system, when the necessary tonnage to keep the mill going would be obtained from previously stoped out blocks, but in cases where, as in this instance, the stopes are chiefly worked underhand, it practically means that so far as available ore for crushing is concerned, the ore broken in this block cannot be taken into account as being of any help in keeping up the required tonnage for the mill. A loss of interest on the money expended in breaking ground by this method is thereby occasioned. If due to carelessness more rock than can really be spared is run off from any one box, the filling sinks too far away from the face, and extra time and trouble is required in rigging up the machines ; in fact it may be impossible to rig them up on the benches necessary to keep the stope in proper shape, and good efficiency in breaking the ground is lost. Efficiency is also affected adversely owing to the fact that a number of dry holes are often necessary either off the side of the rise or to square up the face to make a good bench. The dry holes cause the making of a certain amount of dust and therefore the use of a spray is necessary.

(5) *Conclusions and Possible Further Applicability on the Rand and in South Africa Gener-*

ally.—It is, after all, in a proper regard to detail that most care is needed to ensure success in any operation, and it is due to a proper regard to detail that success or otherwise is generally obtained. This is especially the case with regard to this overhand method of stoping as compared with the usual underhand method in vogue on this field. To obtain good efficiency underground it is first necessary, other things being equal, that the actual breaking of the ground should be done in as efficient a manner as possible. In order that this may be accomplished good benches must be carried and the holes given a full burden. Acting against this in the overhand method of stoping, we have, where the stope width allows of the bar being rigged across the stope, the difficulty of drilling a four hole bench in order to give each hole a proper share of the burden. This will perhaps be better understood from Fig. VIII.

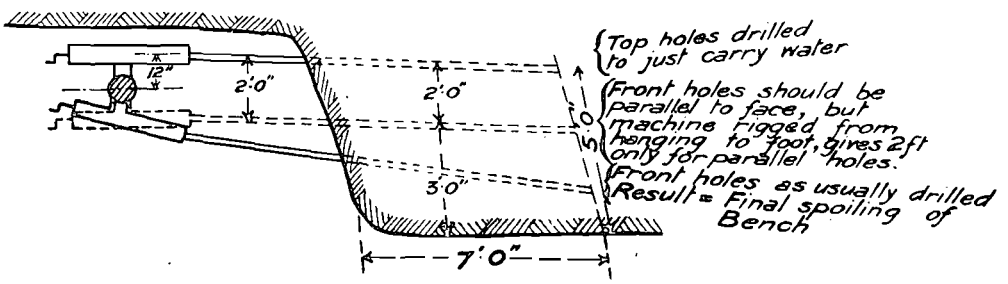


Fig. VIII.

The tendency is, as a rule, to run the bottom holes in such a way as to break off the triangular bottom corner of the bench rather than to run them parallel to the face of the bench. Thus the bench as started on may be a good one, yet after a few rounds the bench is spoilt, and in order to make a good bench again, dry top holes are necessitated and a short and bad round is the result. The chief point in which care should be exercised, is in the rigging of the cross bar in such a position to allow of the top holes being as high as possible and at the same time to just carry water.

As already mentioned, the best angle of dip for the stope face is about 35° (see Fig. VI.). If a steeper angle than this is carried, the broken ground from the bench higher up is liable, on blasting, to run down the stope, and the large rocks will become jammed between the next lower bench and the filling. It is an important point in this connection that a bench should not be carried forward far enough to run in above the next bench lower down the face and form one large bench, which cannot be carried in one lift. When this happens the broken ore will always get jammed against the point of the bench and thus cause delay and danger in shovelling and drilling.

With regard to the building up of the chutes when each has a ladderway, they should be so built up that the ore as it "rills" down will run directly into the ore-pass side and not into the ladderway side (see Fig. VI). This does not apply to the leading or highest chute—in this case the centre chute. The centre chute was carried at first directly up the line of the rise, but owing to the rise having gone off the original line from about the middle of the stope, the east side of the chute eventually came in line with the west side of the rise. This was found to be an advantage, as at first when benches were taken off each side of the rise at the same time, the chute became covered with a large pile of ore which took some time to remove, and eventually only the broken ore from one bench could cover the chute, which was thus more easily cleared in order that the chute might be built on to or the air pipes reached.

With regard to the air pipes, $1\frac{1}{2}$ in. pipes were at first taken up each ladderway and kept close up to the top of the pass, but as already mentioned owing to the difficulty and waste of time in clearing debris from off the centre chute, 2 in. pipes were carried down the rise from the 100 ft. level, these being in 10 ft. lengths in the rise itself, to allow of a short length being taken off as stoping proceeded upwards.

Jumpers at first were carried up the ladderways from the 300 ft. level, but were finally let down by means of a hand winch and kibble from the 100 ft. level, and thrown down the passes to the 300 ft. level when finished with. This gave some trouble, as the jumpers falling down the ladderways and ore-passes smashed up the ladders and ore-pass doors, and it would be preferable to either have part of one of the ladderways bratticed off for jumpers only, or to carry old 4 in. pipes up a ladderway down which the jumpers might be dropped.

Some ore should always be left in the ore-pass sides of the chutes, or otherwise, as the stope

proceeds upwards and the fall increases, the boxes will be continually broken.

It is also important that a kafir should be kept continually "pop holing" and breaking up large rocks in the stope, or otherwise these rocks will eventually become jammed in the box holes as the ore is run off.

As may perhaps be noticed from an inspection of Fig. 6, the centre chute may seem unnecessary, and in the ordinary course it would be so, as the block of ground shown in Fig. 1 might with ease and less expense be stoped without it, but the idea of the management was not only to eventually fill up this stope itself with rock from the waste dump, which lies directly above it on the surface, but also all other stoped areas below the 300 ft. level, which had to be done as soon as possible, and for this reason it was necessary to have a pass way through for the waste from the 100 ft. level as soon as possible, and the centre chute answers this purpose.

As to the further applicability of the shrinkage or back filling systems on the Rand and in South Africa generally, it is possible that the back filling system would be especially suitable to many of the mines in Rhodesia and on the Rand where the dip is steep, and where sand, waste rock, etc., is at hand for filling. The shrinkage method might also be applied where the reef is narrow and highly inclined, in which case a modification of both systems might be employed, the waste being first blasted down for filling and the reef resued and shovelled into the ore-passes.

This might be suitable for mines in the Randfontein district. Either of the above methods would give far greater safety than the underhand method for the highly inclined portion of the reef.

(6) *Bibliography.*—For those who require to go more fully into this subject the following short list of papers on the same may be of interest :—

"Ore Mining Methods," by W. R. Crane (Wiley).

"Shrinkage Stoping in Western Australia." by F. R. Rolfe. *Transactions Institution Mining and Metallurgy*, Volume xviii., p. 29.

"Unsolved Problems in Metal Mining," by Professor H. Louis. *Transactions Institute of Civil Engineers.* Volume clxxiv., p. 240.

"The Treadwell Group of Mines, Alaska," by R. A. Kinzie. *Transactions Institute of American Mining Engineers,* Volume xxxiv., p. 334.

The author hopes that the many modifications of the shrinkage or back filling methods of overhand stoping, which are bound to suggest themselves to his hearers, will be fully discussed at future meetings, and his thanks are due to Mr. Palmer Carter, the manager of the Ferreira G. M. Co., for permission to publish this article.

The President: We are very glad to have a paper which will give us scope for discussion

Mr. C. B. Saner (*Vice-President*): In the early days when we were working from the outcrop downwards, the reef dipped at an angle of 70° or upwards. At the Rietfontein Estate the dip of the reef was 80° in places, at the Henry Nourse and at the New Heriot it was also very steep, and the same shrinkage system was adopted. Then there was no question of big batteries, shortage of labour, or "keep the mill going at whatever cost." So that I think Mr. Smith is rather severe on the poor unfortunate old workers on the Rand when he says in this third paragraph he is going to teach them what ought to be done. Of course the Central Rand, on the whole, has got away from the outcrop. We may have these steep dips at Randfontein, but we have not got them on the East or Central Rand now. This shrinkage method means that three quarters of the rock broken has to lie in the stope, an ideal position that few mines can enjoy. I have great pleasure in proposing a hearty vote of thanks to Mr. Smith for bringing this paper before us.

Mr. M. H. Coombe (*Member*): I take exception to a sentence in the second paragraph of the paper just read, wherein the writer describes the system of "shrinkage stoping" as being new in Rand practice. Shrinkage stoping is the oldest known form of extraction of ores from lodes. It is practised all over the world, is largely used in Rhodesia, and was in general practice on these fields in the early days. I was engaged on this same kind of stoping on the old Van Ryn, the Aurora and Spes Bona, and the majority of the old outcrop mines, if not all, had their ore extracted on this principle to supply the first mills erected on the Rand. So by no stretch of the imagination can shrinkage stoping on the Rand be described as *new*. The author deserves credit for his well written paper and his wealth of detail, but I cannot see that the square setts are necessary. Poles "hitched" in "foot" and "hanging" with pole lagging to form the "passes" are just as efficient and much stronger for the rough usage required. The author did not say how many of these setts were destroyed by blasting, and broken by rocks, necessitating replacing. I also seriously question his costs of box installation.

SAND-FILLING ON THE WITWATERSRAND.

(Read at June Meeting, 1910.)

By Edgar Pam, A.R.S.M. (Member).

REPLY TO DISCUSSION.

Mr. Edgar Pam (*Member*): In addition to replying to the criticism on my paper on sand filling, I will, with your permission, describe the various alterations which have proved valuable since June.

With regard to the arrangements above ground, the method of pumping and dewatering, as described by Mr. Powell, is most interesting, and has undoubtedly some advantages over the Silesian bin method. In my opinion, however, the advantages over the more recent plants built for the Rand Mines, Ltd., will not be such as to justify the considerable extra outlay in capital and working costs.

The bins recently built differ from that which I described in the following details: The bottom of the bin is inclined at 30% instead of 25%. The launders running to the shaft are inclined at 20% instead of 12½%. The apertures are 30 in. × 18 in. or larger, instead of 10 in. × 6 in. The nozzles are 2 in. × ⅛ in., instead of ½ in. diameter. With a bin of this description, it is possible to send down sand containing not more than 35% to 40% of water, and if the pipe line would allow it, I am sure that the proportion of water could be diminished still further. At present, however, the considerable horizontal distances along which the sand has to flow, prevents the use of a thicker pulp.

So far as the efficiency and economy of the cyanicide are concerned, the advantages claimed by Mr. Powell do not appear to be of great moment, as our chemists have demonstrated the possibility of using the cyanicide in the tanks, and I am certain that when the sand is being treated regularly, the proportion of chemicals wasted will be very small indeed.

On the question of soluble gold, I regret that I can express no opinion, but should there be any soluble gold in the current sands, it will surely be the duty of the metallurgists to have it extracted before sending the sand to the mine.

Whether the total elimination of slime will prove beneficial is, to my mind, problematical, as the admixture of slime will probably have a binding effect on the sand. I would also mention that on the mines with which I am connected the percentage of slime in the sand is greater than on the Simmer and Jack, and should the dewatering process be used, a plant for depositing the slime would be needed.

On mines where sand filling on a large scale is contemplated, I feel sure that a tunnel driven from the cyanide works to the shaft at about 20% grade will afford the most economical method of transport. The capital outlay on the tunnel might be very great in some cases, but the working and maintenance cost will be practically a negligible figure.

Pipes.—Several members have advocated the use of launders in place of pipes, and there is no doubt that they are preferable where the sand is not to be carried any considerable distance on the horizontal. It must be remembered, however, that in the deep levels the sand will be lowered through one shaft only and from the bottom of this it will be necessary to carry the sand in all directions with very little fall, and in these cases it will be found, I think, that piping the whole way will afford the most elastic and therefore the most economical service. Ordinary unlined black piping has proved very useful where the pulp is not acid, and on the Village Deep and Village Main Reef 75,000 tons were lowered through the same column before any considerable trouble due to wear and breakage was encountered. Californian red-wood lining was found to wear very fast and Jarrah wood, although better, is far from satisfactory. We have ordered some porcelain pipes from Germany and in addition are trying white iron, earthenware, flint, belt and wood pulp lined pipes.

Barricading.—The stope described by Mr. Hughes is, as he says, an ideal one for sand filling, and whenever possible similar dykes or solid faces are used in place of timber. Mr. Johnson has described the method in vogue on the Ferreira Deep of filling against the south side of the drive, and so minimising the amount of barricading required; this method is proving quite successful.

For ordinary stoping widths up to about 10 ft., I think that where barricading is necessary the method of timbering, as already described, will be found to be be the most satisfactory.

While discussing barricades, I should like to point out that although I questioned the advisability of eliminating all slime from the point of view of the final consistency of the sand pack, yet with regard to the barricading and to the drawing off of the water the cleaner the sand the less the trouble that may be expected. Cocoanut matting which, on the Simmer and Jack, allows water to percolate while preventing any sand coming through it, does not act so well on the other mines. The slime appears to choke the pores in the matting almost instantaneously, and neither water nor sand drain through. For this reason therefore the admirable contrivance for drawing off water described by Mr. Hughes has not been

successful on other mines, and ordinary launders closed up with wooden slats, as the sand rises, are still in use.

On the Rose Deep Mr. Marquard is building a pack, similar to the one he described, and I think it will be successful in holding back the sand. His idea, which Mr. Laschinger also mentioned, of making the barrier porous will, I am afraid, never be realised with the slimy sand which is being handled. The use of a waste pack inside the timber would be undoubtedly valuable, but I do not think the extra expenditure will be necessary. Mr. Marquard and Mr. Johnson propose the use of ferruginous soil or coarse rock to give cohesion to the sand. Mr. Laschinger favours Portland cement for the same purpose. Any of these three might be good, but the cost of adding them would prohibit the adoption of sand filling on a large scale.

Any quite rigid barricade such as concrete would crack as soon as any weight came on to it from above.

I may mention that last week a portion of the side barricade of one of the stopes of the Village Main Reef was removed in order that the condition of the sand might be noted. The face of the sand remained quite vertical, and a considerable effort was necessary to scrape off any sand with the fingers. The stope in question had been filled three months ago. Both Mr. Raine and myself were quite satisfied that the pack would hold any weight which might come on it. On the Robinson mine the slimes dam method of building side barricades, mentioned by Mr. Marquard, is being used with success, but only a small amount of sand can be added each day and the pulp used must be very thick. In the same mine the sand is being sluiced off the dump down the shaft without any handling.

It will interest you to know that since June 300,000 tons of sand have been lowered into three mines, the Village Deep, Village Main Reef, and Ferreira Deep. Nearly all the mines of the Rand Mines-Eckstein Group are installing sand filling plants, and the daily tonnage lowered will steadily increase from about 4,000 tons per day at present to at least double this quantity.

I do not care to encroach on the chemical section of this subject, but would like to say that we are grateful to Mr. White for suggesting the the use of potassium permanganate, a suggestion which is being successfully acted on.

Finally, I tender my thanks to the members of the Society for the kindness of their criticism on a paper which was read perhaps at too early a stage of the process. There is, I am aware, still much room for improvement in our methods, and I trust that should any member have any further ideas on the subject he will give me an opportunity of testing them.

Mr. E. M. Weston (*Member*): I would like to draw attention to the *Sydney Bulletin* mining notes of October 20th. Its Kalgoorlie correspondent writing regarding the collapse of the upper levels of the Horseshoe Mine states this to be due to the stopes having been filled with coarse sand which will not pack properly, but runs under pressure. He states that with the finer grinding policy employed in later years the lower stopes are filled with what is practically slime which packs satisfactorily.

The President: I am sure we all thank Mr. Pam for his interesting and valuable reply. From our point of view, it is very reassuring.

THE EFFICIENCY OF LABOUR UNDERGROUND.

(*Read at August Meeting, 1910.*)

By Tom Johnson (Member of Council).

DISCUSSION.

Mr. E. M. Weston (*Member*): I must congratulate the author on the consistent manner at which he pegs away trying to hammer a few of the elementary principles of efficiency, by which our efforts should be guided, into our heads which appear to have their reputedly enormous brain development covered with a remarkably hard, tough, and impenetrable cranium. The author is apparently never discouraged, and returns year after year to his labours. When I was younger than I am to-day I had the idea that it was only necessary, in order to get a principle adopted and acted upon, to prove its theoretical truth and to show that in practice it had been borne out in actual results obtained. I fondly cherished the idea that every month the heads of the mining industry spent several nights with wet towels round their heads studying the pages of the *Journal*, and particularly Dr. Moir's chemical equations and nomenclatures. I imagined them taking copious notes and sending instructions out on every side to try this or that new idea. It was, I am afraid, in regard to mining at any rate, a vague delusion.

The author has never been deluded nor discouraged. Many years ago he tried to instil into our minds some elementary notions as to the value and need of artificial ventilation in our mines. By his efforts, and those of others, the facts began at last to be appreciated. In this paper he returns to the charge with regard to mine labour. The running of a Rand mine in the past always

reminded me of the efforts put forth to keep some out of date and worn out locomotive going on the tracks. No matter if the tubes were leaky, the feed pump worn out, the bushings required taking up and the wheels were worn irregularly, no matter if she knocked and groaned and hammered and wasted half her power in friction, yet as long as she moved and hauled a truck or two she was kept in service. So with a mine—down below air might be leaking everywhere or wasted in worn out and inefficient machines working with bent and blunted steel. Shift boss might be working against boss, and mine captain at odds with surveyor and sampler. One man might really be undoing the work of another on the opposite shift, and the men working without energy, *esprit de corps*, or security of tenure. Stopes might be 6 ft. on a 2 ft. reef, yet as long as the machine went, though disabled and hampered by friction and lack of co-ordination of its parts, as long as so many trucks of something came every shift from so-called stopes, and as long as so many skips of so-called ore came to the surface to keep so many iron headed rods hammering at a pre-arranged and fixed speed seven days a week year in and year out, everything must be well. I wonder if anyone really believes that this policy of running the mill seven days a week has always paid in the past. I am no strict Sabbatarian of the old school; but I was very much struck by a remark made by a mine captain, who died recently, having given his life to his work. "I wish there was no Sunday." What he meant was this,—the thought of and struggle all the week regarding the necessity of keeping those remorseless stamps pounding on something for an extra day at the end of each week poisoned all the work of the week, filled it with an over-worry, over-care, and tended to rush and inefficiency.

Regarding the prevention of dust, there is no doubt that for drives the water jet is at present the most useful appliance. The Leyner drill has been very greatly improved and simplified since it was last tried on the Rand, and now holds at least three speed records in driving adits in hard ground (granite) in America. I admit that the upkeep of steel is expensive and troublesome, yet I am persuaded that with a little care it could be used with economy for rising (especially on the flatter dips of the deep levels.) It and other hollow steel drills will, however, never be a success until pure neutral water is sent underground for their use, and trained labour available for running them. The Leyner air and water system down hollow steel is the only rational method of introducing water into holes while they are being drilled. A practical miner, of course, knows very well that in working with a piston drill the trouble in a flat upper hole is to get the broken rock out of the hole. In America the miner keeps a thin wire scraper always working alongside the drill in the hole, or uses steel with lugs between the cruciform sections in order to hasten the discharge of the drillings.

I am afraid that any scheme for forcing bagging around the mouth of the hole will prevent, and retard the emission of broken chips and will be neither economical nor effective. With hammer drills working on steep upper holes the question is different, and Dr. Aymard has been long ago anticipated by devices such as sponges held against the mouth of the hole by springs.

Regarding Petersen's respirator, one can only remark that as there is always a danger of an unforeseen generation of large amounts of poisonous gas, for example, carbon monoxide in compressed air It seems, to say the least of it, just a little unwise to provide for the certain death of every machine operator in the mine in the event of a compressor explosion. Besides, are we going to allow the unfortunate native to continue to swallow dust as in the past? If we are not, I see a pretty picture of a machine at work in a rise with four or five masks with their attached pigtails getting hopelessly tangled up, and the supply hoses cut or broken by jumpers, hammers, etc., being thrown on them or by being trodden on.

