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Proceedings
AT
Ordinary General Meeting, March 20, 1909.

The Ordinary General Meeting of the Society was held in the Lecture Theatre of the Transvaal University College, on Saturday evening, March 20th, Mr. R G. Bevington (President), in the chair. There were also present:—

50 Members: Dr. J. Moir, Messrs. A. McA. Johnston, W. R. Dowling, A. Richardson, C. B. Saner, G. O. Smart, Prof. G. H. Stanley, J. E. Thomas, A. Whitby, H. A. White, Prof. J. A. Wilkinson, W. A. Caldecott, W. Cullen, F. F. Alexander, E. Blume, H. O. Bowen, W. Brown, P. Carter, F. T. Chapman, A. A. Coaton, M. H. Coombe, W. M. Coulter, L. Evans, A. D. Gilmour, Dr. J. McC. Henderson, A. Heymann, C. G. James, A. J. Johnson, J. Kennedy, R. N. Kotzé, G. A. Lawson, Hy. Lea, J. Lea, G. Melvill, S. Newton, T. T. Nichol, J. F. Pyles, L. J. Robinson, N. Rogers, O. D. Ross, H. A. Scarf, W. Sharp, F. L. Simmons, R. Stokes, H. Taylor, J. A. Taylor, O. Tonnesen, A. D. Viney, J. P. Ward, and F. W. Watson.

10 Associates: Messrs. J. Bowyer, J. Chilton, J. Cronin, M. Dennehy, H. Stadler, G. G. Thomas, W. E. Thorpe, W. Waters, J. Whitehouse and J. A. Woodburn.

17 Visitors, and Fred. Rowland, Secretary.

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

NEW MEMBERS.

Messrs. H. M. Coombe and P. L. Edwards were appointed scrutineers, and after their scrutiny of the ballot papers, the President announced that all the candidates for membership had been duly elected, as follows:—

ADAMS, HIRZEL ALLEN, Village Main Reef G. M. Co., Ltd., P. O. Box 1091, Johannesburg. Assistant Surveyor.

HÖVIG, PETER, Benkoelen, Sumatra, Dutch East Indies. Government Mining Engineer.

MEHLISS, MAX, M.D., P. O. Box 1076, Johannesburg. Medical Superintendent, Rietfontein Lazaretto.

TONNESEN, KARL OSCAR, C.E., P. O. Box 6647, Johannesburg. Construction Engineer, Knights Central, Ltd.

WERNHER, Sir JULIUS C., Bart., 1, London Wall Buildings, London, E.C.

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

BIRD, FRANK ARTHUR, Ancona Cottage, Ivy Mine, Barberton. Cyanide Manager.

KAY, HERBERT GEORGE, P. O. Box 182, Premier Mine, Cullinan. Surveyor.

NAKAMOTO, HIDEHIKO, Sado Gold Mine, Aikawa, Sado, Japan. Sub-Manager.

The President: It is very satisfactory to see that our Society has now got so far as Japan. Its ramifications are evidently extending all over the world.

GENERAL BUSINESS.

The President: There is one announcement under the head of general business, and that is with regard to our annual dinner. The actual date was not settled in time to place it on the agenda paper for this meeting, but it has since been settled as Saturday, the 24th of April, that being practically the only convenient date. The previous Saturday is our general meeting, and we do not like to depart from our usual practice and give up our meeting for anything. The Saturday preceding that is the Saturday before Easter, which is hardly a suitable date. Anyone requiring tickets can obtain them from our Secretary to-night. The price is one guinea, and I sincerely hope that as many members as possible will attend and make our dinner the success it always has been.

GOLD LAW REGULATIONS.

Mr. F. F. Alexander (*Member*): I would like to ask, under general business, whether the Council has done anything to advise the Government with regard to the covering of our extractor boxes, and whether the Council has taken any steps to advise the industry as to whether the whole of the boxes shall be covered or not?

The President: I may say that the Council considered the question in the first instance, and they decided that it was rather a matter which should emanate from the Government, and from the Consulting Engineers of the various groups. The Council felt that it could not take any action as it was a matter of law and the carrying out of regulations. We considered that it was entirely a matter between the Government and the various Consulting Engineers.

Mr. F. F. Alexander: I have brought this subject up because some of the Government officials express ignorance on it. I think this Society should take active steps in this case, and I believe it would facilitate things. Personally, I do not think that the whole of the boxes need be covered up.

Mr. R. N. Kotze (*Government Mining Engineer*): I think it would fill a useful purpose since so many of our members are interested in this question, if a note were published in the Society's *Journal* to the effect that the Government will not insist upon weak extractor boxes being covered. The question as to which are weak and which are strong extractor boxes will be left to the Inspectors of Mines.