Efficiency in mining can be only increased by educating the workers. Five years ago I urged the necessity of educating the rock drill workers, and pointed out the benefit that would occur from a practical training and a sound knowledge of some of the simpler laws regarding mechanics, explosives, and blasting. My recent experience goes to show that the average adult miner on the Rand has not the slightest belief that he could increase his earning power by any such means. Education of this sort he neither appreciates nor desires, though there are shining exceptions. The problem of the future is to make this education compulsory to the beginner and desirable and attractive to the older miner. Ignorance of important laws governing efficiency in the use of explosives is almost universal. Take the question of the relative efficiencies of machine and hand stoping. I have several times dealt with this question before the Society, and my excuse for wearying some of you is that I have come to believe, with the author, in the policy of hammering it in.

Mr. E. J. Wiseman in the September *Journal* seems worried about the question. He is quite right in all he says, but owing to ignorance of that law of blasting which states that the quantity of explosives necessary to break a hole varies as the square of the burden on it, his explanation is incomplete. A practical miner

writing to the *South African Mining Journal* also appears to have somewhat confused ideas on the subject. He suggests that the conical shape of the machine hole tends to reduce the efficiency of the explosive causing the tamping to be blown out as one reason, and the fact that misfires or badly-placed holes in a bench must ruin holes behind them, as another. With regard to the first cause suggested it need only be said that provided the confinement of the charge is good, *i.e.*, the cartridges have been properly placed in the hole, and that a moderate amount of tamping has been employed, the high rate of detonation of blasting gelatine enables it to give quite as good results in a machine hole as in the more cylindrical hand hole.

The loss in waste of explosives due to badly placed holes in benches is merely the penalty of inefficiency. There should be no badly placed holes, and with a careful miner who understands what he is doing and who starts all his holes himself, there seldom or never are any. Hand stoping is bound to be more economical in explosives used, though not necessarily in work done, for the simple reason above stated—that if, say, in a stope of equal height it takes one plug of gelatine to break a hole with a burden of 1 ft., it will take not two but four plugs to break a hole with a burden of 2 ft., and nine to break a hole with a burden of 3 ft. This law is, of course, modified by the presence of heads and partings, but in practice I found it to hold true on the average. As I pointed out five years ago, the right understanding and appreciation of this principle at once shows us the lines along which our efforts in reducing the consumption of explosives in machine stoping in narrow stopes must run. We must remember first that with piston drills the actual drilling time bears a small proportion to actual working time, the rest being taken up by erecting and moving the machine. Mr. Wiseman suggests drilling one hole benches, but he will find that even if the miner's work does not suffer through the time he has to lose from the supervision of his machines in order to erect bars on other benches, it will take at least ten minutes, as I have proved myself, even in a fairly flat stope of 30° to perform every move, and four or five moves during the shifts means 40—60 minutes running time lost, which means at least one good hole lost, so his remedy is as bad as the disease. We must also remember that it is the starting and first 2 ft. of a hole that takes the time in boring, as the drilling speed is roughly proportional to the amount excavated So that once we have our machine erected and a hole started it must pay to bore it as deep as possible—even if the diameter at the end is very small—if it be rightly placed. Mr. Wiseman thinks because he sees such a large number of 12 in. and 18 in. sockets standing that it is a mistake to drill these long holes, that gelatine often cannot be inserted in them, and that even if it can be, they will represent the diameter of a hand hole only. If this were the real trouble it could be got over by "chambering" the hole; for instance, it would be quite easy for the stoper to fire a plug of gelatine in the bottom of his back holes when he fires his front holes; but if you are going in for "bulling in" stopes with free faces limited in height, you may as well bore only short holes, as long bored holes will rarely break to the mouth but will "bull ring." This method is, however, quite useful sometimes.

In machine stoping we must try as far as possible to combine the advantages of hand stoping with the output of machine-drilling, for which we are prepared to pay some few pence per ton extra in explosives. To do this, first we must get the best work out of our machines in the time allowed, and can only do this by drilling long holes, and we must drill long holes in both narrow and wide stopes. This again I contended for five years ago, and it has only recently been admitted to be good practice. We must avoid any possibility of the first hole in any pair fired being held up. To allow of this we must acquire judgment by practice, and we must work with some knowledge of the principles of blasting.

We must have some idea, and a very good idea also, of what work our first hole is going to do. Unless we rightly judge just how much rock this leading hole will break it is clear that either our second hole will be too heavily burdened or the charge in it will not have half enough work to do. It is just here that half the inefficiency of machine work comes in. I have tried to make three sketches to illustrate this system of stoping. I claim no originality of practice in drilling zigzag holes, though I have always tried to show the reason of the practice, and the absolute necessity of it in machine stoping in narrow stopes. In very wide stopes the top and bottom holes are further apart and do not influence one another quite so much.

I evolved for my own convenience the system of boring 6 ft. and 7 ft. holes with small drills, finishing with a chisel of $\frac{7}{8}$ in. steel, in stopes of from 36 in. to 48 in. high, in the manner shown, and have tried to show how this method complies with the law of blasting already stated, in order to gain maximum economy of explosives and maximum drilling time and efficiency for the machine.

Fig. I. represents a bench one may see any day bored in the stopes of a Rand mine: the dimensions are, of course, relative and alter with varying circumstances such as hardness of rock and presence or absence of planes of weakness on foot

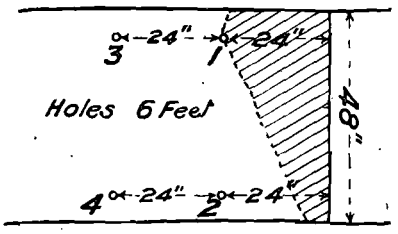


FIG. I.

and hanging wall. Here instead of showing Mr. Wiseman's 12 in. and 18 in. sockets I show 6 in. and 12 in. sockets. It will be seen that hole 1 has torn nearly all the burden off No. 2, which has wasted half its charge. Some miners have some dim realization that this occurs, and give No. 2 hole a slightly smaller charge, obviously confessing that they have bored the hole with too small a burden.

Fig. IA. shows a plan of the bench.

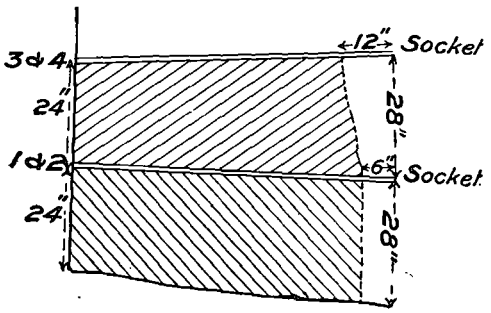


FIG. IA. (Tons Broken $=7\frac{1}{4}$).

In Fig. II. we have shown the correct method

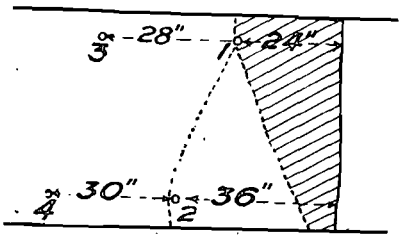


FIG. II.

of placing the holes. If hole 1 has a burden of 24 in. it will "slab off" the rock as shown, and hole 2 can be given a burden of as much as 36 in. I will admit that seeing a bench bored thus appears madness; but I have often proved that it is only common sense. Why, it may be asked, if hole 3 in Fig. I. has only 24 in. burden can you give hole 3 in Fig. II. 28 in. or 30 in. I reply, because for one thing the face of the bench from 1 to 2 presents a larger free face for the hole to break to and the hole is also partially under cut by hole 2, and for the same reason hole 4 can be given 30 in. or more. On looking at the plan Fig. II., it will be seen that all these holes are bored looking out slightly, *i.e.* having several inches less burden on the bottom part of the hole where there is less explosive than on the centre of the hole where there is more explosive. Boring with large drills and finishing off the hole with a chisel of $1\frac{1}{8}$ in. steel, it is usually sufficient in a stope over 40 in. to merely keep the hole strictly parallel to the face, where the ground is tight. Where lines of weakness occur in more broken ground this rule may be modified. For the same reason the position of the holes and the extent to which zigzagging of holes can be carried depend on

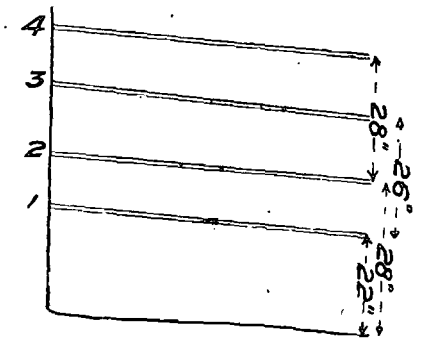


FIG. IIA. (Tons Broken $=9\frac{1}{4}$).

the toughness of the rock and the presence of lines of weakness on the foot and hanging wall. Were the rock solid granite breaking in the proper crater form, zigzagging holes would have little value; but the banket rock of our reefs more or less tends to "slab off" along planes at right angles to the walls, and every machine stoper should on first firing in a stope, fire the front leading hole only of each bench and go back and view the result. He would be surprised in every case. We will suppose that on trial the bench drilled on system (Fig. II.) with holes 6 ft. long gave satisfactory results, the holes breaking to the bottom when their diameter at the bottom was little over 1 in. We then ask ourselves: if it is as easy to blast a 6 ft. hole as a 4 ft., why not drill a 7 ft. or 8 ft. hole finishing up for $2\frac{3}{4}$ in. drills with a $\frac{7}{8}$ in. chisel, and a hole about $\frac{15}{16}$ diameter on the bottom? (Fig. III.). We may take it as a rough guide that in an extended charge occupying a borehole, each cartridge must be responsible for moving the burden in front of it. Then the first precaution to be taken is to see that the small diameter hole at its end has only just sufficient burden to allow it to break the rock in front of it, and this can be done by making the holes look out. It may be objected that by this one would always be tending to narrow up the face of the bench and to lose benches. I reply first that

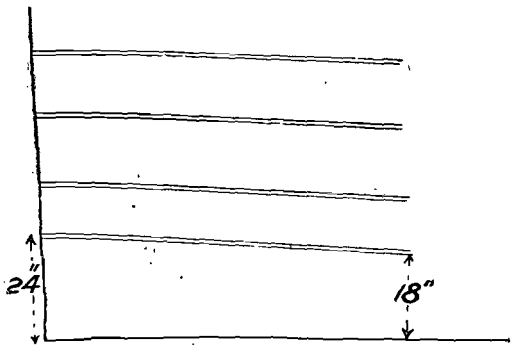


FIG. III.—Seven feet holes finished with $\frac{7}{8}$ in. steel, loaded in end with $\frac{7}{8}$ in. blasting gelatine. (Tons broken $=9\frac{1}{4}+1\frac{1}{4}=10\frac{1}{2}$ tons.)

five holes can often be drilled and the fifth hole can be used for squaring up. I always used the short shift on Saturdays for putting in two hole or three hole benches, to square up benches and keep the stope in shape. Hand labour should also be used in machine stopes. With holes bored as shown the trouble of "bull ringing" arises unless the charge is spread over the hole. All I wish to say on this matter is that if the practice of dividing the charge into two portions each with its separate detonators and fuse is objected to as being dangerous, then if the principles on which this method are founded are correct, and lead to economy and to maximum efficiency (as I believe they do), electric or some other system of firing must be resorted to.

At the Robinson mine with an 8 ft. high stope it is found possible to drill holes 8 ft. to 9 ft. long finishing with $1\frac{1}{4}$ in. chisel, and to fire them without leaving any stumps at all. The efficiency tonnage per machine shift broken is something over 20 tons.

The stope drill contest started off with the regulation (the wisdom of which I queried at the time) that the length of holes bored must not exceed 48 in. It was soon found in practice that the only way to break ground was to increase the length from 5 ft. to 6 ft. During this trial the miners might just as easily have used 7 ft. holes, and the results gained would have been still more favourable as compared with hand drilling. This, again, I had advocated in a paper written in 1905. It apparently cost the mining industry five years and £20,000 to learn what they might have found out by studying the *Journals* of this Society. In these I have always contended first that any relative inefficiency with hand stoping in small drill work was due (1st) to short holes being bored, (2nd) to easily remediable alterations in design of the drills in the direction of making them stronger while using practically the same types then in use, (3rd) to low air pressures at drill due to bad design, want of supervision of air pipe lines and overworked compressors, (4th) that men must be taught their work under practical conditions in underground schools.

Mr. Johnson is always insisting on the necessity of shift bosses being themselves efficient in the work they have to supervise. He will be pleased to hear that in the mines controlled by the Consolidated Goldfields of S.A., a circular was very recently sent round laying down the rule that no one was to be promoted to shift boss who could not show good cost sheets for contract work in machine and hand stoping. The keeping of proper records aids efficiency.

To Mr. C. D. Leslie belongs the credit of evolving a system of records by which the actual efficiency of the work done in any mine can be very fairly judged, and that too without adopting any new fathomage system. By its means the actual cost of obtaining the valuable ore, and its relation to waste rock mined, and to costs and values of total rock broken and to milling ore are easily ascertained every month. This system or a variation of it is, I believe, being largely adopted by all the groups.

Mr. W. Cullen (*Past President*): So far as I can recall I have never taken part in a discussion on a purely mining topic before, and the reasons for this are obvious, but Mr. Johnson's paper raises many points on which a mere tyro can speak. As far as I can judge, however, he has made his point regarding contract work with machines, but whether it pans out in practice only he and other practical men can tell. Judging from analogy his arguments are sound, and it does seem that in his hypothetical case both the best men and the company are done down. This is, however, only one phase of efficiency or inefficiency and since that is the universal doctrine, at the moment it behoves us to look around and see whether we really understand what we are talking about. In what I am going to say I think I may rely on Mr. Johnson's support at any rate. I should like to preface my remarks by saying that my talks with miners and others during the course of the recent election campaign have opened my eyes considerably to certain phases of a subject which may be generally termed "workmen's grievances." I purposely exclude "silicosis" and "ventilation," but to my mind it is farcical to talk of getting efficiency in mines when *the working conditions are known to be bad.* I have been down many mines but I know that if it were my lot to do physical work in some of them my efficiency would not be 10%. And then when we discuss this question in its broadest aspect we think only of white men, but surely the same thing applies to natives. I

heartily endorse Mr. Penlerick's statement that the ventilation installation *of the E. R. P. Mines, costly and all as it was* has already repaid the outlay. If one recalls the speeches of the chairmen of the different mining companies, one cannot fail to be struck with the constant references to the native, his number and his efficiency. Although in a sort of academic way we look upon the native as the one essential factor for economical mining, I am afraid that we frequently forget that what holds for the white man holds equally for the native. Neither a native nor a white man can work on an empty stomach, and good treatment is as essential for the one as the other.

I have some knowledge of what can be got out of the native under different conditions, and I have no hesitation in saying that on those mines where the management takes a personal interest in the native, where precautions are taken to see that he is treated as a human being and not as a dog, the efficiency is high and the numbers keep up. No! the native is our great economic factor: without him few of us would be here, and I venture to suggest that the subject forms one of the greatest unexplored chapters in the great subject of efficiency.

Mr. Johnson says rightly: "Keep men for long periods on the property, if possible, so that they get to look at the place as a home, they will then take an interest and pride in keeping the property to the front. Men thoroughly used to the mine will do more than newcomers."

Surely this is axiomatic, but I am afraid that on many mines the converse holds. As far as the native is concerned there is no immediate prospect of our securing continuity of employment, but Mr. Johnson's principle was our stock argument for getting the Chinese and keeping them as long as we could when we did get them. Here again the same principle must apply to white labour. We constantly talk about attracting a large settled white population, but on the mines are we really going about it in a proper manner? One of the grievances which I hear from my mining friends is the uncertainty of tenure. I don't believe the responsible heads really know how much unnecessary shifting there is among our working men and what an amount of dissatisfaction it causes.

As far as I have been able to ascertain a change of management frequently means wholesale changes in every department of the mine. Surely this is unnecessary. In the case of married men, settled on the property, if shifting must take place, let the men have a month's notice instead of the general one of 24 hours. Every praise must be given to certain mining companies for the excellent accommodation which they give their employees, but that is not all, as I have just indicated.

The question of the purchase of stores such as coal has been referred to frequently in our proceedings so I must say nothing on that subject, but quite apart from the different makes of explosives I am convinced that money, and a lot of it, could be saved by making a more intelligent selection of the explosive. To give an instance, practically only two gelatinous explosives are used on these fields—gelignite and blasting gelatine. An intermediate grade has been introduced from time to time under various fancy names, but at the moment practically none is used. I am convinced that there is a good wide field for an explosive of this nature if only the trouble were taken to try it. It would be a trouble, I admit, but nothing is achieved without it. It has been my constant experience to see an explosive of lesser strength than blasting gelatine doing better work than blasting gelatine itself. One cannot predict these things nor can one dogmatise about what is most suitable for a certain rock. One must simply try. Then on the subject of rock-breaking there surely must be right and wrong ways of doing it, but while thousands upon thousands are devoted to surface experiments, one hears very little about experiments on rock-breaking. Of course I know the answer to this, that it is mostly contract work, and a bad miner will draw very little at the end of the month, but that is only a very partial answer to a very important question. In this connection I do not know whether Mr. Johnson's scheme of tuition is feasible or not, but that something of the sort is necessary all are agreed. The best way, however, is to catch your man young, but so long as our mines have the reputation of reducing a miner's normal life to the present low figure this will be difficult of attainment.

Mr. S. Beaton *(Member)*: The author has dealt with a great many points in his paper regarding small economies in our mines. He lays special stress on the introduction of young inexperienced shift bosses and foremen. These men must be allowed to survive and grow older by means of good ventilation and dust-allaying appliances wherever dust is being formed at stations in stopes and drives. They will then have a commercial value that will make it hard for younger men to oust them from their positions. At present there are not enough technically trained experienced men to go round, which fact allows a number of young and inexperienced men to attain positions for which they are not qualified. Health conditions govern the question of the efficiency of

our underground men, white and black, by improved proficiency and ability to make a more sustained effort.

The author has started to economise at the wrong end, although a bit may be done there. The method of working pursued here is not conducive to economy; our method being an elaboration of the method evolved by what are termed the amateur miners of the early days, which method suited the conditions then prevailing, as labour was plentiful and mechanical appliances at a discount. With the vast improvements in mining machinery and not so much labour this method ought to be abandoned and a more efficient one substituted to take advantage of lower dips and the cheap power which is now available at the mines.

The following method of longwall working can be adapted to most of the mines of the Rand: From the main shaft drives ought to be driven under the reef on lines, all necessary bends having a large radius for haulage purposes. These drives should be sufficiently wide to allow of two tracks being put in. Drives should not be nearer to each other than 600 or 800 ft. on the dip of the reef, and even a greater distance may be employed. At 300 ft. on either side of the shaft crosscuts would be put to intersect all the reef, and at 600 ft. in either direction (east and west) other crosscuts will be driven. From those crosscuts intermediate shafts will be sunk on the various reefs, only clean reef being mined, all waste rock being blasted down and hauled separately, waste being used to pack in different parts of the mine. Those shafts would be sunk to the level below say 800 ft., when they could be used as self acting inclines. At 200, 400, 600 and 800 ft. on the intermediate shafts breast stopes will be started 30 ft. to 60 ft. wide depending on the condition of the hanging and other local considerations in the mine; at each breast a switch would be laid to each side to allow of the trucks being run directly into the stope, where a loop would be put in allowing a supply of empty trucks in each stope so that the shovelling gang may be employed continuously whilst hauling from the other breasts. An electric hoist can be employed for hoisting, the empty trucks hauling down the rope which is attached to the loaded trucks, and when hauled up the trucksare detached and put on to the main haulage and taken to the shaft. In working, the breast would be taken in a distance of 300 ft. when it would meet the one coming in the opposite direction. They would then be cleaned out and filled in with sand, the rails be taken up and all timber drawn. When the stope is full, another breast would be started, the track being laid on the sand and the switch shifted up to suit. The sand should be filled in on night shift so as not to interfere with the ordinary working of the mine; good packs could be built alongside the shaft to retain the sand in position. When only a thin pillar is left between one breast and the one above, this pillar may be stoped home towards the shaft leaving all the sand behind. The output of the mine would almost be obtained on one level with two reefs.

The following advantages may be claimed for this method of working:—

I.—Lower developing costs.

II.—Lower shovelling and tramming costs, as all broken ground would be blasted down to the track and no manual tramming to a greater distance than 300 ft.

III.—Better supervision, work being more concentrated.

IV.—Simpler ventilation and drainage.

V.—New mines would reach the producing stage in less time than at present.

VI.—Greater safety, no great stretches of worked out places being left to endanger the whole mine and surface works.

VII.—No pillars of reef need be left and no cost incurred in raising boxholes.

VIII.—Less timber and timbering.

IX.—Sand would be held safely in position and no mud rushes could take place in a wet stope.

X.—The output from any stope would not be limited by the number of shovelling boys it is possible to crowd into a stope.

VENTILATION AND HEALTH CONDITIONS ON THE MINES OF THE WITWATERSRAND, WITH SPECIAL REFERENCE TO THE VENTILATION SYSTEM OF THE EAST RAND PROPRIETARY MINES.

(*Read at August Meeting, 1910.*)

By S. Penlerick (*Member.*)

DISCUSSION.

Mr. Tom Johnson (*Member of Council*): Mr. Penlerick's paper shows plainly the great benefits to be obtained from increasing the ventilation in our mines. It is to be hoped that the experience of the East Rand Proprietary Mines will give others that encouragement necessary to try to better the ventilation of their own mines.

Although artificial ventilation is what should be aimed at, there is no reason why those who cannot spare the necessary capital to instal fans,

etc., should sit down and say there is nothing they can do to better the ventilation.

As the author pointed out, adequate ventilation does not depend solely on the quantity of air going into the mine, but greatly on how the air is coursed and distributed through the mine: as a proof of the above, the more we course our air the less the quantity circulating. Much can be done quite cheaply by putting up brattice sheets in the entrances of the upper drives between shafts, and an odd door or two, as shown in the paper, so as to conduct the greater part of the air to the lowest point of the workings, instead of letting it get away to the upcast through the upper drives. As the author points out, and as I mentioned in 1903, a standard of purity of air is better than a standard of quantity per man, but more air per man is needed in wide stopes than in narrow stopes, irrespective of quality.

The author does right to lay stress on the water question. As Mr. Judge mentioned, it is not only at the holes being drilled but in all dry and dusty places, that the water is needed, for after all is said and done, what is the percentage of dry holes drilled in a producing mine? It should not be 10%. To send cool air through a warm mine without watering the mine would only aggravate the dust question, for the air as it got heated on its travels would pick up quite a quantity of water, making the mine drier and dustier.

The Rand is not the only place that is troubled with a dust question, many collieries have a more dangerous dust than ours. Here the dust danger is more or less in the handling of the man most concerned, but in many collieries one man may either ignorantly or carelessly endanger not only his own life but the lives of every one in the mine: still in the United Kingdom the death rate from fire damp and coal dust explosions has been reduced from 0·65 to 0·1 per thousand during the last 35 years. If all concerned were to co-operate we ought to pull down our death rate much faster than this. Phthisis is not a necessary adjunct to mining, so if each one concerned will only do his little bit the death rate would be considerably lowered.

There is going to be a great disadvantage in using water in the hotter mines of the future, and I believe we shall have to put cooling apparatus in the workings to cool and dewater the air somewhat, so that it will not be saturated and at a high temperature before leaving the mine; this has been talked of in colliery circles for many years, but I do not think any installation has been put in yet.

The E.R.P. Mines are fortunate in being able to set a shaft aside for purely ventilation purposes. It is very desirable that one shaft should be used solely as a ventilation shaft, but every mine will not be able to do this. It is not a heavy job for an outcrop mine to fix up, but it is an awkward one for some of the deep levels, more especially the mines with all compound shafts. If the deeps cannot come to an arrangement with the outcrop mines to compound the ventilation, then the mines with compound shafts have a little problem to solve. With our present methods of hoisting and handling rock, men and material, we cannot very well fix the the fans on the surface. They will have to go underground, and where are they to be put?

As to the bad effects of compounded ventilation of two mines, I do not see where there should be any trouble; if the whole of the area of the two mines was used as one mine there would be no talk of exhausting bad air into the outcrop portion. The exhaust bad air from the two mines with a fan will be better than the exhaust air from either of the mines with only natural ventilation. I should be glad if the author in his answer would give us the speed of the fans; how and where the water gauge was taken (that is how far from the fan, and in what direction was the pipe of the water gauge pointing), also the H.P. of the prime mover, size of opening in the fan, temperature, and the natural water gauge. I notice from the figures on the sketches that with air at 63° F. the Sirocco is giving up about 22 h.p. to velocity; this is too much. I think it could be brought down to 6 h.p. or 8 h.p. This 22 h.p. represents about 16% of the water gauge. The Barclay, on the other hand, is only giving 6% of the water guage to velocity, which is not because the Barclay is a better fan, but simply because it has a much larger *evasée* chimney. I would recommend the author to take down the Sirocco chimney and rebuild it, giving it an area of, say, about 200 sq. ft. and save 14 h.p. or 15 h.p. I make the present areas of the chimneys from the figures on the sketches to be say, 100 sq. ft. for the Sirocco, and 170 ft. for the Barclay. If these two fans were tested it would be seen that the Sirocco is heavily handicapped because of its chimney. Another thing in fan testing, is that it is quite wrong to take the all-over efficiency of engine and fan for a comparison, as is so often done; efficiency of the fans should be taken alone.

Last year the late Mr. A. H. Stokes,* Inspector of Mines, England, read a paper before the Midland Counties Institute of Engineers on water gauges and velocities. He found by experiment that by putting a bend on the end of the pipe leading to the experimental water gauge, that with a normal height of 2 in. on the gauge on the

* Trans. *Institute of Mining Engineers*, Part 1, Vol. 38.

separation doors, by manipulating the bend he could get variations in height from 0·2 in. to 3·0 in. He found that with the same speed of fan and volume of air that if the area of drift varied so did the height of water gauge. It can be seen how important this is, for water gauge varies as velocity, and velocity varies nearly as area, and the h.p. in the air equals quantity multiplied by water gauge divided by 33,000, this last being used to get the efficiency of the fan. As fans become more general, and that will be soon, we shall have some of the same troubles in testing as they have gone through in other countries, for up to now I do not think a decent comparable manner of testing fans has been evolved; at least it looks like it from the published tables of fan tests. As an example of the curious things we hear of in this connection, a case was mentioned in the discussion on Mr. Stokes' paper, where a new fan was put in to replace a larger diameter fan in the same fan pit. The new fan was to give 62% efficiency, but on trying it, it only gave 50% efficiency; after the fan pit was altered to suit the smaller fan the efficiency was increased to 60%, the water gauge rose from 2 in. to 3½ in., but mark you, the air in the mine remained the same. It is things of this kind that we shall have to be careful of when buying and installing fans, or we shall be doing in this case what we are doing in other cases at present, that is, paying for paper efficiency. I notice that the decrease of temperature is only 2° or 3°. Is this an error, or is it because the difference of air temperature and rock temperature is small?

I am afraid that we shall not be in a position to carry out the author's recommendation contained in the last paragraph of his paper, desirable as it is. In the future deep levels, as members know, I hold the opinion that we shall have to move our faces much faster than we do at present, both on the score of safety and cost, and before we get well set on single shift work we shall have to turn to double shift again in the deeps. This is where I see the greater need for artificial ventilation, for although we shall get larger quantities of air by natural ventilation in these deeper shafts owing to their depth, the quantities will not increase so fast as the need for them.

In conclusion, I must say that the E.R.P. Mines deserve, and will have, the thanks of all for piloting the way towards mechanical ventilation, and the author also for the manner in which he has brought the matter before us.

Mr. T. Donaldson (*Member*) on behalf of Mr. W. Cullen submitted the following contribution:

Mr. W. Cullen (*Past-President*): This paper — as has already been remarked — is perhaps one of the most important which has ever been read before our Society, and it is to be hoped that all the other mining companies will emulate the excellent example set by the E.R.P.M., by installing systems for improving the ventilation. It will come as a surprise to many that the total cost is so very low, but I can well believe the author's statement that it has already been recovered by increased efficiency alone. Quite apart from this aspect of the case, however, it is a moral obligation on the the part of those who control the industry to see that no further time is lost. Of course, it may be claimed that nothing could possibly be done so long as the "Mining Regulations Report" was under consideration, but now that this report is issued, we can see that hardly anything new has been brought to light. Indeed, the report is mainly a compilation (though a most useful one) of information which for the greater part was already public property, with certain deductions from this compilation, and finally certain recommendations made naturally with the idea of improving existing conditions. In case of any misunderstanding I wish to say that in the main I agree with the findings of the Commission, but it does seem a pity that such an important report was delayed for so long a time through circumstances which were apparently unavoidable. This is the only word of criticism which I propose to make regarding the report itself, but it is worth while in passing to note the attitude of the press and the public. The press to my mind did its duty nobly by endeavouring to arouse the public to a sense of the magnitude of the issues at stake, and let us all admit frankly and freely that the public responded equally nobly. Unfortunately, however, the report was issued during a political campaign—a most unwise procedure in my opinion—so that the subject-matter assumed a political aspect: but by this means it became more widely known. To-day nothing is heard of it. All those fine humanitarian instincts which were aroused at the time of the publication are "as they were;" at least I think so. The indifference of the Rand to the loss or the value of human life is to me astounding and incomprehensible. Perhaps, however, I am altogether wrong and it may be that plans are even now being prepared for the installation of artificial ventilation on every mine. If so I shall be delighted: but I "hae me doots" and for the following reasons. Years ago Drs. Macaulay and Irvine initiated what I shall briefly call an Ambulance campaign. They showed us clearly what was required of us: we all admitted that many things were left undone which ought to have been done, but since these days can any one of us, managers, officials and workmen, say that the campaign has made much progress? I say

unhesitatingly, relying on my own experience alone, that the want of progress has been simply appalling. On some mines, which I have visited recently, prominent officials told me that they did not know whether there was any ambulance equipment on the mine, and others made the same reply about the oxygen cylinder, asking at the same time what its uses were. Now one can be excused if the matter involves a large expenditure or troublesome organisation but it involves neither. At the Dynamite Factory 70% of the employes have gone through a course of ambulance instruction—it is a condition of employment and has been ever since I was appointed manager—every department has a complete set of ambulance appliances, and it is the duty of the medical officer and the official in charge to see that they are always in order. Then again with regard to safety helmets or similar devices, mishaps within the past few months have demonstrated again, if that were necessary, that there should be at least two on each mine. But they would be of no use unless the men were practised in them and drilled in rescue work. Indeed, they can, in the wrong hands, become a veritable danger.

At the present moment there is sitting in England a Royal Commission on Mines, which has issued two reports from which I propose to quote freely, as they have an intimate bearing on our conditions here. The first one, issued in 1907, deals almost exclusively with safety helmets, their uses and abuses. The physiological tests, conducted by Dr. Boycott, are extremely interesting, and I commend them to our members. It is true that the mines referred to by the commissioners, when discussing the subject, are coal mines, but the accidental ignition of a case of explosives would furnish an analogous set of conditions to those which the commissioners had in their minds.

They say *inter alia*, page 7. "With the "exception of a few isolated instances of "the use of breathing appliances, little "attention was paid for some years to the "question in this country, and mining "engineers aware of the imperfections of the "existing appliances were apt to regard "discussion of their introduction into general "use as of theoretical rather than practical "value. Within the last decade interest in "the matter was revived by the construction "in 1900 of an experimental gallery for "testing life saving appliances at Altofts "Collieries—in order to accustom men to "the use of the apparatus in conditions "such as are likely to be met with under-"ground."

Page 8. "In some continental countries greater "attention has been given in recent years to "the use of breathing appliances. In "Austria this provision has been made "compulsory, in Germany, though no "regulations have been made by the Govern-"ment on the subject, a great deal has been "done voluntarily and many of the larger "collieries are provided with sets of "apparatus."

We have all of us fresh in our minds that terrible disaster at Courrières, and the excellent work performed by the German rescue brigade from Westphalia.

Then again, page 11. "We think that the "risks attaching to the use of the appliances "can be reduced to a minimum by a proper "system of training, and we desire to lay "particular emphasis on this point. In the "first place it is important for purposes of "training to select men who are most suited "for special work of this kind on the ground "of general intelligence, physique and "temperament. The men should then be "given a thorough knowledge of the con-"struction and working of the appliances "and should be accustomed by degrees to "wearing them, at first in an experimental "gallery and afterwards, when they have "attained proficiency, in actual operations "underground."

The report then goes on to recommend the establishment of rescue stations which would serve either one mine or a group, but that is a detail. I think myself that each mine should have two at least, and the necessary men to man them together with reserves. The expense would be infinitesimal, and I do not believe for one moment that it would be grudged by any one in authority.

The second report issued last year deals with a very large variety of topics, but rescue and ambulance work—especially the latter—bulk very largely. Incidentally two of the commissioners record their impression of a visit of inspection to some of the German rescue stations. At the Shamrock, one of the largest collieries in Westphalia, it appears that the rescue corps consists of 31 men and 50 officials, and that during 1906 there were 163 practices with breathing appliances, which lasted on an average two hours each. Only on two occasions were the practices interrupted by small defects in the apparatus, which shows that they have now attained a high degree of perfection. Of the 67 officials on the mine 62 are fully trained in the work of rescue. The commissioners also record that since their last report the provision of

breathing appliances has become compulsory in both Belgium and France.

I do not believe that rescue corps on anything like this scale are necessary here, but all must admit that something wants doing. It did occur to me that if the expense were such a bugbear as it appears to be, that the members of the fire brigade might be trained for this rescue work, seeing they are already accustomed to the use of safety helmets and ambulance, but there are obvious objections to this course not the least of which is the men's want of experience of mining conditions, for there is nothing more demoralising than feeling lost in the bowels of the earth. In this connection I may say that I have suggested to Professor Lawn and the Senate of the School of Mines that every mining student should be made to go through both an ambulance and a rescue course—the latter to include practice with breathing appliances. The Board of Trade have recently issued an order requiring that all candidates for the certificate of master or mate should possess ambulance certificates. How much more is it necessary with our mines! I shall avoid being discursive and therefore only quote further the recommendations of the 1909 report, under this heading, which recommendations will no doubt soon be embodied in a legislative act.

Page 209, par. 68. "*Use of Breathing Appliances.*—Schemes for the systematic provision of breathing appliances and the training of men in their use should be pursued with greater energy.

67. "Every mine should be provided either with a properly trained brigade of its own or to have the right to call for a sufficient number of equipped and trained men from a rescue station."

69. "*Ambulance Work.*—The number of men qualified in ambulance work is comparatively small, and there is room for considerable improvement both in the training of the men and the provision of appliances. There should be co-operation between the colliery owners and the St. John Ambulance Association. The officials of a mine, including managers and under-managers, should be required to possess ambulance certificates."

70. "Ambulance appliances should be kept underground as well as on the surface. Proper arrangements for conveying injured men to the hospital ought to be made at all collieries.

71. "The Secretary of State should have power to make regulations specifying the nature and number of appliances to be provided in proportion to the number of men employed, the places where they are to be kept, and the number of men required to be trained in their use."

72. "*Organization of Rescue Work.*—A great deal could be done in systematising the arrangements necessary in the event of a serious accident. We suggest the appointment of committees of the colliery owners in each district to provide for the erection of rescue stations, the training of men, formation of brigades, etc., and we also recommend the issue of a handbook containing instructions in rescue work.

73. "Instructions to workmen in such matters should form part of the ambulance training, and the preparation of a text book for the use of workmen in such classes would be an advantage."

Of course all these recommendations show that we are after all not really so very far behind the Old Country, but it must be borne in mind that the conditions are vastly different. Here we live in a new country, we have started afresh, we are concentrated in a large area, and above all we consider ourselves very up-to-date. In the Old Country traditions, prejudices and trades unions have all to be taken into consideration when any change in working conditions is proposed, consequently many necessary reforms are neglected. For instance, take the mere question of change houses—the provision of which by the way is compulsory in Germany as it is here. In Great Britain there is a distinct prejudice against them, although here and there one finds them. To anyone who has lived in colliery district as I have, and known the revolting conditions under which the men live, the small uncomfortable dwellings, it seems incredible that they should prefer to return to their houses covered with dust and grime rather than leave it behind at their work. Such, however, is the case. Turning now to the Report of the Mining Regulations Commission we find that both ambulance and rescue work are given great prominence, and the recommendations are on all fours with those of the English Commission and in the Draft Consolidated Ordinance (Chapt. XV., p. 185) proposes to give them legislative sanction. As already said, whatever difference of opinion there may be about some of the recommendations there can be none about these, and I sincerely trust that they will pass as they stand.

Coming now to the question of ventilation, with which the author's paper is more particularly concerned, any one who has followed the local discussion for the past six years must admit that the E.R.P. Mines, Ltd., are to be congratulated on the excellent results which have been attained by its ventilation plant. I have no direct know-

ledge of the condition of the mine atmosphere 12 months ago, but all who now work under the altered conditions are agreed that they feel different men. One hears similar testimony from the Turf Mines and the Village Main Reef mine. I do not know whether an analysis has been made of the atmosphere of these latter two mines. If not, then I strongly recommend that this be done without delay—indeed, I go further to say that every manager ought to know the state of the atmosphere right throughout his mine. I am only saying now what I said over a year ago, but so far as I am aware little or nothing has been done, as we have had no results placed before the Society. It is, of course, not incumbent on those responsible to do this but results which indicate progress generally find their way into our proceedings. Might I therefore again suggest that the carrying out of this work is a moral obligation and should be done at once. It will have to be done in any case when improved ventilation becomes compulsory, as I am convinced it will be. In this connection I shall be glad to give all the advice and assistance which it is in my power to give, but at the same time I do say that a proper gas analysis survey will cost a good deal of money, because the processes are slow, special apparatus is required, and above all, very special skill in manipulation. There is another, but a minor aspect of the same question which I think has been forgotten. All will remember that the Report of the Miners' Phthisis Commission drew attention to the possibility of the compressors themselves being actual manufacturers of carbon-monoxide and since then we have been reminded here and elsewhere by the loss of a good many lives that this is no picture of the imagination. I wonder how many tests have been carried out in order to ascertain whether the compressed air is free of carbon monoxide or not, since the issue of that report. I venture to suggest few, if any; I personally know of none. I do not suggest for one moment that any is produced at all, but I do say that on a vital question like this we should have "clear consciences." Here again I must say that I cannot understand the general attitude to questions of this sort, because anything which affects the health and efficiency of employees should surely come first. Above ground we are trying to save pence per ton and spending thousands in experiments, while down below there is a possibility of saving shillings through health and efficiency alone—and all this, of course, apart from the moral aspect. I am convinced that those in authority are actuated by the very best motives and are sincerely anxious to do their best for all those under their charge, but at the same time I must confess that much more might have been done in the past.

Mr. H. Stuart Martin (*Member*): I had the honour to propose the vote of thanks for this paper, and at that time promised I would at a later date, when the paper was being discussed, add a few remarks on the subject laid before us.

The author's paper is illustrative of what has been done in mechanically ventilating those mines under his charge, and has certainly proved that there is little difficulty in adequately ventilating the gold mines on these fields on the same lines as at coal mines at home.

The cost of installation, a figure no less than £25,000 (already redeemed by improved efficiency), and the running cost of only 1d. per ton, are facts that certainly deserve our consideration. £25,000 appears to be at first sight a large sum of money to be spent on ventilation alone. However, in this case I have no doubt a considerable sum included in the estimate was spent underground in making the necessary connections and drives in an already large and extensive mining property which in its earlier days was not laid out for mechanical ventilation. Yet, on the other hand, the dykes running from N to S have formed excellent natural brattices, and by their presence have facilitated the splitting of the air.

I am glad to say, following close on the heels of East Rand Proprietary Mines, an interesting ventilation problem has also been successfully carried out at the Village Deep mine. The only interesting points of difference as compared with the East Rand Proprietary Mines are that at the Village Deep the fan is erected underground at a depth of about 2,000 ft. from the surface, the quantity of air passing through the fan is about 280,000 cub. ft. per minute with a W.G. of 2·7 in., the cost of installation, including cost of cutting fan chamber, air doors, etc., did not exceed £6,000, and the running cost on the tonnage milled, covering the whole area ventilated by the fan, does not exceed 1d. per ton.