Mr. F. F. Alexander: I had occasion to go into the question to-day. It is, in my opinion, an irksome law which enforces the covering of the whole of the extractor boxes, or even the whole of the strong boxes. I believe it is not too late now for this Society to take action and approach the Government, so as to guide them as to what is really right in the matter.

Mr. H. A. White (*Member of Council*): I might say that, at my instance, the Council of the Society appointed a sub-committee to go into this question and review these regulations, a copy of which was submitted confidentially to them by the Government Mining Engineer for the purpose. We found certain small details which might have been improved, but we were informed that these regulations were the result of a compromise between the Government and a committee appointed by the Mine Managers' Association. We were satisfied that the gentlemen responsible for the technical control of our works had agreed to accept these regulations, and under these circumstances we decided that the alterations we might have put forward were perhaps better left alone. We, therefore, wrote to the Government Mining Engineer to the effect that we had considered the matter, and that having been informed that these regulations had been agreed to by the Mine Managers' Association and others who are responsible, we decided to make no suggestion.

The President: Yes, I may say that is exactly what took place.

Mr. F. F. Alexander: I still assert that we have time to represent our views. Why put these points as being trivial? I think they are not trivial, and that they affect us, as workers in the industry, very much. I think the Council should formulate some scheme, and represent their scheme through the proper channel to the Government.

Mr. H. A. White: It was suggested that we should ask members to bring forward descriptions of appliances, but it was agreed by the Council that these matters were better left in the hands of the Consulting Engineers of the groups.

Mr. F. F. Alexander: Then I understand the Council have really shelved a matter of great importance?

The President: I do not think you can say that, Mr. Alexander. The matter is one of law.

Mr. F. F. Alexander: I do not think it is a matter of law until the Government enforces it, and I cannot believe the Government wishes to hamper us with any law that is irksome.

The President: It became law on the 2nd of January. Of course, Mr. Alexander, it is quite competent for you, as a member, to move a resolution instructing your Council to take action.

Mr. F. F. Alexander: I move a resolution that the Council be instructed to take immediate steps to investigate the matter, and to combine with the Mine Managers' Association and through them the Chamber of Mines, in drafting arrangements whereby these extractor boxes may be wholly or partly covered, subject to the approval of the Government Inspector.

Mr. J. E. Thomas (*Member of Council*): I beg to point out that what will suit one mine will not suit another. I think it should easily be arranged with the Inspectors of Mines in each particular district in which a mine is situated.

The President: Will anyone second Mr. Alexander's resolution?

Mr. W. Cullen (*Past-President*): I will second it.

The President: Are there any amendments or any discussion?

Mr. A. McArthur Johnston (*Vice-President*): I beg to move that the action of the Council be confirmed.

Mr. G. O. Smart (*Member of Council*) seconded the amendment, which was carried by 24 votes to 7.

COLOURED WRAPPERS FOR EXPLOSIVES.

Mr. W. Cullen (*Past-President*): The suggestion contained in Mr. Brett's letter has been made before, but that does not detract from its value; indeed, to a certain extent, it has already been adopted in other directions, *vide* the white wires attached to electric detonators for use in coal mines. Anything, however, which will decrease the risk of accident is worth investigating, consequently I made up a lot of dummy cartridges wrapped with paper of different colours, and tried them underground by ordinary candle-light. Speaking generally, under ordinary working conditions, it was found that red and green showed up best against the rock, so I have accordingly ordered a quantity of paper of these colours in order to try the experiment on a more extensive scale. The present wrappers are found to be distinctive enough for coal mining, but it stands to reason the colours would require modification for other conditions. For instance, white cartridges would show up best against haematite. It will take some considerable time before the paper I have just referred to will arrive, for a pigment will require to be found which will not affect the chemical stability of the explosive and, therefore the heat test, which is the index of chemical stability. This is far from being an easy matter, and I could not suggest any pigment straight off.

The paper for cartridge wrappers must be specially pure for the reason I have just indicated, and I should not be at all surprised if that is the main reason why coloured wrappers have never been used before. Of course, where a stick of explosive has been stripped of its wrapper by the force of the explosion there is nothing to be gained by having a distinctive colour, and I should therefore like to know from practical men whether it is more common to come across the stripped or the unstripped cartridge.

The President: I should like to thank Mr. Cullen for his note, and also to ask if there are any practical men present who can give us any information on the point he has raised?

Mr. J. P. Ward (*Member*): With regard to the coloured wrappers for gelatine, I think it is very seldom that a stick will come out of a hole as neat in appearance as Mr. Cullen's specimens on the table. The paper is always burst in ramming the gelatine home, while in case of a partial explosion, the paper of contiguous cartridges is blown to shreds. I think, however, it would be a great advantage to have the paper coloured red as it would make men careful—seeing red paper anywhere, they would look for and expect to find explosives. Besides, white paper, or old newspaper, is generally used for tamping cartridges. Personally, I should prefer the red wrapper, the bright red exhibited would be more distinctive than green amongst "blue" rock, and sufficiently distinctive in "free-milling" rock.