There are five other mines of the Eckstein-Rand Mines Groups, viz., City Deep, Durban Roodepoort Deep, Crown Mines, Nourse Mines and Robinson G. M. Co., for which fans have been actually ordered, and in a short time others will probably follow suit. The fans vary in capacity from 50,000 cub. ft. at 1 in. W.G. to 250,000 cub. ft. at 4 in. W.G., some to be erected on the surface and others underground. Frequently it will be found more economical to instal two, or even more, fans in a mine rather than one large one, as, if only one fan is installed, the W.G. must be sufficiently high to circulate the necessary quantity of air through the workings with most resistance, necessitating regulators which should be avoided. The placing of a fan underground is not so good as on the surface, when possible and economical to do so. The

Village Deep fan was placed underground in order to gain the necessary quantity of air at a low W.G. The small free air space, and the enormous frictional resistances in the main rectangular vertical shafts, prohibited the placing of the fan on the surface.

Speaking generally on the question of the ventilation of mines on these fields, undoubtedly the outcrop mines with their numerous shafts and outlets to the surface have provided ample ventilation in the past and have not called for mechanical means for better ventilation. However, following the outcrop mines, with the deeper levels (some of which already exceed 4,000 ft. of cover), and with the prospect of greater depths and fewer shafts—working much increased areas and larger tonnages—and increasing rock temperatures, the question of ventilation is going to be a serious one.

Apart from the installation of mechanical ventilation, suitable shafts with minimum frictional resistances and maximum free air space, as well as carefully thought out plans as regards underground lay-out and ventilating roads, will have to be considered, otherwise it will be impossible to work the deep deep levels economically and under favourable conditions. In consideration of this fact, at the deeper mines controlled by Messrs. H. Eckstein & Co., main levels and incline shafts are being driven in the footwall. These ways will act as intakes, and being driven in solid ground a minimum loss of air due to leakage will result. These main ventilating roads incidentally become excellent haulage roads for cheap mechanical underground transport, and therefore cannot altogether be charged against ventilation costs.

The author lays much stress on the quartz dust—the havoc it plays amongst our miners, and I am sure we all agree with him that with united action by the mine managers and their staffs much might be done to prevent the diseases due to this cause.

Improved ventilation, however, does not assist us in keeping the dust down, but rather helps to make matters worse, dries the mine, stirs up the dust and carries the fine particles throughout the mine workings wherever the air currents travel. Ventilation does its part by carrying away the foul gases and gives us fresh air to breathe, puts energy into our workers and gives us improved efficiency.

I cannot agree with the author when he makes the statement (p. 60 of the *Journal*) that "the fine dust in suspension must be constantly carried out of the mine, and that this demonstrates one of the necessities for adequate ventilation." With the proper application of water dust can be arrested. I am under the impression that the real mischief maker is this fine dust, and if we are going to do any good this dust must be arrested at the seat of origin, and to do this successfully the whole Rand must act together. Dr. Aymard's apparatus no doubt does arrest the dust where machines are used, whether in the face of a drive, raise or stope, but the difficulty is how to get the miners to use it. Unless the whole of our mines co-operate, Dr. Aymard's apparatus, or any other apparatus which may be equally as good, is going to make but little impression and will die a natural death.

The author seems to consider it advisable to place brattices or pipes in all dead ends, but a better practice would appear to be, where possible, to work only single shift and keep winzes well advanced. By this means the dead ends would seldom be more than a few hundred feet in advance of the main air current, and the air used by the machines is sufficient to ventilate the drives, except in special cases where local ventilation may be adopted. One of the great difficulties in working in high temperatures is when the atmosphere is also saturated. When working at great depths with high temperatures the air should be kept as cool as possible and dry, and therefore the application of water whilst arresting dust should be limited as much as possible.

In respect to health conditions of our miners, I entirely agree with the author that the proper housing of our workers is most necessary, and I can say that much has been done by Messrs H. Eckstein & Co., in this direction, and we are exceedingly proud of the new villages now being built for our workers, and in time look forward to holding all the best men on the Rand in our ranks.

There are many points in the author's paper I have not touched upon that call for discussion.

I again thank the author for bringing forward this important subject.

Mr. G. Hildick Smith (*Member*) referred to an article by Dr. Haldane,* and said:—

Although, unfortunately, unable to be present at the meeting at which the author read his interesting paper on "Ventilation and Health Conditions on the Mines of the Witwatersrand," I have read that paper with greater interest than perhaps most metal mining men, owing to the fact that I served my apprenticeship to mining in the large collieries of the Midlands of England, where the question of ventilation is of paramount importance and the first rule of the Coal Mines Regulation Act, states that "an adequate amount of ventilation shall constantly be produced to render harmless all noxious gases, so that all

* See this *Journal*, pp. 227-230.

working places, travelling ways, etc., shall be in a fit state to work and travel therein." There is no reason at all why this should not also be the first rule in the local Mines' Regulations, and it is pleasing to note that this has at last been realized and systematic ventilation, as opposed to the old haphazard methods, is at last being adopted slowly but surely. In the installation of a mechanical ventilator, in the shape of any well known type of fan, we require for the fan itself, capable of producing 100,000 cub. ft. of air per minute, about £300. To this must added the cost of erection, ventilation doors, etc., the total cost of which, as the author remarks, is easily repaid by the increased efficiency obtained from the human factors in the mine, even if we assume that the only result of the ventilating current is the reduction of the wet bulb temperature alone. The production of a ventilating current presents no difficulties, but the proper circulation of this current underground in the mines of this field will be more difficult. When we come to the health point of view put forward by the author, we are dealing not only with an important question looked at from our point of view as mining men, but we are faced by a question of national importance and of vital consequence to the community at large. It is a well-known fact, I believe, that, on the whole, unhealthy parents bear on an average larger families than healthy parents. Unless, therefore, the phthisis and tuberculosis danger is tackled quickly and efficiently, we shall eventually have a community, in this new country, afflicted with consumption.

Going back now to the mining point of view, and looking upon the mines as the breeding grounds of the consumption, we may ask — What is going to be the result of the coming ventilation schemes? A general improvement in health conditions undoubtedly. But is it going to be the great improvement it appears on the face of it? Let us take the chief cause of phthisis—the dust —this opens the way for infective processes, etc., but is in itself apparently the primary cause. Prevent the dust getting into the lungs and cutting them, and we then have done a great deal towards the eradication of phthisis. We can never hope to completely eradicate it, but we shall have done all that can be done, so far as the present state of our knowledge goes. Are there therefore any known means by which we can absolutely prevent the dust from being inhaled by the miners? The answer I think is certainly, yes. An efficient respirator would do this. The author says in his paper, "I would here express my agreement with the opinion of the Mining Regulations Commission with reference to respirators. For continuous wear they are useless, although for occasional wear they are useful." How do the author, and the members of the Mining Regulations Commission know this? Have any of them in actual practice and for any length of time used respirators underground and noticed the difference in the state of their throats, nose or lungs as compared with the state of those organs after working over a long period underground without a good respirator. Have any experiments ever been conducted, with various materials such as sponge, gauze, etc., both when used wet or dry? This is the chief point I would like to emphasize. Experiments should be carried out by aspirating air containing dust, through various suitable materials, in order to ascertain which is the best material, of which an efficient respirator could be made. Personally I use a respirator of my own pattern, made from a sufficiently fine textured flat sponge, and I find that even in the work of shift bossing, which entails more heavy breathing than perhaps most other work underground, a respirator worn continuously causes no inconvenience in either breathing or talking. The coming ventilation schemes, as Mr. Stuart Martin remarks, will excite the dust, even with watering. Water pipes in practice will often be broken by blasting, etc., and as any breakage in the pipe lines will not directly affect the output, breakages of this description will always be the last job to be repaired, giving time for the ventilating current to dry up the stopes, etc., and pick up the finest and most dangerous dust as it goes along.

In conclusion, therefore, the respirator question should be gone into from a proper scientific standpoint, various types should be experimented with both in the laboratory and underground. An efficient one having been designed, it should be made a standard pattern, procurable by those of the underground workers at any rate who have sense enough to realize the danger to their health. This, quite apart from the question of fresh air and elimination of deleterious gases would tend to increase efficiency, as the class of miner who is careless of his health and seems to be anxious to suck in as much dust as possible and die quickly, will also probably be careless about his work. Given a sufficient time, therefore, the result would be that the sensible and careful man will still be mining after the careless men are "planted."

Mr. C. Toombs (*Member*): I would like to point out to the last speaker that the author described and gave an illustration of an apparatus used on the East Rand Proprietary Mines for accurately determining the dust floating about in mine air.

Mr. E. J. Laschinger (*Member of Council*): I should like to reply to Mr. Cullen that with

regard to the amount of carbon monoxide which might be present in the air delivered by compressors, that our immediate Past-President, Mr. McArthur Johnston some years ago (after there had been a few explosions with air compressors on the Rand), made some experiments as to the amount of carbon monoxide in the air used by compressors. He made these experiments at the Simmer and Jack and the Simmer Deep, and said he could absolutely detect no carbon monoxide at any time in the air delivered by compressors. I am glad to make this announcement, for there has been a doubt on this point in the minds of mining engineers on the Rand for many years. After these tests I felt quite satisfied that no troubles due to carbon monoxide in the air delivered from the compressors need be feared if proper care were taken in using good oil and in efficient cooling of the air during and after compression.

Mr. C. Toombs: Were these carbon monoxide determinations made by the iodine pentoxide method? If not, I am sure that negative results obtained by volumetric absorption methods are not of much value.

Mr. E. J. Laschinger: They were not made volumetrically. They were done by the method mentioned.

The President: Perhaps Mr. McArthur Johnston might publish his figures.

Mr. A. McArthur Johnston (*Past-President*) (contributed):—Mr. Laschinger is correct in saying that in some tests we conducted two and a half years ago, no carbon monoxide was present in the compressor air. The samples were taken when the plants were under full working load, and were obtained from the Simmer Deep new compressor plant at the Rudd shaft, with an indicated pressure of 100 lb., from the Simmer and Jack Proprietary Mines plant — pressure 80 lb., and from the Simmer East compressor plant when the pressure was 70 lb. The samples were taken by blowing the compressed air into dry aspirator bottles, and this air was displaced by mercury when being passed over the iodine pentoxide. The tests showed in each case negative results, but the samples examined were insufficient in number to entitle us to conclude that carbon monoxide was always absent from these plants. Our main conclusion was that our crusade against the use of low flash point or adulterated oils, which we had previously waged, had borne good fruit.

The President: I am able myself to confirm what you say. I made some experiments, and my figures were published in the report of the Mining Regulations Commission this year, as also air-analyses since installation of the Village Deep fan. Apparently, however, Mr. Cullen prefers to discuss these things without taking the trouble to examine the recent existing data.

Prof. J. A. Wilkinson (*Member of Council*): Before this discussion closes, there is one point of chemical interest to which I should like to refer. As is well known, this Society has for some years past taken a very lively interest in the composition of mine air. In the case in question very full details are given of a method of improving this by mechanical means, and this contrasts very sharply with the sparse details given, to show what improvements have been effected. These consist of a few average results only. Now I would ask the author to remedy this by stating in his reply the results obtained in the fullest possible manner, and thus to enhance considerably the value of his paper.

The President: I have great pleasure myself in supporting that request. It struck me as the only weak point in the paper. I must point out that the figures for average CO_2 which the author gives are practically impossible, or else they represent gases sampled in absurd places. The figures are much higher than any ever found on the Rand by any other experimenter. I think it would be as well too if he could explain why these two very large fans have only reduced the temperature by a few degrees, because apparently in Europe a fall of 10° or 15° might have been expected. Possibly the cubic volume of his mine may be the explanation as I notice that the ventilation has only reduced the CO_2 to ·15 in places where he has sampled it, whereas something like ·08 might have been expected.

The meeting then closed.

Contributions and Correspondence.

THE DESTRUCTION OF CYANIDE.

By their footnote on page 152 of the October *Journal*, Dr. Moir and Mr. Gray do not appear to understand the arithmetical reasons involved in the moisture figures given by Mr. O. P. Powell, in reference to sand-filling.* Mr. Powell is now absent on a holiday, but the following tabular statement shows in detail the correctness of his calculations.

% Moisture in Pulp.	Tons Sand in Pulp.	Tons Water in Pulp.	Tons of Pulp.	Tons water retained in Sand at 12% moisture.	Tons of drainage water to be pumped.
30%	1,000	428	1,428	136	292
40%	1,000	667	1,667	136	531
50%	1,000	1,000	2,000	136	864

* See this *Journal*, August, 1910, p. 78.

November 23, 1910. G. O. Smart.

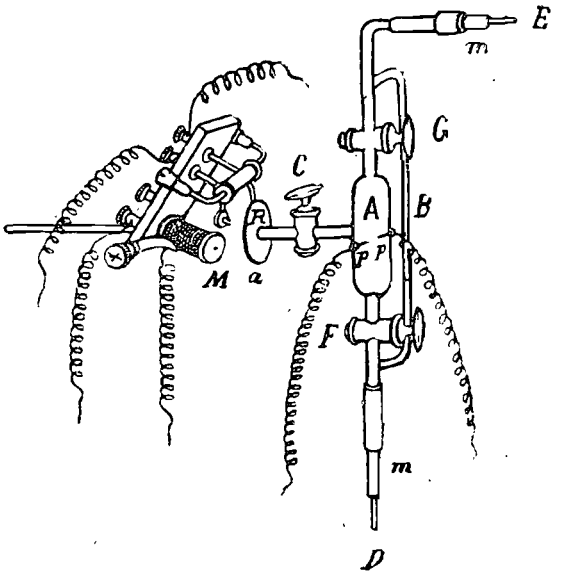
Notices and Abstracts of Articles and Papers.

CHEMISTRY.

DETERMINATION OF SULPHUR IN PYRITES.—"The most important sources of error in the determination of sulphur in soluble sulphates, under the conditions usually obtaining in practice, have been investigated by the authors and are found to be the solubility of the barium sulphate precipitate and the occlusion by it of sodium or other sulphate and 'free' sulphuric acid which is lost on ignition. In making a determination, it is recommended that nitrates, chlorides, and ammonium salts be, as far possible, avoided and that the following conditions be observed:—The solution, measuring 350 cc., is acidulated with 2 cc. of 2% hydrochloric acid and heated to boiling; precipitation is effected slowly (about 4 minutes for 2 grm. of precipitate), with constant stirring, and the precipitate is allowed to stand for 18 hours, when it is filtered off and washed until 25 cc. of the washings show a barely perceptible opalescence with silver nitrate; the filter is subsequently burnt very slowly, and heating of the precipitate is continued until a constant weight is obtained. To obtain results which are accurate to 0·1—0·2% of the total sulphur, corrections, which are preferably determined by experiment for the case in hand, are applied for the errors mentioned above. A very good uncorrected determination may be made by precipitating rapidly, but in this case there is a partial compensation of variable errors, and the method is not so reliable as that of slow precipitation with subsequent correction."—E. T. ALLEN and J. JOHNSTON, *Journal of the American Chemical Society*, 1910, *32*, 588—617.—*Journal of the Society of Chemical Industry*, June 15, 1910, p. 692. (A. W.)

RAPID AND ACCURATE METHOD FOR THE DETERMINATION OF TITANIUM IN ORES.—"The method is outlined for the determination of titanium in rutile and iron ores is based on the the volatilization of the silica by hydrofluoric acid in the presence of sulphuric acid, evaporation to dryness and fusion with sodium carbonate and a little sodium nitrate to convert the iron and titanium to insoluble ferric oxide and sodium titanate, extraction with hot water to remove the soluble phosphates, sulphates and aluminates; solution of the ferric oxide and sodium titanate in hydrochloric acid, extraction of ferric chloride with ether; and then either reduction of slight traces of iron with sulphur dioxide, precipitation of the titanic acid by boiling in acetic acid solution, filtration and ignition to titanum oxide; or a colorimetric determination by means of hydrogen peroxide. The authors find that the volumetric method based on the reduction of titanium to the trivalent state and oxidation with permanganate gives low results. With the sulphur dioxide reduction when considerable iron is present, the titanium product is usually contaminated with some iron. The ether separation removes practically all iron from the titanium very quickly. Re-fusion of the ignited product with sodium carbonate for purification, which requires considerable time, is never necessary by use of this method. The final titanium oxide is pure, unless zirconium is present, in which case this element is removed by the usual phosphate precipitation. The method combines the colorimetric and gravimetric determinations and the colour comparison is made in solutions always totally free from iron. For convenience hydrochloric rather than sulphuric acid solutions of titanic acid are used. This method is accurate and not long."—O. L. BARNEBEY and R. M. ISHAM, *Journal of the American Society*, 1910, *32*, 957—962. *Journal of the Society of Chemical Industry*, Sept. 15, 1910, p. 1061. (J. A. W.)

EXPLOSION INDICATOR.—"The instrument is intended to detect in a convenient place and at a distance from the locality tested, the existence of explosive gases in mines etc. The figure shows the construction of the instrument which is an explosion



pipette, *A*, with safety packings of wire gauze at *m*, *m*, and provided with a by-pass, *B*. The end, *D*, is connected by a long 'composition' or tinned iron pipe with the place the air of which is to be tested, and the end, *E*, with a pump, so that a continuous current of the air is drawn through. The cocks, *F*, and *G*, are closed (the current still streaming through the by-pass) and the spark from an induction coil passed between the wires. If no explosion occurs, it is because the proportion of combustible gas is either above or below the explosion limit. Should it be above, then on opening for a while the cock, *C*, enough air will diffuse in to cause an explosion on sparking again. Should it be below, then if *F* and *G* are opened and sparks passed while the gas is streaming through, flame will be seen in a darkened room with a proportion of combustible gas considerably below that necessary for explosion. If the explosion be made to take place with the cocks *F* and *G* closed and *C* open, and the suspended iron plate, *R*, closing the side tube, the force of the explosion will drive R against the electromagnet, *M*, which will hold it, and electrical connections made by this contact may be made to ring a bell at the place tested, and thus give warning."—N. TECLU, *J. prakt. Chem.*, 1910, *82*, 237—240.—*Journal of the Society of Chemical Industry*, Sept. 15, 1910, p. 1084. (J. A. W.)

NEW REACTION FOR COPPER.—"An intense blue coloration is obtained when a dilute solution of a copper salt is treated with an alkaline solution of 1·2-diaminoanthraquinone-3-sulphonic acid. The latter solution is prepared by dissolving 0·5 grm. of

the 1·2-diaminoanthraquinone-3-sulphonic acid in 500 cc. of water and adding 40 cc. of a solution of sodium hydroxide of 40° B. The blue coloration is plainly visible with 0·0000019 grm. of copper in 1 cc. of solution, and even one-tenth of this quantity may be detected by means of the test. The reaction is characteristic of copper, other metals not yielding a blue coloration."—R. UHLENHUTH, *Chem.-Zeit.*, 1910, *34*, 887.—*Journal of the Society of Chemical Industry*, Sept. 15, 1910, p. 1085. (J. A. W.)

POTASSIUM CYANIDE AND SILVER NITRATE.—"The interaction of potassium cyanide and silver nitrate is generally represented by the equations:—

(1) $2KCN + AgNO_3 = KAg(CN)_2 + KNO_3$;

(2) $KAg(CN)_2 + AgNO_3 = 2AgCN + KNO_3$.

Experiments were made with silver nitrate solution of such strength that 10 cc. were just sufficient to produce a white precipitate of silver cyanide with 10 cc. of the potassium cyanide solution. If equation (2) be correct, excess of silver nitrate should be present after 20 cc. of the silver solution have been added. The author found however that 25 cc. were needed before a reaction for silver nitrate in the solution could be obtained with potassium bichromate or sodium arsenate."—J. C. BAILAR, *Western Chem. and Met.; Mining World*, Chicago, Aug. 13, 1910.—*Journal of the Society of Chemical Industry*, Sept., 1910, p. 1085. (J. A. W.)

THE SYNTHESIS OF AMMONIA.—Up to the present time the liquor obtained in the manufacture of coal-gas or from the coke ovens has been the sole source of ammonia, and its synthetic production has not been regarded as possible of attainment commercially. Attempts are being made to prepare ammonium salts from cyanamide, but even if these should meet with some measure of success the process would probably be too costly to prove remunerative. The Badische synthetic method is independent of electric power, and now that both hydrogen and nitrogen are obtainable with facility at a low cost it is by no means unlikely that the electro-chemical processes of manufacturing cyanamide and nitrates will find in it a strong competitor in the production of fertilizing material; but the available information hardly justifies a definite forecast, although there is apparently a fair prospect of success for the new process.

It is stated that the Badisch Anilin & Soda Fabrik has solved the problem of the direct production of ammonia from its elementary constituents, and will probably soon effect the preparation of synthetic ammonia on the commercial scale. Prof. Haber and M. R. le Rossignol, after much experimental work, found that at a high temperature and under a pressure approaching 200 atmospheres, the desired combination of hydrogen and nitrogen occurred, and the action was increased by certain catalytic agents, particularly osmium, but, in consequence of the high price of this metal, uranium is used in practical working. The difficulties involved were great, but the Badische staff appears to have been successful in grappling with them. The mixed gases are compressed and brought into contact with powdered uranium in a chamber where a pressure of 175 atmospheres is maintained at a high temperature, with the result that the union of a large part of the mixture of gases takes place. The ammonia and the residual uncombined gases pass into a freezing apparatus whence the liquified ammonia is withdrawn, while the unchanged gases pass through and with further quantities of hydrogen and nitrogen, are again treated in the pressure chamber. The heat evolved in the formation of the ammonia is utilized to assist in bringing the fresh mixture of gases to the requisite temperature. A satisfactory yield of ammonia is reported to have been obtained at the Badische works.—*Metallurgical and Chemical Engineering*, Vol. viii., No. 9, 539, Sept., 1910. (J. A. W.)

COMMERCIAL CYANIDE.—"Dissolve 20 grm. in 150 cc. of water, filter into a 250 cc. flask, wash, ignite and weigh the insoluble residue. Add $Ca(NO_3)_2$ to the contents of the flask, shake well, dilute to the mark, mix, let stand an hour, filter off $CaCO_3$ and wash. The amount can be determined by strong ignition and weighing CaO.

The filtrate and washings are brought to 500 cc., of which 100 cc. is taken for hydroxide determination. Add $Mg(NO_3)_2$ solution, let settle for an hour, filter off, etc., ignite and weigh MgO. The weight $\times 0·85$ gives that of hydroxide in 4 grm. of the sample.

For cyanide take 50 cc. of 500 cc. dilution, add 5 cc. strong ammonia and 5 cc. of the KI indicator. Then titrate with standard $AgNO_3$. Cl factor $\times 0·7324 = CN$ factor.

For Cyanate. To 50 cc. of the above add excess of strong $AgNO_3$ to precipitate all Cy, CNO and Cl. Shake well, filter and wash with ice-cold water. When washed, place a clean flask under the filter, and pour over the filter 10 cc. of standard HNO_3 made up by mixing 100 cc. HNO_3 with 900 cc of water. Titrate acidity with standard NaOH. Each mol. of HNO_3 neutralized corresponds to 1 mol. CNO.

For Cl. Heat up 1 grm. of the sample in a porcelain crucible and add gradually a mixture of 5 parts Na_2CO_3 to 1 part KNO_3 until all CN is destroyed. Cool, dissolve, acidify with HNO_3, filter, and determine Cl in the solution by titration.

For K and Na. Decompose 0·25 gm. by heating with HCl, finally evaporate to dryness and weigh NaCl + KCl. Determine K by $PtCl_4$.

Commercial cyanide containing more or less NaCN is usually preferred, the lower combining weight of Na giving the greater proportion of CN."—*School of Mines Quarterly*, Columbia University, April, 1910, p. 280. (A. R.)

METALLURGY.

LABORATORY SCREENS.—"In order to form an opinion as to the results of most metallurgical and engineering operations, we have accurate weights and measurements and methods of analysis; the qualitative result of the operation of grinding ore, on the other hand, is not so easily subjected to measurement, and the various systems proposed have so far received no general sanction. The suggestions here made are an attempt to reconcile the conflicting elements in former suggestions, and at the same time to present a system which has merits hitherto wanting; also to present a practical mechanical method of making screen analyses.

There are two phases to the problem:—

1. The measurement of the result of crushing different ores in the same machine.

2. The measurement of the result of crushing the same ore in different machines, or with different adjustments or combinations of the same machine.

Having decided to arrange a sample of crushed ore into portions, the individual particles of which have as near as may be the same size, and later perhaps to subject the different portions to chemical analysis, we are immediately confronted with the questions:

1. How many screens?

2. What sized holes in each?

Thesis.—In laboratory screen analyses of crushed ores how many screens shall be used? What size shall the apertures in the screens be? and, as a corollary, what relationship shall the holes in one screen bear to the holes in the successive screen?

Imperfections to Date.—Rittinger's series is too short, Richards' series is too long, De Kalb's series is too short and disappears too soon, and all three are based on the metric system. The I.M.M. standard is not a regular series.

Cube Root Series.—A geometric series of diameters of apertures for a set of screens for laboratory testing is now proposed;* its advantages are pointed out, a comparison of all the series available is made, and various curves are plotted to show their merits and demerits.

Based on $\sqrt[3]{2} = 1{\cdot}2599208635$.

Inches.	Diameter of Aperture, Millimetres.	Meshes per Inch, roughly.
1·0000	25·3995	0
0·7937	20·1585	1
0·6300	15·9999	1
0·5000	12·6998	1
0·3969	10·0793	2
0·3150	7·9999	2
0·2500	6·3499	2
0·1984	5·0396	3
0·1575	4·0000	3
0·1250	3·1749	4
0·0992	2·5198	5
0·0787	2·0000	6
0·0625	1·5875	8
0·0496	1·2599	10
0·0394	1 0000	12
0·0313	0·7937	15
0·0248	0·6300	20
0·0197	0·5000	25
0·0156	0·3969	30
0·0124	0·3150	40
0·0098	0 2500	50
0·0078	0·1984	65
0·0062	0·1575	80
0·0049	0·1250	110
0·0039	0·0992	130
0·0031	0·0787	160
0·0025	0·0625	200
0 0020	0·0496	250

To state the problem as simply as possible—We will start with an aperture of 1 in., and divide 1 in. by the $\sqrt[3]{2}$; this gives us our second aperture. Dividing the second aperture by the $\sqrt[3]{2}$ we get the third aperture, and so on down to the 28th term of the series, where we reach the limit of effective screening in a screen with an aperture of 0·0020 in. (roughly, 250 mesh). The decimals in this series were calculated accurately in five places, and when the fifteenth term of the series was reached it came to 0·03937 in., which decimal will be recognised as the legal British and American equivalent for 1 mm. Here, then, is a series which may start from either 1 in. or 1 mm. and, by varying the successive apertures so that they are in ratio of $\frac{1}{\sqrt[3]{2}}$, get a series of screens which obviates this and all other objections urged. Between the limits of 0·0197 in. screen (roughly 25 mesh) and 0·0020 in. screen (roughly 250 mesh) we have:—

Common series	...	...	4 screens
Rittinger's ,,	...	...	7 ,,
Richards' ,,	...	...	14 ,,
De Kalb's ,,	...	...	6 ,,
I.M.M. ,,	...	...	11 ,,
Cube Root ,,	...	...	11 ,,

The I.M.M. and the Cube Root series have the same number of screens, but the I.M.M. series is so unevenly spaced that much objection can be raised to it.

The Cube Root series proposed obviates the objections which have been raised from time to time to all the other series."—T. J. HOOVER.—*Bulletin of the Institution of Mining and Metallurgy*, May 9, 1910, p. 1. (H. A. W.)

* Several series of screens have been suggested for use in laboratory sizing-tests, but none of those in use were satisfactory for a certain purpose in connection with work for the Minerals Separation, Ltd., of 62, London Wall, London, E.C., on their flotation process. After a large amount of research this new series was decided upon, but on looking over the literature of the subject further, and after this paper was written, it was found that this series was mentioned by Robert H. Richards in a paper read before the American Institute of Mining Engineers, referred to in the bibliography herewith, but he does not advise its use.

GRADING ANALYSES AND THEIR APPLICATION.—"THE PRESIDENT thought it would be convenient to discuss these two papers together. He would therefore ask Mr. Theodore J. Hoover to introduce his paper,* after which Mr. Walter McDermott would introduce Mr. Stadler's paper.†

Mr. THEODORE J. HOOVER gave a brief summary of his paper, in the course of which referring to the series proposed by the Committee of the Institution which he had criticised, he said that all standardisation schemes had to have a start, and had not the Institution made a start, a considerable time might have elapsed before anyone would have taken the trouble to figure out any series at all. The action of the Committee certainly gave an impetus to standardisation in general.

The work which gave rise to the paper necessitated a mechanical appliance, which he had invented, in order not to have to rely on ordinary hand-shaking screens. Having had considerable experience of a more or less unsatisfactory character with patents, he had decided not to take out a patent for the machine, but to make a present of it to the engineering profession.

He wished to point out one advantage of the series, based on the $\sqrt[3]{2}$, which he recommended for adoption, namely, that the English and metric units of measurement were brought into a semblance of harmony. It was purely a coincidence; and as far as he knew, throughout the range of mathematics there was no other occasion in which the English and metric system were brought into so near a relationship.

Mr. WALTER MCDERMOTT, in introducing Mr. Stadler's paper, said that by comparing the sizes of the apertures, it would be found that a large portion of the author's grades agreed very closely with those of the I.M.M., and therefore, in practice, all his conclusions could be put in use and practically applied by the use of the Institution's standards only. At the same time, it was a scientific paper. It was an attempt to give a value in energy units to each grade, and, of course, it was worked out scientifically. It was worked out to five places of decimals of an inch, and while they knew that, in practice, such refined measurement would be impracticable and useless, still, for the purpose of a scientific paper establishing standards, it was perfectly justifiable.

* See this *Journal*, p. 219.
† See this *Journal*, Vol. X., May, 1910, pp. 382-390.

Starting from a different requirement, and proceeding by a different process of reasoning, the author arrived at exactly the same conclusion as Mr. Hoover, their standard list being alike, except that Mr. Stadler worked to five places of decimals and Mr. Hoover to four.

The author had proved by a number of experiments that although in breaking various kinds of rocks the forms of the particles were very irregular, still, in effect, and on the average, they were subject to the same laws as though broken into perfect cubes, so that he was able to value the same various grades by what he called energy units, representing the proportional power required in crushing.

There were a great many figures and formulæ given for applying the author's conclusions; but in effect the argument was this, that the energy required in making a fine grade of material was more than the energy required in making the same quantity of a coarser grade. For example: in comparing the efficiency of the work of a stamp-mill with the work of a set of rolls, it was not sufficient to compare simply the quantity which each of them would pass through a 60-mesh screen; but the product must be graded below the 60-mesh into all the various sizes, even to below the 200-mesh, and the relative value allowed for in energy units for each grade, and in accordance with the proportional weights of the grades.

Many present had, of course, recognised generally the principle involved in this consideration of the efficiency of crushing machines, and it was referred to in the discussion of Mr. Caldecott's recent paper on coarse crushing by stamps; but Mr. Stadler had translated what was broadly recognised into actual figures, and had given a formula by which they were enabled to express duty done in actual energy units, and therefore to make comparisons, as the author did for instance, between a stamp mill and a tube mill.

Mr. Stadler's conclusion, which led to a great deal of discussion in Johannesburg, was that the tube mill was very inefficient mechanically; that the stamp mill was not credited properly with the proportion of fine grades it produced; and that the actual finishing work done by the tube mill was over-estimated as compared with the mechanical energy consumed in that work. Of course, this conclusion applied only to mechanical efficiency; and in itself, therefore, was only one element in calculations determining choice of machines.

The paper was one to which it was difficult to do justice in a summary; but the objects in view and the theoretical skill shown in working them out would command the admiration of all.

Since the paper had been written, Mr. Stadler had sent over the following note referring to a different method of estimating the efficiency of crushing machinery suggested by Mr. Caldecott, which, with the permission of the President, he would read as having some bearing on the paper.

Mr. STADLER's communication was to the following effect:—

Since the above paper was written, Mr. Caldecott has published a method of his own for computing efficiency. He has introduced a new 'nominal crushing unit,' which he defines as a stamp with a running weight of 1,250 lb. and an assumed new weight of 1,350 lb., dropping 100 times per minute, with set drop of 8 in. and actual drop of 7½ in.

It is difficult to see the necessity or desirability of introducing a new unit for the energy of different stamps, when the two well understood and scientific units of foot-pounds and horse-power are available. The value in nominal crushing units of any other stamp of varying conditions has still to be determined by the ratio of foot-pounds of the particular stamp to that established as the nominal crushing unit; so that the usual units might be taken as the standard at once. It is further assumed that a tube mill 5½ × 22 ft. is equal to 30 of the nominal crushing units and entails an equal consumption of power, taken as about 100 h.p. It is admitted that this ratio of tube-mill to stamps 'varies somewhat according to the conditions of operating' It would perhaps be more correct to say that it varies very largely with changes in conditions of working.

It is not correct to take the production of −90 mesh as a measure of the work done in crushing, as Mr. Caldecott does in saying that 'the more tons of this material that can be obtained per stamp, or per tube mill, or per horse-power, the better.' Such a basis of calculation gives undue credit to the 'finishing' work of the tube mill on the already finely crushed feed, and the high duty of the preceding crushing to the fine condition is ignored.

By similar reasoning, Mr. Caldecott might have said that the ultimate object sought is the reduction of the +60 mesh, and in this case he would have arrived at the opposite conclusion, namely, that practically all the work done in crushing is performed by the preliminary crushers, and little is left for the tube mill to do. The figures obtained by his method do not in effect show any advantage in favour of coarse crushing; indeed, according to his figures, the production of −90 grade is for the same stamp weights about the same for any coarseness of battery screens; whereas, by my system of calculation by energy units, the greater efficiency now being claimed for coarse crushing stamps is clearly shown, as given below for a stamp of 1,250 lb. running weight:—

Mesh and Aperture of Battery Screen.	Duty per 24 Hours.	Mechanical Value of Pulp.	Relative Mechanical Efficiency per h.p.
9 mesh, ·272 in.	13·0 tons	18·0 E.U.	234·0
200 ,, ·053 ,,	7·5 ,,	21·5 ,,	161·3
600 ,, ·028 ,,	6·5 ,,	23·2 ,,	150·8
1,200 ,, .017 ,,	5·2 ,,	24·3 ,,	126·4

Mr. W. MCDERMOTT, in opening the discussion on the two papers conjointly, said it was rather curious that they should have had two papers both of which were based on exactly the same proposed standard of laboratory screens; but the two papers were different in object, and could therefore be criticised in a different manner.

In the Transvaal and other places there had been criticisms of the I.M.M. screen based on what seemed to him to be a misapprehension of what was intended. The I.M.M. standard was practically a tool; it was the introduction of a uniform tool into practice which had been so varied in the past that there had hitherto been no means of comparing results between engineers. When engineers had written in the past about grading tests by various mesh screens, no one was able to judge exactly what the sizes of the apertures of the screens were; the meshes had been often specified entirely without regard to apertures.

Now in Mr. Hoover's case, and in the case of some other criticisms, the other extreme had been gone to, of describing entirely by apertures, and that to

an extent which was not scientifically warranted by the conditions of laboratory tests. He thought no one could question that Mr. Hoover was absolutely correct in his theoretical conclusions. His standard table was a better one than the I.M.M. standard theoretically; and yet, if this column of figures and apertures were examined beside that of the I.M M. it would be found that in only a few instances was there any departure which amounted in actual practice to anything worth mentioning. By omitting the 70 and 90-mesh screens from the I.M.M. table—which any engineer could do at his discretion—there was really nothing left of Mr. Hoover's improvement except the introduction of a 25-mesh size.

In his opinion, any discrepancies between the I.M.M. table and Mr. Hoover's ideal one were far more than outweighed by the inaccuracies which existed in the use of any screen as a tool, and particularly of some of the suggested screens. The screen was not a scientific instrument; it was a mere approximation, an empirical method of getting grades which experience had shown were useful divisions for the various purposes mentioned by Mr. Hoover. But he thought that in not a single one of the uses to which Mr. Hoover had referred was the accuracy at which he aimed either obtainable or necessary.

The inaccuracies due to the actual method of using screens; to the weaving; to variations in time; to the nature of the material; were vastly more than the theoretical advantages claimed by the author over the standards adopted (after consultation with engineers all the world) as useful average grading sizes for laboratory use.

Of course, the slight departures from regular progression between the I.M.M. apertures when translated into a curve might appear revolting to a pure mathematician; but what did such departures actually amount to in size of aperture? He could not help thinking that in those refinements regarding screens the main object of laboratory grading was lost to sight. After all, a grade of particles contained all variations between two different apertures, whatever the standards adopted; and, therefore, extreme scientific accuracy was unattainable, and screening could only be a rough classification.

What was wanted chiefly was a standard by which they could translate the work done by a man at one end of the world into the knowledge of a man at the other end, so that they should both be working on the same basis; or that at least, if a man liked to use another set of screens but translated his results into the nearest I.M.M. standard, people of more limited devotion to pure science but with greater interest in its practical application, would be perfectly satisfied.

All the conclusions which had been reached by Mr. Stadler and Mr. Hoover would be equally well understood and applied if the nearest I.M.M. screens were mentioned.

On that point he might be permitted to say something on the subject of the use of the term 'mesh.' Of course, to the extreme scientific mind, the use of the word was still objectionable. It was said: Why should we use the word 'mesh' when it means nothing at all? Well, in the case of the I.M.M. standard it did mean something. It now meant the same all over the world, whereas formerly it was not definite. He thought all those who used screens would agree that it was very much easier to think in mesh. He found it impossible to either remember or realise four places of decimals of an inch; it conveyed nothing to his mind. But with the knowledge that the I.M.M. screen was one in which the aperture was equal to the wire thickness, and that a 40-mesh screen was double that of the 80-mesh screen, they had a simple workable standard with a consistent ratio between mesh and aperture. It was based on something which could be visualised in one's mind's eye.

Mr. H. L. SULMAN said that it was a coincidence that they should have two papers on the same subject, involving questions of considerable complexity, upon which the authors arrived at markedly similar conclusions—these being in apparent opposition to the recommendations of the Institution of Mining and Metallurgy.

Messrs. Hoover & Stadler exhibited an uncanny agreement in the standard they independently adopted for seriating mesh apertures. That both authors without collaboration should select the cube as the typical geometric particle, and the cube root as the function to determine particle volume and aperture area in a new scheme for the grading of ore, at first sight appeared to imply a mathematical principle from which there was small chance of escape.

The question to be settled was: shall a simple, if arbitrary, scheme of apertures primarily dominate the size of the particles; or, shall some chosen one of several irrelevant functions, derivable from a geometric ideal, determine the size of apertures through which particles of a wide range of shape and irregularity must pass!

Now as to the cube as the ideal geometric particle, Messrs. Hoover and Stadler, in common with Richards, Rittinger, and other authorities who had suggested mathematical standards, appeared to impute some magic property to the cube, taking it to represent the solid geometric form necessitated by a square aperture. He could not accept that view. Cube particles were conspicuous by their absence in any crushed ore; even were the latter derived from cubic minerals such as galena, or pyrites, it produced nothing nearer than irregularly shaped tablets, etc., of wide variation in configuration.

Apart from punched and slotted screens, the simplest way to produce a sieve was to plait two parallel series of wires at right angles, thus obtaining square holes; that was the original of the cube ideal.

Had it been found easier to weave a sieve as a cane seat is woven, and to make hexagonal or octagonal in place of square holes, the mathematical ideal for aperture seriation would be founded upon a geometrical solid with a hexagon or octagon as the cross section. With punched screens the natural geometrical figure selected would be the sphere, with slotted screens some form of the prism, rhomb, or slab; thereafter in each case would follow the choice of the most *convenient* function from amongst the many appertaining to such geometrical shapes.

It was obvious that no seriation based on the functions of a cube could apply with the same mathematical cogency to circular holes or to slots; on the other hand, any rational series must comply equally with every shape of aperture.

The eye was unable to compute variations in average particle *volume* with the readiness it could appreciate variation in average particle *diameter*.

This had been recognised in formulating the I.M.M. series; their fundamental difference from the authors' recommendations was that particle size shall be quantified by average diameter, and not by volume or cross area. But, when all was said and done, every mathematical scheme had finally to express its apertures by fractions (more or less cumbrous) representing *widths*.