Mr. R. N. Kotze (*Government Mining Engineer*): In connection with this question of coloured wrappers another point appeals to me. It is not so much that explosives should be given coloured wrappers different from those now in use as that the various factories should distinguish their products in that way. When accidents happen in these mines, it is often very difficult to determine what particular dynamite was used, and from what factory it was drawn, especially in gassing accidents. If it could be ascertained from what particular factory the explosives were drawn which were concerned in accidents of various kinds, the knowledge would be useful. The provision of coloured wrappers would assist in gaining the necessary information. I think this is a point which might be considered by Mr. Cullen, and other people who manufacture dynamite.

DIAMETER OF EXPLOSIVES CARTRIDGES.

Mr. Wm. Cullen (*Past-President*): I have had it represented to me by practical miners that the $1\frac{1}{4}$ in. cartridge, at present supplied by all manufacturers, is too large for the bottom of the hole. They say in effect that what they want is a 1 in. diameter cartridge for the bottom, because the ordinary size seldom reaches the bottom, its diameter being too large, consequently a certain amount of drilling work is thrown away. What they ask is, that every 50 lb. case should contain a certain number of 1 in. plugs. I have made enquiries in several quarters, both official and private, but the answers received are so contradictory that I am left in very considerable doubt as to what is the best course to follow. I should like, therefore, to have the point discussed by the members either in the form of correspondence or at the usual meetings.

Mr. J. P. Ward (*Member*): With regard to the idea of including a few 1-in. sticks of gelatine in each packet of $1\frac{1}{4}$ in. gelatine, I do not think this very necessary, as in the few cases where the $1\frac{1}{4}$ in. is too large for the bottom of the hole, the sticks may be very quickly unwrapped, reduced in diameter by rolling in the hands and rewrapped to suit. The men can tell by the fit of the charging stick in the hole when this is necessary. The inch sticks might be preferred by some men but it is so vitally important to fill the bottom

of the holes tightly that I would not recommend them where $1\frac{1}{4}$ in. steel is used. On the mine I am now connected with, we formerly used 1 in. finishing steels in all our machine stopes, and 1 in. gelatine was procured to suit. It was found that in every contract, and for a period of about six months, the consumption of gelatine per fathom broken was considerably greater than formerly. It was decided to revert to $1\frac{1}{4}$ in. steel and gelatine and immediately the consumption became normal. It was thought at the time that the excessive consumption was due to the men not correctly gauging their charges, but as the test ran for six months, this may not have been so much due to that, as to a wrong distribution of the explosive in the hole.

The President: Do I understand that the men find a difficulty in gauging it?

Mr. J. P. Ward: They find a difficulty in estimating the quantity of dynamite to put into each hole.

The President: I think the use of some wrapper on explosives which would show itself is decidedly an advantage. In the course of my experience, I have found sticks of dynamite in all sorts of awkward places, and where you probably do not want to see them. Occasionally one finds a piece amongst the ore in the ore bins. You see a sausagy looking thing which bears a resemblance to the ore, and find that it is a cartridge of gelatine. If a coloured wrapper had been used a little bit of it would probably be found sticking on it, and might have called attention to it during previous operations. You sometimes find pieces on the sorting tables, passing the sorters and getting away down to the ore bins. With reference to Mr. Kotzé's remarks about the different manufacturers identifying their product by different coloured wrappers, that I think, is a good idea.

Mr. Lewis Evans (*Member*): I would like to say that the question of putting a number of inch cartridges in each packet of explosives has been considered by the Mine Manager's Association, and we decided against it. We think it preferable to have the holes made full bore at the bottom, because that is where we want the full power of the explosive.

FUSED SILICA WARE.

Mr. W. Cullen: At our last meeting Prof. Stanley referred to the use of certain forms of apparatus made from silica, and I gather from what he said that he was not aware that silica thimbles are made by the same two firms which have done so much to develop this special business. I saw them at a Royal Society Conversazione in London about 18 months ago, and although I cannot remember the price, they were infinitely cheaper than platinum.

The President: I understand that these thimbles are for parting and are made out of fused silica.

IMPROVED GLASSES FOR TESTING FOR POTASSIUM.

Dr. J. Moir (*Vice-President*): As you know, the usual test for K is done by examining the flame colour which the substance gives when held in the bunsen, through cobalt-blue glasses or an "indigo-prism." The principle underlying this is that these media cut off the yellow sodium light, but transmit the red potassium light. The great drawback of this simple test is that it is sometimes indecisive, for example, lithium, lead, arsenic, copper, etc., give flames which can be mistaken for the potassium flame when viewed through blue glass.