Not only was there an absence of any directly useful relationship between a selected function of a cube and the particle passing an aperture founded upon such, but a little consideration would show the cube itself to be inadmissible as a basis for determining the areas of even square holes.

Imagine a screen with square apertures of absolute regularity, a number of cubes placed upon it which would just pass through them (*i.e.*, cubes with sides of smaller area than the holes), and the sifting to begin. Not a single cube would pass through; it would require considerable dexterity to place one so that it would drop through its aperture. As soon as movement commenced the cubes would tilt cornerwise and become jammed; their *solid diagonals* instead of their sides would tend to become parallel with the plane of their meshes, the "screening" of the cubes ceasing before it commenced. The selection of the cube as the natural geometric solid to pass through a square hole of equal side area was an incorrect mathematical premiss. The only regular shape which would pass a square aperture of slightly greater width without jamming was the sphere.

This aspect received support from Mr. Stadler's experiments. No screen, taken singly, meant anything definite in relation to grading; it merely determined what particles should pass or be rejected by it; for the grading effect it must be considered in connection with the next below it in the series, the mean of the two apertures being taken. Thus, the grading effected between a 10 and 20-mesh screen would be the retention of particles of an average diameter corresponding to 15-mesh, given a uniform relation between the meshes of the screens.

In the case, however, of hypothetical cubes graded between any two sets of apertures, Mr. Stadler had shown that a large empirical deduction had to be made from the *mean* cube value before any approximation to the actual size of an ordinary (Rand) ore particle could be reached.

Mr. Stadler conducted nine experiments, and found that the mean cube content must be reduced to an amount varying between 58·3% and 66·9% of the theoretical value, and adopted the average figure of 61·4% for Rand ore.

But there was no mathematical relationship between a cube and 61·4% of a cube; in other words, the assumed cubic relationship was exploded by the quantifications which Mr. Stadler had made as to actual particle volumes.

Again, 61·4% (the mean between fairly wide figures) was only the average observed in regard to a particular quartzose ore; with differing rocks this empirical figure would likewise differ widely. Given on the one hand an ore of, say, igneous origin, and on the other, one of schistose or micaceous nature, one would find, respectively, long splintery particles or flakes, or, so to speak, minute lead-pencils or pancakes as extremes.

It was evident that the volume constant to be applied to the hypothetical mean cube as determined for Rand ore, would here suffer further distortion; *i.e.*, a fresh empirical constant was necessary for each and every ore.

He would ask Mr. Stadler for information as to the manner in which the particle volumes, ranging between 58·3 and 66·9%, were determined. It seemed that these could only be arrived at by some method of counting the grains retained between two given screens; details of the method adopted by Mr. Stadler would be of great interest.

It was curious to notice that 61% of a cube was nearer to the volume of a sphere (52·36%) than to that of the original cube, viz., 100%; the seriation proposed by Messrs. Stadler and Hoover was thus actually closer to the sphere in practical grading effect than to the cube.

Of course, he did not suggest any fresh seriation based upon the sphere; he merely used the illustration as a *reductio ad absurdum* in disposing of the cube as the mathematical basis.

Considering now the selection of the particular 'function' of the cube; what was the overpowering reason for choosing the *cube root?* Rittinger adopted the square root, and Richards took the fourth root of two; the authors have given good reasons for discarding these. But there were plenty of other fractional relations which might be selected in order to get any desirable or varying number of sieves between two given limits. It merely needed the application of a simple formula to determine this. The cube root was only a fraction, and a somewhat lengthy one, and stood in no necessarily simpler relation to a cube than did any other decimal or fraction. It happened, coincidentally, that the fraction, which in this case represented the square-root, gave a series of the same number of screens as had been recommended by the Institution between the same limits; it was, therefore, merely a convenient *choice* after all, and no compulsory mathematical basis was involved. This, to him, seemed to dispose of the mathematical argument.

He would next deal with the practical limit of accuracy attainable by any screen series, and refer to Mr. Hoover's photo-enlargements of typical sieve apertures. The author had adopted the method, which he (Mr. Sulman) had used in connection with the I.M.M. investigation in 1907, in showing the wide range of error inseparable from the most carefully woven material. Mr. Hoover had also given the formula by which to compute the average diameter of the apertures, but refrained from drawing attention to their great variations.

In the photo-micrographs he made three years ago, he found a difference of over 22% between the largest and the smallest apertures in a given screen; he thought it probable that the differences shown in Mr. Hoover's photographs would be larger.

When one introduced this inevitable error into any series of precise aperture widths, the result was startling. Assuming a variation of 20% only and dividing this equally above and below the normal, it would be found that an aperture of 1 mm. in the cube root series would in some cases be increased to 1·1 in an upward direction, and decreased to 0·9 in a downward direction in others. The next screen of 0·7937 mm. would similarly vary between 0·864 and 0.71, and so on. In other words, owing to the unavoidable differences in weaving screens, whether of wire or silk, the mesh extremes almost overlapped. That did away with any theoretical accuracy, whatever the proposed basis.

Although both authors advocated the cube root seriation, there was a curious difference in their conclusions with regard to the Institution series. Mr. Hoover was very severe, and could say but little good of it. Mr. Stadler, who had gone further by introducing the important crushing-energy factor, gave a table, which, by application of the factor to both 'I.M.M.' and 'cube root' apertures, showed the two to be almost identical in practical result; certainly so within the limits of screen accuracy.

Mr. Stadler here virtually controverted Mr. Hoover. But the latter's statements that there is no relationship capable of simple expression in the I.M.M. series, and that 'the only simple law adhered

to is that the series is arranged in the order of meshes per inch, and each succeeding screen has an aperture smaller than the preceding one,' should not be permitted to pass without criticism.

There *was* a simple and definite relationship between the I.M.M. apertures, and, with the exception of one or two meshes interpolated for practical convenience, the series could be plotted in a definite and regular curve.

Perhaps the most striking aspect of Mr. Stadler's paper was the author's extremely important introduction into grading analyses of the energy factor—and its qualification in the crushing of a given rock to a product of definite sieve analyses. The proposal, and the work done in connection with it, was new to him and called for careful consideration. It appeared to be a departure likely to prove of great practical value in the future.

Author's Reply to Discussion.

Mr. THEODORE J. HOOVER: Mr. McDermott and Mr. Sulman have defended their case valiantly and with adroitness. Most of their arguments are, however, effectually and sufficiently answered by the original paper and the accompanying mathematical curves; but the following points deserve attention.

Mr. McDermott's inference that too much scientific exactness can be introduced into laboratory methods is apt to be misunderstood. Certainly the I.M.M. stands for the very highest possible scientific accuracy, and any method or appliance which will in any way tend to increase scientific exactness is admissible. The attempt of the I.M.M. to standardise screens illustratates their true position, and it is not an effectual answer to say that the method is inaccurate at best and let it go at that. Science is never pessimistic; it is always looking for improved methods and standards, and is never content to say 'this is good enough.'

The strongest argument against the I.M.M standard screens is the fact that two men, working on problems of ore crushing and ore sizing, where the very highest attainable scientific accuracy was necessary, have found the I.M.M. screens, within two years after their adoption, practically useless.

The author cordially recognises the pioneer work of the I.M.M. Standardisation Committee. A pioneer in any field does good work when he lays the foundation for greater order; but the fact that he is a pioneer precludes the possibility of finality, and he need take no offence if those following him need something better than his best. A pioneer's log cabin is only a preliminary affair and is always followed by structures more adapted to later needs." Discussion on STADLER and HOOVER.—*Bulletin the Institution of Mining and Metallurgy*, June 16, 1910, p. 5-33. (H. A. W.)

CALCULATIONS OF PERCENTAGE OF RECOVERY.—"In a recent article on the calculation of recoveries in concentration operations, the genesis of certain formulæ was given.

Where a = assay of ore
b ,, tailing
c ,, concentrate
x = tons of ore
y ,, tailing
z ,, concentrate

$$\frac{100c(a-b)}{a(c-b)} = \text{percentage of recovery.}$$

$$\frac{c-b}{a-b} = \frac{x}{z} = \text{ratio of concentration.}$$

The formulæ have several applications in practice other than those previously given, as for instance.

In a mine examination it is sometimes necessary to construct tentative figures as to the concentration of the ore. The assay of the ore will be known and a rough panning test can be made for ratio of concentration. A certain percentage of recovery can be assumed.

Thus:
a = assay of ore = 4% copper
b = ,, of prospective tailing = unknown
c = ,, of prospective concentrate = unknown
x = tons of ore = 100
y = ,, tailing = unknown
z = ,, concentrate = unknown

Assume 80% recovery
Then $a\,x = 4$ tons of metallic copper
$b\,y = 0.8$ tons ,, ,,
$c\,z = 3{\cdot}2$ tons ,, ,,

y and z can now have values from 0 to 100, b can have any values from 0 to 4 and c can have any value from 0 to say 30% (depending on what mineral carries the copper). However, if we assign a value to either b, c, y or z, the value of the others becomes fixed.

Suppose we assume that c, the assay of the concentrate, will be 20% copper, then

$$\frac{20}{100}z = 3\ 2 \text{ tons}$$

$$z = \frac{3{\cdot}2}{0{\cdot}2}$$

$z = 16$ tons of concentrate
$y = x - z$
$y = 100 - 16$
$y = 84$ tons of tailing
$b\,y = 0{\cdot}8$

$$b = \frac{0{\cdot}8}{84}$$

$b = 0{\cdot}95\%$ copper

Our figures then are
Assay of ore = 4% copper
Tons ,, = 100
Assay of tailing = 0·95% copper
Tons ,, = 84
Assay of concentrate = 20% copper
Tons ,, = 16

We can now apply freight and treatment charges, and then proceed with the estimates of mine valuation.

In a case where it was important to learn something of the work in a mill where the management was very secretive, the following data were available:

Surreptitious samples of the tailings assayed 1·4% copper
Surreptitious samples of the concentrate assayed 15% copper

By counting, from the hotel veranda, the sacks of concentrate shipped per week, it was possible to obtain a fairly close estimate of the concentrate produced viz., 60 tons per day.

A question put to the mill-man elicited the information that 300 tons per day was being treated; this was also checked by counting the buckets on the tramway from the mine.

Assay of ore = unknown
,, ,, tailing = 1·4% copper
,, ,, concentrate = 15% copper
Tons of ore = 300
Tons of concentrate = 60

$$\frac{x}{z} = \frac{c-b}{a-b}$$

$$5 = \frac{0{\cdot}15 - 0{\cdot}014}{a - 0{\cdot}014}$$

$5a - 0{\cdot}07 = 0{\cdot}15 - 0{\cdot}014$
$5a = 0{\cdot}206$
$a = 4{\cdot}12\%$ copper
$a\,x = 12{\cdot}36$ tons copper per day in ore
$c\,z = 9$,, ,, ,, ,, concentrate
Percentage of recovery $= 72$

Later when access to the mine was secured the mill and mine records showed an actual recovery of 70·7% for the month during which the week's observations were made from the hotel veranda. The ore for the month, however, had an average assay of 4·5% copper.

The books showed:

Tons of ore for month $7511 = x$
,, concentrate $1602 = z$
Assay of ore for month $4{\cdot}5\% = a$
,, concentrate $14{\cdot}91 = c$
,, tailing $1{\cdot}37 = b$

$$\text{Theoretical recovery} = \frac{100c(a-b)}{a(c-b)}$$
$$= \frac{(100)\,(14{\cdot}91)\,4{\cdot}5 - 1{\cdot}37}{4{\cdot}5\,(14{\cdot}91 - 1{\cdot}37)}$$
$$= 76{\cdot}6\%$$

Actual recovery;
$7511 @ 4{\cdot}5 = 337{\cdot}995$
$1602 @ 14{\cdot}91 = 238{\cdot}8582$
$238{\cdot}8582 \div 337{\cdot}995 = 70{\cdot}7\%$ recovery

The difference between 76·6% and 70·7% or 5·9% represented the losses and inaccuracies, and is not worse than occurs in many cases.

A famous mine not a thousand miles from Melbourne publishes an interesting annual report. The following figures were on one occasion given, suppressing the information as to the tonnage and assay of the tailing:

Tons of ore = 237,265
Assay ,, = 11·5 oz. silver, 16·3% lead, 18·7% zinc
Tons of concentrate = 43,493
Assay ,, ,, = 29·7 oz. silver, 59·8% lead, 10·3% zinc

The tons of tailing can be determined by subtraction and the assay thereof by calculating the contents and then dividing, but it is easier to apply the formula:

$$\frac{x}{z} = \frac{c-b}{a-b}$$
$$b = \frac{33{\cdot}09}{4{\cdot}46} = 7{\cdot}4 \text{ oz. silver}$$
$$b = \frac{29{\cdot}2}{4{\cdot}46} = 6{\cdot}54\% \text{ lead}$$
$$b = \frac{91{\cdot}8}{4{\cdot}46} = 20{\cdot}6\% \text{ zinc}$$

Multiplying according to the old system and dividing by the tons we find that the results are correct within the limits of the slide-rule.

In another mine not far away the following figures were given:

	Tons.	Lead per cent.	Silver oz.	Zinc per cent.
Ore	46,301	12	9·1	10·1
Concentrate ...	5,924	61	29·3	5·5
Jig-tailing ...	17,672	4·3	4·9	8·6
Table-tailing ...	17,085	3·5	5·7	12·0
Slime ...	5,618	10·6	11·0	13·8

For purposes of subsequent treatment of these residual products by flotation processes it is desirable to know what will be the probable assay of the combined jig-tailing, table-tailing and slime. This can, of course, be ascertained by multiplying the tonnage of each by its assay to get the metal contents, adding together the metal contents and dividing it by the total tons. In order to show the usefulness of the formula this old standard method is given below and the same result obtained by the formula. I spent two hours on the first calculation, whereas the second was accomplised in ten minutes.

Ore	46301·1 tons	× 12%	= 5556·132 tons Pb	
	46301·1	9·1 oz.	421340·01 oz. Ag	
	46301·1	10·1%	4676·4111 tons Zn	
Concentrate	5924·1	61%	3613·701 tons Pb	
	5924·1	29·3 oz.	173576·13 oz. Ag	
	5924·1	5·5%	325·8255 tons Zn	
Jig-Tailing	17672·8	4·3%	759·9304 tons Pb	
	17672·8	4·9 oz.	86596·72 oz. Ag	
	17672·8	8·6%	1519·8608 tons Zn	
Table-tailing	17085·6	3·5%	597·996 tons Pb	
	17085·6	5·7 oz.	97387·92 oz. Ag	
	17085·6	12%	2050·272 tons Zn	
Slime	5618·6	10·6%	595·5716 tons Pb	
	5618·6	11 oz.	61804·6 oz. Ag	
	5618·6	13·8%	775·3668 tons Zn	

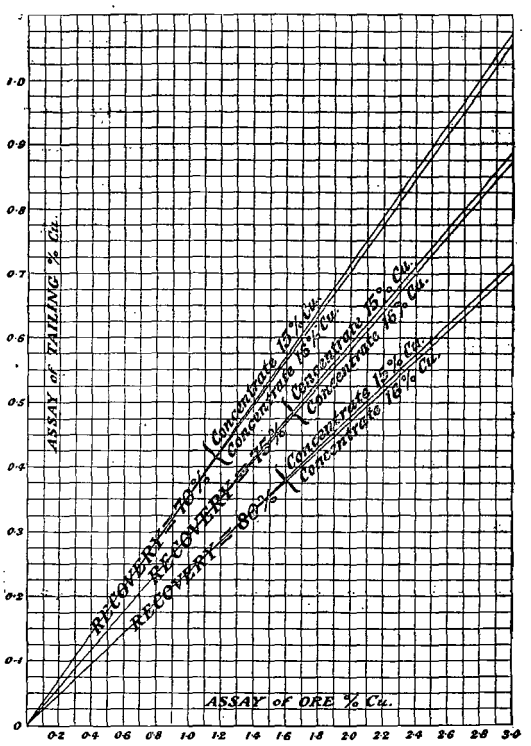
Then add, to get total tons of metal in total residual products:

	Lead.	Silver.	Zinc.
Jig-tailing ...	759·9304	86596·72	1519·8608
Table-tailing ...	597·9960	97387·92	2050·2720
Slime... ..	595·5716	61804·60	775·3668
Total ...	1953·4980	245789·24	4345·4996

Then add the contents of the concentrate as a check on the figures:

Lead.	Silver.	Zinc.
1953·498	245789·24	4345·4996
3613·701	173576·13	325·8255
5567·199	419365·37	4671·3251

These last figures compared with the contents of the original ore show a commendable lock-up of a year's results.



Now the total of residues is :

Jig-tailing	17672·8
Table-tailing	17085 6
Slime	5618·6
Total	40377·0

Now divide the contents by the tons :

$$1953{\cdot}498 \div 40377 = 4{\cdot}8\%\ \text{Pb}$$
$$245789{\cdot}24 \div 40377 = 6{\cdot}1\ \text{oz. Ag}$$
$$4345{\cdot}4996 \div 40377 = 10{\cdot}8\%\ \text{Zn}$$

which we know from subsequent investigation fairly represents the average value of the combined residues.

By applying the formula

$$\frac{x}{z} = \frac{c-b}{a-b}$$

For the lead assay :

$$\frac{46301{\cdot}1}{5924{\cdot}1} = \frac{61-b}{12-b}$$
$$7{\cdot}816 = \frac{61-b}{12-b}$$
$$93{\cdot}792 - 7{\cdot}816b = 61 - b$$
$$6{\cdot}816b = 32{\cdot}792$$
$$b = 4{\cdot}81\%\ \text{Pb}$$

For the silver assay :

$$7{\cdot}816 = \frac{29{\cdot}3-b}{9{\cdot}1-b}$$
$$b = 6{\cdot}13\ \text{oz. Ag per ton}$$

For the zinc assay :

$$7{\cdot}816 = \frac{5{\cdot}5-b}{10{\cdot}1-b}$$
$$b = 10{\cdot}77\%\ \text{Zn}$$

All these calculations were worked twice, once by hand and checked with the slide-rule.

The above figures for tonnage call for some attention. The tons of ore and the tons of concentrate were probably obtained with fair accuracy by weight. But what shall we say of the other figures, which are given to tenths of a ton. It may be taken that the assays are fairly accurate, for if even the arithmetic average of the days' runs be taken the error will not be big enough to vitiate results. It is safe to say that the jig-tailing, table-tailing, and slime were not weighed. They may have been measured by survey and calculated from cubic contents. When the difficulties of moisture and losses are taken into account, the accuracy of the results is noteworthy.

Based on the above formulæ certain tables can be constructed for the use of the staff, so as to give graphically a clear idea of the result of a shift's work. As an example let us assume that samples are taken of each shift's work ; that the material is a simple concentrating ore assaying less that 3% copper ; that recoveries are never lower than 70% nor higher than 80% ; that the concentrate never assays higher than 16% copper nor lower than 15% ; these are all assumptions that could fit a case.

Taking the formula :

$$\text{Percentage of recovery} = \frac{100c(a-b)}{a(c-b)}$$

and writing it as a simple equation without the percentage we have

$$R = \frac{c(a-b)}{a(c-b)}$$

where R is a constant which is a decimal part of unity. Then by transformation :

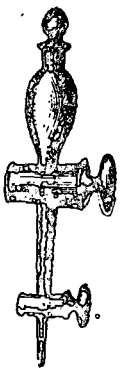
$$b = \frac{ac(1-R)}{c-Ra}$$

By giving a various values and taking values of R and C as above indicated we can plot a series of curves as in the diagram. These are of great use. Having done this let us suppose the shift samples assayed :

Ore	2·6% copper
Tailing	0·75 ,,

By reference to the co-ordinates we find that the point (2·6) (0·75) falls just above the curve for 75% recovery and 15% grade. By making a large wall-chart and plotting curves at intervals of ½% recovery between 70% and 80% the results of a shift's run can be read at a glance with a considerable degree of accuracy. Attention should be called to the fact that any variation in the assay of the concentrate between 15% and 16% has little influence on the curve ; in fact, for over half of its distance the y co-ordinate of the points is within unity whether the assay is taken as 15% or 16%. Near the end of the curve the difference becomes just sufficient to plot the two curves separately with comfort. On a mine where the average assay of the concentrate over a long period is known it would be sufficiently accurate to plot the curves with a fixed constant. This chart would also show at a glance any inconsistency in the assay results, as for instance, if the assayer reported ore 2·9%, tailing 0·55%, concentrate 15·2%, he may be suspected of inaccuracy unless the alternative assumption is made that there has been a marked improvement in recovery to much above 80%. The use of this chart constructed on assumptions of theoretical recovery has a salutary influence on the members of the staff when once they understand its import ; they are on the look out for losses on the one hand and inaccuracies in method on the other."—T. J. HOOVER.—*Mining Magazine*, Aug., 1910 (J. E. T.)

SEPARATING FUNNEL FOR AMALGAM TESTS.—"The conical shaped glass separating-funnel shown has proved to be an excellent elutriating apparatus in making bottle amalgamation tests in which the tailing and mercury were separated and assayed, much loss having occurred with other devices because a part of the slime was carried away by the wash-water. In using this funnel the lower cone is filled with clear water and both cocks closed. After amal-



gamation has been completed in the bottle, the content is emptied into the upper cone, water added to dilute the pulp, and the stopper placed in the top. After being shaken an instant the mercury settles and is drawn into the lower cone by opening the larger glass cock. Some of the sand will pass through with the mercury, but very little slime will accompany it if the lower cone has been filled previously with clear water. The glass stopper is then drawn and the tailing emptied on a filter, then the smaller cock

is opened and the mercury drawn into a dish. If the mercury be floured it can be quickened with a little sulphuric acid and the quickened mercury poured off from the small amount of sand which descended with it. The separation can thus be made with no loss of tailing."—W. H. COGHILL.—*Mining and Scientific Press*, July 9, 1910, p. 53. (K. L. G.)

MINING.

THE VENTILATION PROBLEM IN DEEP MINES.—"As coal and other minerals near the surface become exhausted, mines are constantly tending to become deeper. New problems are in consequence presented to the mining engineer, and one of these relates to ventilation.

Mine ventilation fulfils several main purposes, any single one of which may be of predominant importance in any particular mine. It will be useful first to review these purposes.

1. *Removal of Blackdamp.*—In nearly all mines some constituent of the exposed mineral is liable to oxidation. As a consequence oxygen disappears from the air, and carbon dioxide (CO_2) is liberated. . . . . .

2. *Removal of Fumes from Explosives.*—The fumes from explosives are usually not merely unpleasant, but also poisonous; and where little or no firedamp or blackdamp is produced, as in many metalliferous mines, the removal of fumes may become the main object of ventilation.

3. *Removal of Firedamp.*—In coal seams which are fairly near the surface very little firedamp is usually given off by the coal, as the seams can drain themselves of the gas through the superincumbent strata. With increasing depth this natural drainage becomes less and less, however, with the result that the coal may be heavily charged with firedamp. . . . .

4. *Cooling effects.*—When a mine is very warm, the working power of the miners is diminished, and with increasing heat work finally becomes impossible. Where, therefore, the cost of the miners' labour bears a large proportion to the value of the mineral, as in coal mining, it becomes a matter of vital importance to keep the workings cool. The deeper a mine goes, the higher will be the temperature, other things being equal, and the greater the need for ventilation directed with a view to cooling. *In very deep mining, therefore, the main object of ventilation is not to remove impurities, but to keep the men cool.*

We may thus roughly distinguish a zone nearest the surface where the chief aim of mine ventilation is to remove blackdamp and fumes; an intermediate zone (in coal mines) where removal of firedamp is the chief aim of ventilation; and a deeper zone in which the chief aim of ventilation is to keep the men cool. It is with mine ventilation in this deeper zone that the present paper is concerned.

The problem of keeping the men sufficiently cool to enable them to work, not merely without injury to health, but with the maximum of efficiency, is evidently in part a physiological one; and the physiological side of the problem will be discussed first. Man, in common with other warm-blooded animals, has a normal internal body temperature, which during health is maintained within narrow limits. This temperature, as measured in the rectum, is two or three degrees lower during the night and early morning than during hours of maximum activity in the day. We may roughly state the normal range as between 98° and 101° F.; and the gains and losses of heat by the body must be so balanced as to keep the body temperature within these limits.

How is this balance maintained? Heat is constantly being produced in the tissues of the body by the oxidation processes which are constantly going on. During rest this heat is produced at a rate sufficient to heat the whole body by about 2½° every hour, and during work the production may be increased about three or four times. The corresponding loss of heat occurs partly by conduction to the air and radiation, and partly by conversion into latent heat in the evaporation of moisture. In ordinary cool air only a small part is lost by evaporation; and the loss by conduction and radiation is regulated by variations in the circulation through the skin. In a cold place the skin circulation is diminished so as to prevent the excessive loss of heat which would otherwise occur; whereas in a less cool place, or during muscular work, the skin circulation is increased so as to promote loss of heat. In a warm place, particularly during work, the increased skin circulation cannot get rid of the surplus heat by conduction and radiation; and in this case the loss by evaporation is increased by sweating. When the external temperature equals or exceeds the body temperature, as often happens in tropical or subtropical climates, the whole of the heat is got rid of by evaporation.

Provided the air-temperature is below the body-temperature, heat-loss by conduction to the air is evidently increased greatly if the air is in motion; and similarly evaporation is promoted if the air is in motion, provided that the dew-point is below the body-temperature. Movement of the air is thus of the utmost importance in keeping a man cool, although there is practically always some movement of the air close to the body, in consequence of the convection currents produced by the warming or cooling effects of the skin on the air surrounding it.

Evaporation from the body can only occur if the dew-point is below the body-temperature; but even if the dew-point were below the body-temperature it does not follow that any cooling action would be produced; for if the air were warmer than the body it might communicate more heat than was lost by evaporation. The body during sweating is, in fact, comparable to a wet-bulb thermometer, which indicates, not the dew-point, but a temperature intermediate between the dew-point and the actual temperature of the air, and dependent on both. If the wet-bulb temperature is below the body-temperature heat is lost by the body; but otherwise there can be no loss of heat.

In order that the blood returning from the skin may be sufficiently cooled to prevent actual rise of internal body temperature it is clear that the skin must be kept cooler than the inside of the body. To effect this there must be a constant and free loss of heat by evaporation and conduction, or by evaporation alone, from the skin. Hence the wet-bulb temperature must be well below both the body-temperature and skin-temperature. But movement of the air greatly increases the loss (or gain) of heat by evaporation and conduction; hence with the air in motion the wet-bulb temperature need not be so low as would otherwise be necessary in order to get rid of the necessary amount of heat.

In order to determine the maximum wet bulb temperature at which abnormal rise of body-temperature can be prevented, the writer recently made a number of experiments, partly in Cornish mines, and partly in the laboratory or a Turkish bath. The result was that if the wet-bulb temperature exceeded about 88° F. (31° C.) in still air, it was impossible to prevent rise of body-temperature during rest and

with all clothing removed except light flannel trousers. The result of remaining in a wet-bulb temperature exceeding 88° was that the body-temperature gradually rose, and symptoms of heat stroke began to develope. The more the wet-bulb temperature rose above 88°, the more rapidly did the body-temperature rise. In a good current of air, however, a wet-bulb temperature several degrees higher could be borne. It made no appreciable difference what the actual air temperature was. *Only the wet-bulb temperature and the movement of the air mattered.*

During muscular work, as could have been predicted with certainty, the body-temperature rose rapidly with a wet-bulb temperature at 88° in still air; and even with the wet-bulb at 78° the body-temperature could hardly be prevented from rising during moderate work in still air. In a good air current, on the other hand, continuous moderate work was possible with the wet-bulb at 85°.

The effect of clothing is, of course, to lower considerably the wet-bulb temperature which can be tolerated. A wet-bulb temperature exceeding about 60° or 65° may, for instance, be very dangerous to soldiers marching in heavy uniform. Under such conditions a man may go on until he drops from heat-stroke; and unless he is promptly treated by applying cold water, ice, or whatever means are available for cooling him down, he may die. Among miners heat-stroke hardly ever occurs. The miner is free to divest himself of clothing and to stop work if he is too hot; and it is only in rare cases that he fails to do both. The only case of heat-stroke known to the writer was the tragic death of Welsby, the Yorkshire miner, who, while wearing a rescue apparatus, lost his life in a gallant effort to reach the men who were cut off by the fire at Hamstead Colliery in 1908. The wet-bulb temperature in the return air-way, which he and his companion Whittinghame were traversing, was probably about 80°, and they had ordinary clothing and a heavy apparatus to carry in a road which was low and not meant for travelling. They had pressed forward regardless of the heat, and in returning they both had all the symptoms of heat-stroke. When Welsby could go no further Whittinghame struggled on in the hope of bringing help, but he himself only reached the shaft with great difficulty. The symptoms which he described to the writer were clearly those of heat-stroke, and nothing was wrong with the Garforth rescue apparatus, with which both men were equipped. . . . .

When the body-temperature rises beyond 102° various symptoms of distress begin to appear. Any exertion becomes very difficult, and produces great panting and exhaustion. There is also marked disinclination for any work, either physical or mental, and considerable irritability. The pulse becomes very rapid, even during rest. With further rise of temperature a comatose condition developes, which may end in death. The following description of the symptoms is quoted from a letter to the writer by Dr. Boycott, who had somewhat incautiously moved about as usual while investigating Ankylostomiasis in Levant Mine, which runs out under the sea, and had at the time no ventilation which the writer could measure at the top of what were believed to be upcast or downcast shafts. The only moderately cool places in the deep workings were where compressed air was allowed to escape freely. Dr. Boycott was fortunately accompanied by Mr. (now Professor) Cadman, who was less seriously affected, and could thus render assistance :—

'We climbed straight down one of the shafts (temperature 78° F. at the bottom) to the 278 fathoms level, and walked out under the sea by the pony road, soon discarding our coats, and going in vests and thin trousers only. We then climbed down the submarine shaft (86° wet and dry bulb at the top and 87° at the bottom) to the 302 fathoms level, and walked out westwards, about half a mile I should think, to an 'end,' where the temperature was 93° wet and dry bulb. The going was bad, and I had to stoop most of the way, as the road was largely timbered. When we reached the 'end' we had been underground I dare say three hours, as we had been collecting samples of fæces, etc. I felt very hot, and was glad to sit down. My mouth temperature was 103·5° by a clinical thermometer which Cadman read for me. There was a man and a boy in the 'end,' supposed to be hand-drilling, but they did not seem to be doing anything except sweating. As you know, the men are said to wet the drill-holes by pouring the sweat out of their boots! Coming back, I did pretty well till we had to come up the ladders from the 302 to the 278. The ladders were, I suppose, about three fathoms each, and I had to lie down at the top of each one and gasp. When we got up to the pony road and had a drink of water, I soon felt pretty well all right, and came up the man-engine (my first experience) all right, except that I lost my light. By the time I had washed and changed I felt quite well, and regretted that we had wasted our opportunities of collecting samples. After being in the end I felt no interest in the matter.'

The foregoing discussion makes it possible to formulate roughly the requirements which must be fulfilled by ventilation designed to keep men cool enough for continuous work. It is clear that what is needed is to maintain such an atmosphere *that the body-temperature does not rise beyond normal limits during work, assuming that the men are working stripped to the waist, or entirely naked, as is the custom at present in some of the deepest mines.* This end may apparently be attained by keeping the wet-bulb temperature under about 78° if there is no air current, or under about 85° if there is a good current. More accurate and extended data on this subject are still required, however.

If the limits of wet-bulb temperature are overstepped, the inevitable result in a mine is that the working power of the miners must fall off very seriously. In other words, they have to be paid much more highly for any definite piece of work; and even so they will rightly object to working in places where they are uncomfortable from the heat, and feel that they are mostly wasting their time. The increased cost of production of coal, or any mineral of similar low value per ton, must thus make the mining of it impossible under present conditions, so that the ventilation problem becomes one of life or death for the success of the mine.

We may now consider the conditions which lead to excessive wet-bulb temperatures in mines. The most important of these is doubtless the increase of rock temperature with increasing depth. It is often assumed that, except in volcanic districts, the rate of increase in rock temperature with increasing depth is fairly uniform. There seems to be no doubt that there are in reality wide variations in this respect. To take two or three instances, there is no doubt that at Dolcoath Mine, in Cornwall, the real rate of increase in virgin rock is about 1° F. in 90 ft., whereas at Hamstead and other deep mines near Birmingham, the rate of increase is only about 1° in

110 ft., and in the Transvaal the increase is a great deal less—about 1° in 220 ft.

The actual temperature in a mine depends on the ventilation as well as on the depth and rate of increase of rock temperature. . . . . .

The return air represents, however, oxidation processes occurring not only in the roads and working places, but in large extents of goaf or closed workings. What is of importance is whether the air is cooling or warming the parts where men are working or passing. We can judge of this by comparing the actual air-temperature with the original temperature of the virgin coal or rock, if this is known, as it ought to be; but a far clearer insight can be obtained by analysis of the air at different points in the ventilation current. To be of any use these analyses must be accurate. So-called 'technical' methods, which allow of errors amounting to more than 0·1% in the oxygen, are of little use.

From the existing knowledge of mine air, to which an exceedingly valuable addition has been made in the recent report made to the Royal Commission on Mines by Professor Cadman and Mr. Whalley, it seems clear that as a general rule under present conditions the ventilation of coal mines has a very considerable heating effect, except on the intake roads. On the working places themselves, the effect may be either cooling or warming, according to the amount of ventilation, as compared with the rate of oxidation. In some collieries the rate of oxidation on the intake roads themselves is so great, or the ventilation so small, that before the air reaches the working face it is heated much above the natural rock temperature.

When air is passing month after month, or year after year, down a shaft or along a roadway, the surrounding rock is either cooled or warmed to near the average temperature of the air. The rock round an upcast shaft is, for instance, gradually warmed, while that round the downcast is cooled. Any effect on the temperature of an alteration in the ventilation is thus very gradual, and may take months or years to develop fully. The gradual effects of ventilation are, however, often very striking. Thus in Dolcoath tin mine, in Cornwall, where the natural ventilation is much more than sufficient to carry off all the heat produced by oxidation, the temperature of the open stopes at the deep levels is much below the natural rock-temperature, so that in spite of the great depth and a rock-temperature of nearly 100° at the bottom, work can be carried on fairly freely. A large new vertical shaft, which is now approaching completion, will supply abundant cool air direct to the deepest workings, and obviate any further difficulty from the heat for a long time to come. In Levant tin and copper mine, on the other hand, where the ventilation does not carry off more than an insignificant fraction of the heat produced by oxidation, the temperature of the deeper workings, which are not nearly so deep as at Dolcoath, was found by the writer to be much above the calculated rock-temperature, and so high that very little work could be done by the men.

A rapid increase of rock-temperature with increasing depth may be in some respects not so much of a disadvantage as at first sight appears; for the greater the rate of increase of rock-temperature the more assistance will be given to natural ventilation. *A fan would, for instance, be wholly superfluous in the case of a deep Cornish mine. In the Transvaal, on the other hand, natural ventilation is so feeble that much difficulty is experienced in the removal of fumes from explosives, etc., and it seems as if fan ventilation may require to be introduced as the mines get deeper.* As the air descends or ascends a shaft it is heated or cooled by the compression or expansion which occurs, the rate of heating during the descent being about 1° F. in 180 ft. But in the Rand mines the rate of increase in rock-temperature is only about 1° in 220 ft. It appears, therefore, as if there might often be no natural ventilation at all in these mines. Cooling by evaporation occurs, however, in the downcast shaft, while cooling of the upcast is hindered by the condensation of moisture and consequent liberation of latent heat, which tends to occur in all upcast shafts, and so helps to keep them warm. Once the ventilation has started, as it must do in cold weather, it will thus tend to continue, since the upcast shaft will become permanently warmer than the downcast.

The previous physiological discussion showed that the problem of ventilation for coolness may probably be summed up in that of keeping the air moving and the wet-bulb temperature below about 80° F. We can now follow the process by which this critical point is reached in a deep mine. As the air descends the shaft or incline, it is in the first place heated by compression. This effect is, of course, not directly measurable from day to day, as the shaft walls may themselves either warm or (in hot weather) cool the air. Heating by compression warms the air about 5½° F. for every 1,000 ft. of descent. Hence, in a mine 4,000 ft. deep compression accounts for a rise in air-temperature of about 22°. If we assume an increase of 1° in 70 ft. the rock-temperature at the bottom of the mine will be about 50° + 57° = 107°. The mean air-temperature at the shaft bottom will be about 50° + 22° = 72°, apart from the influence of the shaft wall in warming or cooling the air by conduction or evaporation. The more rapid the air-current, and the wider and drier the shaft, the more closely will the mean temperature at the shaft bottom approach the latter value. As the air passes to the working face it will pick up heat from the rock surrounding the intake roads; but the more rapid the current the less will its increase in temperature be. It will also pick up moisture: and if coal dust or small coal is lying on the roads or exposed on their sides this will continuously oxidise and thus help to warm the air. In some mines this oxidation may assume very formidable dimensions. At Hamstead Colliery, for instance, the oxygen percentage of the air in an intake 3,000 yards from the downcast shaft was found to be reduced by 0·88%, which corresponds to oxidation capable of continuously heating the whole air-current of the road by about 230° F. At that time the main intake roads were in the thick coal. The actual temperature on the road was only about 15° above the natural rock-temperature, so that most of the heat was escaping into the surrounding strata.

If the intake air takes up much moisture in the downcast shaft or main intake roads it will, of course, be correspondingly cooled or prevented from heating. Every grain of moisture added by evaporation to a cubic foot of air corresponds to a cooling effect to the extent of about 8° F. Average fresh air in this country contains about 3·2 grains of moisture per cub. ft., while air saturated at 80° contains 11 grains. In saturating an air-current to 80° the heat required would thus be the same as in warming it (11—3·2 × 8 = 62 degs. Hence, air at a mean temperature of 50° takes up about twice as much heat in being saturated at 80° as in being warmed to 80° in a mine. There is no immediate physiological disadvantage in cooling the intake air by evaporation of moisture, as in spite of the increased saturation of the air the wet-

bulb temperture is slightly lowered. A little consideration will show, however, the secondary effects of such a course must be disadvantageous in a deep or warm mine; for the cooler the intake air is kept the more raridly will it tend to take up heat from the surrounding strata, and if it is already highly charged with moisture, *the range within which its temperature can be allowed to rise without interference with the working capacity of the men is much restricted.* For instance, if air has already been saturated with moisture at 70° its temperature cannot subsequently rise beyond about 96°, even if no further moisture is added, without affecting working capacity; whereas, if no moisture had been added far higher temperatures could be borne without inconvenience. Watering the main roads is certainly desirable as a safeguard against coal dust explosions. But if the roads will not stand it, or if it causes such a serious rise in wet-bulb temperature at the working face that working costs become excessive, other means must be taken for guarding against explosions.

It is clear that, so far as difficulties from heat are concerned, air may *be kept in a fit state for working in by either keeping it from taking up moisture or keeping it cool, or by both methods combined.* To keep the air cool in a deep mine, the air-current to the working face must be sufficiently increased to cool the rock round the downcast shaft and intake roads to the desired extent; the oxidation processes along the roads must at the same time be so reduced that their effects in warming the air remain within manageable limits. To keep the air dry all unnecessary moisture must be avoided, and the air-current must be so large that the percentage of moisture which it picks up along the roads is as small as possible. With the ventilation properly laid out to secure these objects there should not be much difficulty in working coal at depths up to at least 5,000 ft. Success will certainly depend, however, on realising the nature of the difficulties and taking the proper means to meet them.'—DR. J. S. HALDANE. *University of Birmingham Engineering and Mining Journal*, July, 1909, p. 49-57. (G. H.-S.)