I have invented three different kinds of colour screens for this purpose, all of which are improvements on blue glass, and one of which is absolutely decisive even when lithium is present. They are made by staining gelatine films (old photographic plates "fixed" and washed), with various organic dyes, chosen so that the absorption bands of their spectra overlap, and therefore transmit only the extreme red (together with other colours, such as blue, which do not interfere). The films are then dried and cemented together.

The deep purple one is a combination of three films dyed respectively with fuchsine, methyl-violet, and haematoxyline. It transmits the extreme red down to C of the solar spectum, also a little blue round the F line. The greenish-blue screen is made of two films, one deep methylene blue (which cuts off lithium light) and the other methylorange. This transmits light from A to B and from E to F only. The third screen is single but is rather difficult to make; a film is dyed in very strong methyl-violet and then carefully dipped in picric acid until it transmits only the light from A to C. This screen is of no use if lithium is present; in fact it is quite similar to the deepest ruby-glass of the photographer.

The greenish-blue screen is very near theoretical perfection because, (1) the only red flame which it can transmit is that of potassium, and, (2) the green light which it transmits is not given off by any common substance, and, (3) the red K flame is seen in the midst of a contrasting background of greenish-blue due to the carbon monoxide of the bunsen flame and is therefore unmistakeable.

The principle of these glasses might be extended to the detection of other elements which give definite spectrum lines; it would be quite easy, for example, to invent combinations of dyes

which would transmit only lithium or thallium light.

The President: I am sure we are very much indebted to Dr. Moir for his interesting little note, and I wish to thank him for bringing the matter before us. I think this is another practical proof of the advantage of holding our meetings in this theatre, where we have facilities for such demonstrations as Dr. Moir has given us.

ALTERATION OF RULES.

The President: You will notice that on the back of the agenda paper, there is a notice of a special general meeting, to ask you to sanction the alteration of a rule in regard to student members. It has been thought by the Council that it is a pity we cannot extend our student membership in some way, and not confine it only to the students of the University Colleges. We want to give young apprentices and young men who have lately taken up cyanide and battery work, and to some extent mining, an opportunity of receiving our *Journal*, and of coming here and hearing our papers read, and if they feel inclined, to take a part in the discussion. Therefore, I need not read this proposed alteration. You will see that it covers the admission of apprentices or learners as well as *bona fide* students of a Technical College.

Mr. A. Richardson (*Member of Council*) read the following paper:—

THE PILGRIM'S REST GOLD FIELDS AND MINING METHODS.

By J. Moyle-Phillips (Member).

Geography.—The township of Pilgrim's Rest—the centre of the Pilgrim's Rest Gold Fields—is situated in the district of Lydenburg, in the Eastern Transvaal, and 30 miles N.E. of the town of that name. It is distant about 80 miles from Machadodorp, on the Delagoa Bay railway, from whence there is a tri-weekly mail and passenger service by coach. Heavy transport from Machadodorp and Pilgrim's Rest is effected by means of mule and ox wagon, the transport rate at present ruling being 4s. 6d. per 100 lbs. This high rate of transport is in a great measure accountable for the apparent by high mining costs, especially when compared to the mining costs on the Rand. Recently a mail and passenger service has been inauguarated between Nelspruit, on the Delagoa Bay line, and the Sabie Gold Fields, 13 miles south of Pilgrim's Rest, which it is eventually intended to extend to Pilgrim's Rest township. The Nelspruit route is, without doubt, the route par excellence for Pilgrim's Rest. Being only sixty miles distant, as against eighty miles to Machadodorp, it is not only the shortest route but runs through the centre of the Sabie Gold Fields and thus taps the two richest and largest mining camps of the district.

Ten miles N.E. of the township of Pilgrim's Rest is the eastern escarpment of the Drakensberg and from the edge of the Berg a splendid panoramic view of the "low" country can be had, extending—on a clear day—to the distant Lebombo mountains. In some places there is almost a sheer drop of 1,500 feet from the top of the Berg to the bottom. To the S.W. of Pilgrim's Rest and from eight to sixteen miles distant are Mount Anderson, 7,489 feet above sea level, and Mauch Berg, 7,340 feet high, two of the highest peaks in the Transvaal.

The Blyde river, a tributary of the Oliphant's river, has its source about ten miles S. of Pilgrim's Rest and it is from this river that the mines obtain their main supply of water power.

History.—The Gold Fields of Pilgrim's Rest were first worked in the early '70's as alluvial diggings. Gold to the estimated value of £10,000, at 75s. per oz., was obtained in '70, whilst up to '77, diggers took out gold to the value of £500,000. It has been estimated that close on 2,000 diggers were at one time engaged in and around Pilgrim's Rest, but at present, owing to the scarcity of water, which has decreased 38 per cent. during the last ten years, no digging is being carried out. The famous Pilgrim's Creek, from which the bulk of the gold was won, runs between the hills on which the Jubilee and Theta Mines are situated, whilst the township is situated on both banks of the creek, about a mile above its junction with the Blyde river. Reef mining was started about '76 and the first mine—5 stamps—was erected at Stanley Bush, six miles S. of the township.

Geology.—The auriferous beds of the Pilgrim's Rest district lie mostly in the shales of the Pretoria Series and in the Dolomites, and occur in the following descending order:—

1. Languages Reef, in shales.
2. The Shale Reef, in shales.
3. Bevets Reef, at the base of the Pretoria Series.
4. Theta Reef, roughly 100 feet below the base of the Pretoria Series.
5. Beta Reef, 300 feet below the base of the Pretoria Series.

Of these, the only ones which are being worked profitably at the present time are the Theta and Beta Reef by the Transvaal Gold Mining Estates, Limited.

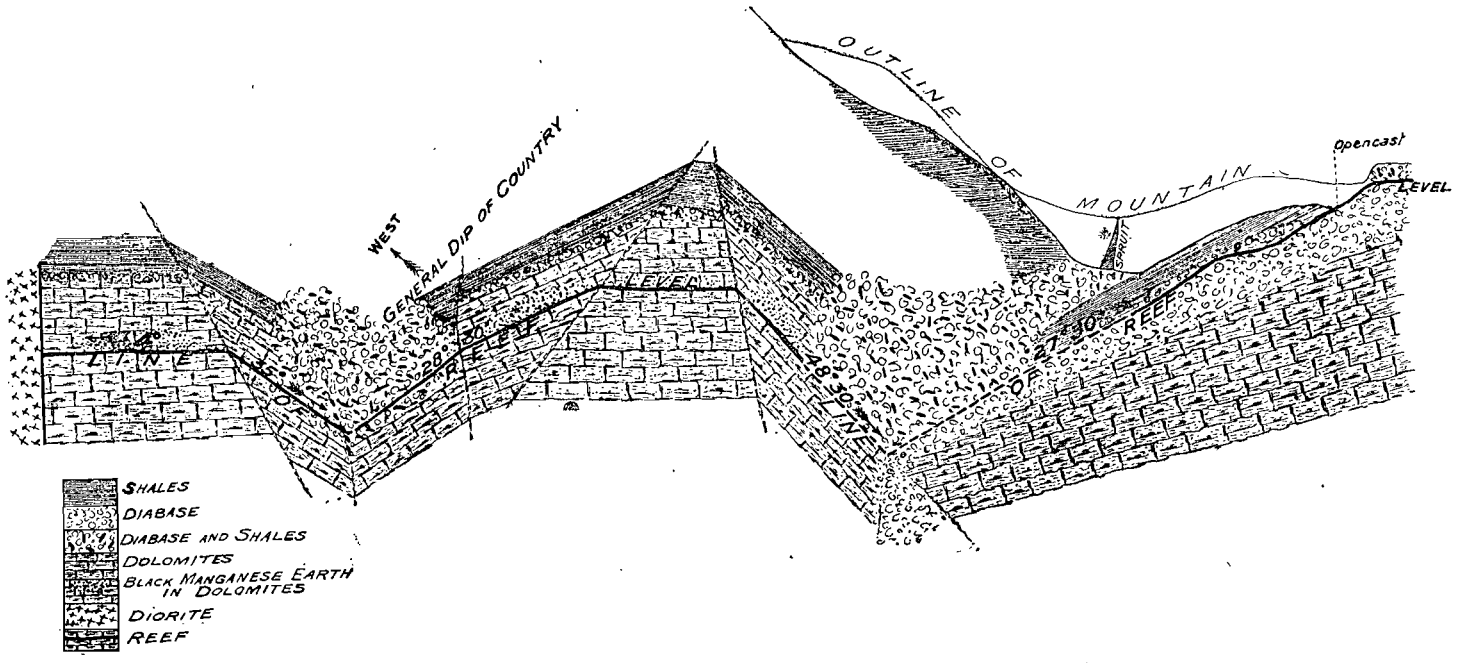

FIG. I.—Vertical Section along strike; scale 2,000 feet to 1 foot.