DEPTH IN MINING.—"In *Bulletin* No. 424, issued by the United States Geological Survey, the question of maximum depths of mining is discussed by Mr. George H. Ashley. Reviewing the advance of deep mining in Europe, he remarks that there appears to be no engineering or mechanical difficulties which cannot be successfully overcome within the limit of 4,000 ft. In the United States to-day coal lying more than 3,000 ft. below the surface is being disregarded in connection with the disposition of coal lands on the public domain, whereas coals below 3,000 ft. are being successfully and profitably mined in Belgium, England, Wales, France and Australia—a group of countries which supply about half the world's production. Of course it should be recognised that as coal-mining in different parts of the United States is extended to great depths, there may be localities in which the difficulties arising from deep mining may combine in such a way as to make further operation impracticable. It is believed, however, that such localities will be highly exceptional and that in general the obstacles encountered in the United States will be no greater than those which have been successfully overcome in other countries. At present the deepest mining which occurs in the anthracite region reaches about 2,200 ft. The absence of deep mining is due to the fact that, in the Appalachian region, where coal-mining began and the greatest development has taken place, the coal beds, with few exceptions, are near the surface, and the structure of the rocks is such that the coal beds do not extend to great depth. Likewise in the northern and eastern regions of the Interior province, where mining has gone on for a long time, the structural basins are relatively shallow, and the coal beds are near the surface. In the Rocky Mountain province, where the coal generally lies in deep structural basins, mining is not far advanced, and for the most part is confined to the margins of the basins. The deepest workings in the bituminous fields of western Pennsylvania probably do not exceed 1,000 ft.; in Ohio and in West Virginia, which includes the New River and Pocahontas fields, the present depth or cover of coal-mining is still less. The same maximum is found in Eastern Kentucky, Tennessee, and Alabama, except possibly at a few places. In fact, throughout the Appalachian region, with the exception of the south end, or that part in north-eastern Alabama where there are some deep basins, the maximum cover of the unworked coal beds, probably does not exceed 2,000 ft. In the Rocky Mountains province the deepest mine about which information has been procured in mine No. 1 of the Union Pacific Coal Company, at Rock Springs, Wyo., where a depth of 2,000 ft. has been reached. Of the eight regions included within the Rocky Mountain province five are large structural basins in which the coal beds outcrop more or less persistently around the margins, and if these beds are continuous underground they must reach great depths in the centres of the basins. The coals are late cretaceous to tertiary in age, and are contained for the most part in rocks which are prevailingly soft. It is obvious that as mining is extended to greater depths in the soft coal bearing rocks of the west, more or less difficulty may be expected."—*The Science and Art of Mining*, Sept. 3, 1910, p. 31. (A. R.)

STATE CONTROL OF MINING.—"Mine managers are hedged about by too many laws and regulations. They are frequently trying to master their duties, as laid down in voluminous acts of Parliament, when they might be better employed studying the natural peculiarities of their own particular mines. The gentlemen who frame these laws seem to imagine that all mines can be worked upon one uniform principle. Sufficient allowance is not made for the varying natural conditions. Too many laws of an inflexible nature imposed upon all mines and all managers regardless of local and individual circumstances, must have the effect of checking initiative and inspiring the idea that it is more the business of the State than the manager to safeguard the workers. The State, having gone so far in the matter of law-making, surely ought to follow this up by maintaining an inspector at every mine to see that the law is observed alike by worker and employer. If the State is so eager to protect labour that it leaves mine owners and managers with but a mere shadow of freedom then let the State carry its policy to a logical issue and relieve owners and managers of the duty of looking after the workers' as well as their own interests. Let there be an ample staff of inspectors to attend to the safety of the workers and leave the managers free to attend the business of the employers, subject to the requirements of the inspectors. Either let us have less State interference or full State control in matters of safety. The policy of the State being what it now is, it is not sufficient that the inspectors employed by the State should only, as a general rule, visit the mines in case

of accident. Let the inspectors be on the spot constantly to prevent accidents. If we had fewer laws and more inspectors it would be better."—T. GOOD, *Cassier's Magazine.—The Science and Art of Mining*, Sept. 3, 1910, p. 25.

COAL WINDING RECORD.—"The Bolsover Colliery Company's Mansfield colliery now claims to its credit the world's record in bringing coal to the surface. The amount of coal raised to the surface for a week, which consisted of 5½ working days, ending July 19th, was 24,582 tons, which averages about 4,469 tons per day of 7 hours 40 minutes winding. This works out at 9·7 tons of coal raised per minute throughout the week's winding."—*Science and Art of Mining*, Aug. 6, 1910, p. 601. (A. R.)

PROSPECTING FOR COAL.—"1. (*a*) To prospect for coal in an unexplored country, the exposed strata must be carefully examined for contained fossils in order to determine to which formation the strata belong. To do this requires some knowledge of geology, and having decided that the strata belong to the carboniferous, it remains only to find the coal either by exposing an outcrop by digging, or putting down bore-holes.

(*b*) In a well mapped and well developed country, there will usually be no doubt as to the kind of strata in any particular district, and ordnance maps will be of great assistance as indicators of outcrops and also of rate and direction of dip. The search is then largely an intelligent use of the map, intelligent observation in the field, and perseverance with the pick and shovel. Bore-holes may be required in this case also.

2. For boring to this depth a derrick or shearlegs will be necessary, together with a windlass, so as to easily raise and lower the rods when required. The rods will be of a uniform length of 6 ft. and joined together by the usual male and female joint. From 4 to 6 cutting bits will be required so as to allow time for sharpening, etc.

A bracehead will be required, and for the first 30 yds. the percussive motion of the tool will be provided by from two to four men at the bracehead. Beyond this point it will be necessary to relieve the men by the application of either a rocking lever or a spring pole. With the latter appliance boring will be easily accomplished to the required depth. Other appliances necessary during the work will be sludgers for cleaning the hole, keys for the support of the rods while unscrewing or screwing up, and spanners for the same operation. As important appliances as any will be those for dealing with accidents, should any arise. They are usually the crow's foot, the screw bell and the spiral worm.

3. The principal methods employed for deep boring are:—

(*a*) Hand-boring by rigid rods.
(*b*) Mather and Platt system of boring by flat rope.
(*c*) Diamond system of boring.
(*d*) The American system of boring by round rope.

The points to be considered in deciding which method to adopt are:—

(1) The object for which the boring is made.
(2) Probable nature of the strata to be passed through.
(3) Speed of boring by the different systems.
(4) Cost of boring by the different systems.
(5) Whether a supply of water is available, this being necessary in some of the methods employed.

4. The advantages of sinking a shallow shaft at the mouth of a bore are:—

(*a*) It saves lining the hole through the same distance because this will usually be loose ground.
(*b*) It increases the effective height of the shearlegs.
(*c*) In exposed situations it provides good shelter for the men.
(*d*) It is often cheaper to sink a shallow shaft than to bore that same depth at 50 or more yards.

By the latter statement is meant that two or three yards of shaft, say five or six feet diameter through loose soil, can be sunk at a less cost and with infinitely less trouble than boring the last two or three yards of a boring which is 50 or more yards. We will take an example:—Suppose a bore-hole to go to a depth of 100 yds. is contracted for at the rate of 3s. 6d. per yard for the first five yards and increasing by 3s. 6d. per yard for every additional five yards. The first yard costs 3s. 6d. and the hundredth yard cost $\frac{100}{5}=20$ times as much, *i.e.*, £3 10s. The last three yards of such a bore-hole will be at the rate of £3 10s. per yard. If, therefore, three yards of shaft be sunk at the commencement the hole will only need to be 97 yds. deep and a sum equal to 3 × £3 10s. = £10 10s. will be saved.

5. The most frequent are the following:—(1) Breakage of the boring apparatus; (2) Tools, etc., falling down the hole. There is also in deep boring an excessive vibration set up in a long length of rods, loosening the sides of the hole, causing the rods to buckle and often to fracture. To overcome the latter difficulty various forms of sliding joints and free fall cutting tools have been devised, among the best known being Oeynhausen's sliding joint, and the free fall apparatus of Kind and of Dru.

The remedy for accidents (1) and (2) will be in the direction of the recovery of the broken pieces; and failing that, if the obstruction is large, to pound it to pieces small enough to be brought up by the sludger, or to pound them into the strata at the side of the hole and out of the way of the boring. Another process is to bore alongside a broken tool which has become firmly embedded and afterwards put into this hole a heavy charge of dynamite in the hope of loosening the broken tool. Before adopting these extreme measures, the tools usually employed are those for the recovery of broken pieces, such as the crow's foot, the screw bell, and the spiral worm.

6. (*a*) Since it is a fact that bore-holes are very rarely vertical, it is sometimes desirable to have the course of bore surveyed; and if such is done a crooked bore gives just as valuable information as a straight one.

(*b*) Beds of conglomerate composed of very hard pebbles in a comparatively soft matrix will readily cause deflection; a mixture of flint and chalk, the flint being on one side of a bore and the chalk on the other will do likewise; and highly inclined beds of varying degrees of hardness also give considerable trouble in this direction.

(*c*) To straighten a crooked bore-hole, the crooked portion may be filled with cement or concrete, and after it has set well, re-boring may be carefully carried out.

(*d*) To survey a bore-hole, probably the very best apparatus is the Clinostat of MacGeorge. Several of these should be encased in a suitable brass tube, and, after liquifying the gelatine, lowered to the desired point by means of a strong hempen cord or copper wire. Generally it will be advisable to take observations at regular intervals, say every 50 ft. throughout the length of the bore, and if the deflection at each of these points is noted, a line drawn through these points will represent the course or path of the bore-hole."—*The Science and Art of Mining*, Sept. 3, 1910, p. 35. (A. R.)

EXPLANATION OF EFFICIENCY OF SAND AS TAMPING MATERIAL.—By way of illustrating the advantage of sand over clay for tamping, the author fills a glass tube like the globe of a lamp with clay and shows how easily the contents can be pushed out by means of a wooden rod of a diameter a little less than the bore of the glass tube. He next fills the same tube with sand tightly packed in, and shows that it is practically impossible to force it out again.

He holds that clay tamping approaches the state of things which would be produced if a wooden or iron rod were used as tamping.

The pressure which would be exerted at the first instant of the explosion, on the inner end of the rod would immediately be communicated through its whole length and it would begin to move bodily from the hole. The rod in fact would act as a projectile and would probably be quite clear of the hole before the full energy of the charge had been exerted.

On account of the non-cohesive nature of sand tamping, and the fact that even when pressed together the grains are separated by air spaces, the pressure exerted on the inner cross section of the tamping by the gases from the explosion is not felt through the whole mass of the tamping and some of the energy is expended in forcing the sand against the sides of the borehole. The gases from the explosion therefore get time to exert something like their full force against the surrounding rock. The author recommends sand cartridges, and instances cases in which after the explosion of heavy charges these cartridges were found still in the boreholes. As further support for sand tamping, he refers to the fact that snow in the muzzle of a gun barrel can cause bursting of the barrel with a normal explosive charge in use.—W. SCHULL.—*Zeitschrift für Gesamte Schiess und Sprengstoffwesen*, No. 13, July 1, 1910. (T. D.)

DEEP MINING ON THE RAND.—"At the Transvaal University College Mr. Lionel Phillips, one of the Rand mining magnates, in lecturing on the "Rand Gold Deposits" pointed out that mining operations on the Rand could, with profit, be carried on to a depth of at least 7,000 ft. from the surface, and that, therefore, the Witwatersrand goldfield would not be exhausted during the present century.

It is generally considered here that, owing to the extremely favourable conditions for deep mining prevailing on the Rand, that mining operations will be possible at a much greater depth than 7,000 ft. from the surface, providing an adequate method of hoisting is available, and the coming extensive use of electrical power for all mining operations on these fields seems the best likely to afford a solution to the deep winding difficulty. It must not be forgotten that on the Rand, as the mining operations deepen, the rate of increase in underground temperature steadily decreases. At first it was estimated at 1° F. for every 82 ft., but when actually tested down to a depth of 1,000 ft., it was found to be 1° F. for every 100 ft. Recent observations down to a depth of 3,500 ft., however, showed 1° for every 208 ft., and more recent tests have shown that over 4,000 ft. the rate of increase was only 1° F. for every 255 ft. There seems every likelihood of a constant decrease in the temperature gradient as greater depths are attained on these fields. Mines are working here at a depth of 4,500 ft. without the slightest attempt at fan ventilation and, as a matter of fact, are dependent on natural ventilation and the compressed air used for power purposes. Eventually, it seems possible that the temperature gradient will fall as low as 1° F. for every 400 ft. or 500 ft. in depth, so that as regards temperature that does not at present seem likely to form an insurmountable obstacle to deep mining on these fields. Then it must not be forgotten that the native labourer here is capable of working efficiently at much higher temperatures than the white labourer and this, again, is a valuable local asset to the prospects of ultra deep mining on these fields. The atmosphere here is also much drier than in other parts of the world, whilst the hanging and foot walls leave nothing to be desired. The banket beds are equally hard and capable of standing a pressure of 7,000 lb. per square inch and, even if crushed, are not like coal seams, but will still retain their value. No matter what is said to the contrary the Witwatersrand is destined to become the scene of the deepest mining operations in the world, and is destined for many years if not for centuries, unless some new field is discovered, to continue to be the premier gold producer of the world."—*Canadian Mining Journal*, Sept. 15, 1910, p. 571. (A. R.)

MISCELLANEOUS.

34TH ANNUAL REPORT OF H.M. INSPECTORS OF EXPLOSIVES, 1909.—"*Factories and Magazines.*—There were under continuing certificate 33 factories and 90 magazines, and under licence 107 factories and 350 magazines. 261 visits were paid to factories and 494 to magazines.

Accidents.—The number of accidents by fire and explosion of which the department had cognisance was 453, causing 60 deaths and injuring 445 persons. The number of accidents in factories for explosives was 51, causing 6 deaths, and injuring 12 persons. The number of deaths was below the average for the decade (7·3).

Importation. — The amount of foreign blasting explosives containing nitroglycerine, imported in 1909, was 1,311,488 lb., against 1,086,205 lb. in 1908. Of the former amount 829,480 lb. were transhipped to other countries. The non-glycerine explosives imported amounted to 11,635 lb. against 6,100 lb. in 1908. The number of detonators imported was 18,146,815. The importations of fireworks amounted to 783,502 lb.

Chemical Advisers' Report.—Messrs. Dupré report that of 496 samples examined by them, 62 were rejected for the following reasons: Low heat test (21), exudation (20), and incorrect composition (21). 41 new explosives were examined, of which 30 were passed, 8 passed preliminary tests, and 3 were rejected.

Testing Station at Woolwich.—Captain Desborough reports that 66 explosives are on the 'permitted list.' 15 explosives were tested during the year, of which 5 successfully passed the test.

Amount of Explosives Used in Mines and Quarries in Great Britain.—The following table gives the details:—

Explosives.	Lb.	Percentage of Total.
Permitted explosives ...	8,502,232	28·3
Gunpowder	17,595,475	58·3
Gelignite	3,085,529	10·2
Blasting gelatine and gelatine-dynamite...	616,436	2.0
Cheddite	123,531	0·4
Dynamite	117,260	0·4
Various	51,424	0·2
Total	30,091,887	100·0

The following list gives, in tons of 2,000 lb., the quantity of the various permitted explosives, with a consumption of more than 50 tons annually. Bobbinite (559 tons), arkite (375), monobel (369), ammonite (277), samsonite (253), saxonite (252), rippite (242), roburite (239), westfalite (222), bellite (221), carbonite (215), stow-ite (130), Faversham powder (113), ammonal* (107), albionite (101), excellite (88), rexite (83), abbeite (60), and permonite (60 tons)."—*Journal of the Society of Chemical Industry*, Sept. 15, 1910, p. 1082. (J. A. W.)

Reviews and New Books.

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

PRINCIPLES OF CHEMICAL GEOLOGY. By JAMES VINCENT ELSDEN, D.Sc., F.G.S., is made up into 222 pages, 10 mo., with index. There are 10 chapters and 44 diagrammatic illustrations. Published by The MacMillan Co., New York. City, and Whittaker & Co., London, England. Price $1.60 net.

"The main object of the book is to present in a concise form the application of the principle of equilibrium to certain geological phenomena. To the student of geology nothing is at first more bewildering than the multiplicity of changes which rocks and minerals undergo in nature. To elucidate and stimulate interest in this branch of geology, Mr. Elsden has divided the subject into ten chapters, of which the following are synopses: I. Equilibrium between the crystalline and amorphous states, based on Le Chatelier's principle that 'any external change in the factors of equilibrium of a system is followed by a reverse change within the system.' II. Equilibrium influenced by viscosity plays an important part in retarding the transition of minerals. III. Diffusion as a factor of equilibrium in geological transformations may be due to gases, liquids, or solids. IV. Surface tension as a factor of equilibrium is considered as one of the most important features in supersaturation. V. Vapour pressure as a factor in equilibrium is based on experiments that show that many solids possess an appreciable vapor pressure at moderate temperatures. VI. Equilibrium conditions of polymorphous forms, that is where transformation of a mineral from one crystalline form to another with differences in specific gravity, melting point, and other physical properties. This subject possesses geological interest owing to the effects produced by dynamical metamorphism. VII. Equilibrium in solution. From the point of view of the phase rule, unsaturated solutions are trivarient systems that possess three degrees of freedom, and can only be defined by the three variable factors, pressure, temperature, and concentration. VIII. The eutectic theory is elucidated from ice to its projection on a plane triangle, making it necessary to examine how far the eutectic theory may apply. IX. The theory of solid solutions applied to geological problems, covers 23 pages, and forms one of the most interesting chapters. X. is on the conditions of chemical equilibrium in geology. The book is interesting throughout and will appeal to the advanced mineralogist, physicist, chemist, and geologist." *Mines and Minerals*, Sept., 1910. (A. R.)

* Including non-permitted Ammonal.

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(C.) 509/09. Francis Joseph Watts (1), Fraser Joseph Alexander (2). Improvements in electrically actuated apparatus for cleaning and scouring amalgamating plates or surfaces. 10.11.09.

This patent application covers an electric motor with gearing attached for rotating a suitable brush for cleaning amalgam plates.

(C.) 525/09. Alfred George Newkey Burden. Improvements in guides for the stems of the stamps of stamp mills. 17.11.10.

This application refers to metal guides for stamp mill stems constructed as a base piece permanently bolted to the guide girt and the guide sleeve proper constructed to seat into a dovetailed groove in the base piece and secured thereto by a single bolt.

(C.) 122/10. William Arthur Caldecott. Improvements in means for transporting and de-watering pulps. 23.3.10.

This application relates to dewatering sand pulp in connection with "sand-filling." Sand from treatment vats is sluiced or trucked to a sump and thence pumped to dewatering cones placed above a borehole delivering to the mine. The overflow is gravitated or pumped back for re-use with a further quantity of sand. Claims 1 and 2 relate to the combination of means of transport, with dewatering classifiers having in view the saving of pumping back surplus liquid and the confining of the carrying liquid to a limited circuit.

(C.) 384/10. Hans Peter Hansen. Improvements in and relating to the manufacture of pipes, tubes and the like from wood and in apparatus therefor. 5.8.10.

This invention relates to a process for the manufacture of wooden tubes and pipes by means of wooden shavings spun round a collapsible mandril which is inflated with air or other fluid during the spinning operation. This mandril can be collapsed to permit of the removal or shifting of the spun pipe, etc.

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

This application relates to an improved process of treating complex ores for the recovery of the contained metals. The essential features of the process consist in roasting the ore to remove volatile constituents, converting the copper to sulphate, and iron to oxidised condition, then removing iron as dry powder and boiling the residue in water and sulphuric acid by means of the "hot blast," then removing the liquor and washing thoroughly when the precious metals are recovered by solution in cyanide solution or other known means.

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

This application relates to an apparatus for concentration of finely divided ore by means of a shaking table. Different portions of the surface of this table have different inclinations, and their arrangement is specially adapted to the concentration of slime.

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

This application relates to improvements in the cyanide process which consist in methods of regenerating cyanide solutions or converting cyanogen bearing compounds into solvents for gold by the addition of amide compound in the presence of an electric current.

For example, carbamide added to potassium cyanate and electrolysed, will form a solvent for gold in alkaline solution.

Again, the electrolysis is performed while the mixture is in contact with the ore to be treated, of which several examples are given with modifications, such as using halogen compounds to promote oxidation, etc.

(C.) 455/10. Henry Hooke. Improvements in centrifugal separator or filter. 14.9.10.

This application relates to a centrifugal separator or filter in which the "basket" is provided with a

filtering medium, and is revolved by a hollow shaft, through which is a solid shaft carrying a conveyor revolving at a higher or lower speed, thereby causing continuous discharge of the separated solids. The separator differs from the ordinary centrifugal machine in being placed horizontal instead of vertical, and in having means for continuous discharge.

Selected Transvaal Patent Applications.

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