This section is taken along the strike East to West and shows how the strata has been disturbed. It will be noticed that the reef follows the undulations of the strata although it is evident that it was formed *before* the disturbances took place, as all dykes encountered penetrate through the reef and appear on the surface. There are several theories discussed here regarding the formation of this reef as to whether it was formed by infiltration, sublimation, or by the two, and also some people contend that there are not three separate reefs (Bevets, Theta, and Beta) but that the so-called Theta and Beta reefs are one and the same reef. It is said that the Theta and Beta reefs have never been found separately in one locality. On the Beta and Kameels Mine there is only one reef—Beta—whilst on the Jubilee and Theta Mines there is only the so-called Theta reef. Above the Beta Mine and on the contact with the shales is a reef corresponding to the Bevets, but there is no trace of the Theta Reef between it and the Beta Reef below. On the Clewer Mine again there is every indication of the three reefs —so-called—having come together, as in places the roof of the reef is the shales and there are distinct partings in the reef. The top portion is invariably the richest but very good values have been obtained from the bottom portions. On the Frankfort Mine there is a reef next below Bevets which the mining men of this district have not yet "placed" to their mutual satisfaction, some maintaining it to be the Theta, others maintaining it to be the Beta.

The Theta Reef may be termed a coalesence of two or three reefs, as, although in places it is met with intact, yet more often it is split up into two, and sometimes three parts, with a well defined parting between each. This reef runs from six inches to ten feet in thickness, but the average width may be taken as two feet. The Beta Reef is one well defined bed having an average thickness of ten inches, and stopes of only twelve inches in width have been worked in the Beta Mine. From two to thirty feet above the Theta Reef is a quartz leader from three to six inches thick, which is known as the "indicator." Immediately overlying this is a sheet of diabase two to forty feet thick, and next above is chert one to eight feet thick, and then the shales.

A remarkable body of ore was discovered in the Duke's Hill section of the Clewer Mine. It took the form of a gutter or channel with an average thickness of seven feet and increasing from fifteen to one hundred and fifty feet in width; after extending for some distance it suddenly terminated against a sharply defined fold of the shales.

The thickness of the strata between the Theta Reef and the diabase sheet is very variable, as in places the diabase forms the hanging wall of the reef whilst in others it may be forty feet distant, or, again, may be absent and the cherty dolomites or shales form the hanging wall. Pockets or masses of black manganese earth are frequently met with in the upper dolomites, and the conditions of development and stoping vary considerably in consequence. Wherever these earthy masses are encountered in developing, heavy timbering is necessary, whilst in the solid dolomites no timbering is required. Faults are frequently met with (Figs. I., II., and III.), and cracks in the formation varying from a mere fracture to several feet in width are often encountered, extending

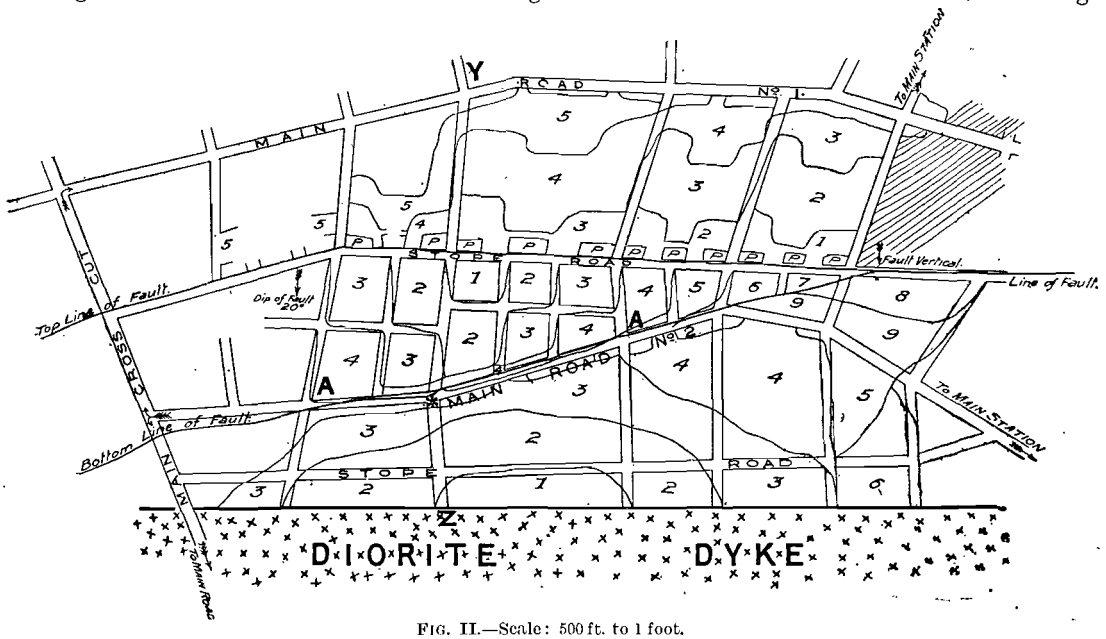

FIG. II.—Scale: 500 ft. to 1 foot.

From the main road No. 1 to the stope drives were put in until they reached the top of the fault. These were then connected by another small drive running on the line of the fault. Stoping was started by attacking one of the sections, from the stope drives on each side of it, ten feet away from the drive running on the fault. After the stope had been cut into for ten or twelve feet it was again connected with the fault drive thus leaving a pillar ten by twelve feet, a series of pillars were thus left along the fault. The stope was not worked out strictly on the long wall system as in coal mines but on the method shown, which gave a longer stope face and thus allowed of more hammers being employed at the same time. A pillar ten feet thick and the whole width of a section was left standing alongside the main road No. 1. The main road No. 2, and the drive parallel with the dyke, were driven simultaneously, the main road being driven along the bottom of the fault and the stope drive about ten feet away from the dyke: these were connected by other stope drives and the section thus blocked out. Stoping was done on the same method as adopted on the top of the fault with the exception that no pillars were left alongside of the dyke. In the meantime the section *on* the fault was being blocked out in the manner shown. When the bottom section had been stoped to within fifteen or twenty feet of main road No. 2, stoping operations were started on the fault. The tram line was broken at the point marked*, the pillar 1 was then taken out, and the remainder then attacked as numbered. The pillars left standing on the top section alongside the fault were taken out as they were reached and the whole section was thus stoped. Props were used to support the roof where deemed necessary, but 75 per cent. of these were afterwards withdrawn. From the point* the reef was trammed in both directions along the main drive to the main station. When the points A A were reached the roof collapsed but the faces were always at a safe distance from the fall. The only pillars left standing alongside main road No. 2 were on the dyke side, where, in places, the reef had pinched to six inches or less.

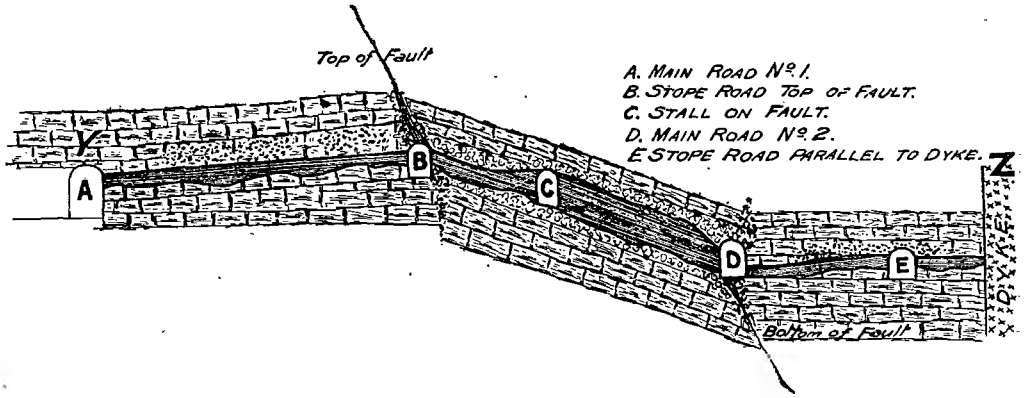

FIG. III.—Section from Y to Z. Scale 250 ft. to 1 foot.

A. Top main road. B. Stope drive along top of fault. C. Gallery.
D. Bottom main road, along bottom of fault. E. Stope drive parallel to dyke and 10 ft. from same.

The shaded portion indicates *Reef*. It will be noticed that the fault was entirely filled in with reef matter which was all payable and which opened out to eight feet thick in places.

both up and down through the strata to unknown distances and in places opening out into huge caverns. There are several dykes, mostly diorite, traversing the formation and one on the Beta Mine measures one hundred feet across, whilst others are from ten to fifty feet in thickness. The reefs themselves are composed of loose white opaque quartz embedded in, or surrounded by, a red or reddish brown earthy matter, probably the result of the decomposition of the pyrites which the reefs originally contained and which has left the quartz with a cellular or honeycombed appearance.

The gold in the oxidised portions is found "free," but there is an absence of "visible," although the values may run into ounces per ton: in fact the quartz itself carries very little gold, the bulk of it being in the loose earthy matter.

When the reef is highly pyritic the ore is very refractory, necessitating modification in the treatment at the reduction works. When the reefs are below twelve inches in thickness they are generally solid and require blasting, but above that width they can be generally worked out by picking. Around the outcrops and for considerable distances on the dip the reefs are oxidised, but as they get deeper under the strata they become pyritic. The dip of the strata averages 7° W., but in the vicinity of the mines it is only 3° 45′.

The mines of the Transvaal Gold Mining Estates, Ltd., are situated within a radius of 16,000 feet from the Central Mine and the lengths of their respective tram lines are:—

Clewer, 5,000 yds. Peach Tree, 2,735 yds.
Jubilee, 5,560 yds. Theta, 2,780 yds.

The following table shows the principal mines and the reef worked, also the height above sea level.

Company.	Mine.	Reef.	Feet above sea level.
	Frankfort	Shale Bevets Theta or Beta (supposed)	
Gras Kop Theta ...	Gras Kop Theta ...	Theta ...	5,200
Transvaal Gold Mining Estates Ltd.	Clewer ...	Theta ...	4,387
	Peach Tree	Theta ...	4,310
	Beta ...	Beta ...	4,220
	Theta ...	Theta ...	5,100
	Jubilee ...	Theta ...	5,525
	Kamel or Chi ...	Beta ...	5,185

Development.—Owing to the irregularities in the formation, and the numerous changes in the levels of the reefs, no uniform system of development can be strictly adhered to. The adits cannot always be driven at equal distances from each other but have to be put in at those points which, with due regard to economic haulage, are deemed best, but, whenever possible, the main drives are carried from 250 to 300 feet apart and the main crosscuts 500 feet apart. These blocks are again intersected by stope roads 50 feet apart. Main drives and crosscuts are driven 5 ft. x 6 ft. high, and when timbering is necessary the sets are cut with 3 ft. caps, bottom spread 5 ft., and 6 ft. vertical height. In the

solid dolomites no timbering is required, but where the black manganese earth is met with or the reef leaves the dolomites and comes in contact with the diabase, chert, or shales, heavy timbering is generally required. As showing the variable conditions of driving, the same drive may cost anywhere from 5s. to 45s. per foot, timbering included, but not the cost of the timber. The rate of development is from 40 feet in dolomites to 130 feet in shales per month, working double shifts. Each ganger on development has charge of from four to six faces which he has to keep timbered, if necessary, and has also to superintend his own tramming. All drilling is by hand labour, the average depth of hole in dolomites, per boy per shift, being 36 inches. Drilling is easy in the reef, but a boy has hard work to drill 36 inches in dolomite.

Stoping.—As no uniform method of development can be adhered to, neither, for the same reasons, can any particular system of stoping be adopted.

The size of stope roads depends on several conditions.

(1) If the nature of the surrounding strata allows, without undue timbering, the large mining trucks of 16 cubic feet capacity are led directly into the stopes and filled at the working face, but

(2) where this is not feasible, a smaller truck, 6 cubic feet, is utilized. This is filled at the working face and then either dumped alongside the main drive, to be again shovelled into the big trucks, or tipped direct into them—if the level of the reef is sufficiently high above the main roads to allow of this being done.

(3) When the extraction of the reef alone, does not give sufficient height for the passage of the stope trucks, the bottom of the drive is taken up to give the requisite height.

The general mode of stoping is to cut up the blocks by a system of stope drives and then commencing from the centre of the block, to extract the reef either by the "long wall" system or, when the ground is too heavy or loose, by the pillar and stall system. The latter method entails the splitting up of the small stoping blocks by means of galleries. When the block is thoroughly intersected in this manner the pillars are removed, commencing from the centre, and the roof allowed to fall in, but wherever practicable the "gobs" are filled in with the waste rock from development. When the reef is tilted at an acute angle as in Figs. 1 and 2 the overhand and underhand methods are resorted to, and thus in one particular section of the mine one may find all the different methods mentioned, in force, and this may even apply to one particular block in any section, such being the peculiarities of the formation in this district. However small the reef is, down to six inches in fact, it is taken out clean, that is, none of surrounding strata is broken with it, or, in mining parlance it is "resued." It may seem incredible, yet it is a fact, that stopes of only twelve inches in height have been worked in these mines, of course this has only been in those parts where the foot and hanging walls are solid.

These 12 in. stopes are still worked mainly by "umfaans." The ore is removed from the face to the stope roads on a sheet of iron roughly 2 ft. 6 in. by 2 ft. and slightly concave; a rope is attached to each end of this iron sheet, one end being held by the boy at the face and the other by a boy in the stope road, and thus the sheet is dragged to and fro along the floor of the stope. The gobs are kept open when required by means of props, or cogs, and chocks of mining poles or waste rock. Holes drilled in stopes are seldom more than eighteen inches deep, if drilled deeper they are very liable to excavate at the bottom instead of bringing their burden. The charge seldom consists of more than one cartridge, sometimes only half a one, of gelignite, this being sufficient to loosen the reef and allow of its being picked out. Stope gangers have two or more blocks to over see and have to put in all props, cogs, etc., and timber stope roads—if necessary, also "boss up" their own trammers. Each stope gang consists of from thirty to fifty boys, including hammer, tramway, timber, drill carriers, and boss boys.

Timbering.—The timber used in the mines of this district is the ordinary bush timber, such as is found growing in the kloofs on Government lands and private farms. A contract for the supply of mining poles is let by tender yearly and is for cutting and transporting poles 7 ft. long x from 4 in. to 9 in. in diameter. Poles exceeding these dimensions are called "specials" and are paid for accordingly. The majority of the poles come from below the Berg, from whence they are hauled to the top on aerial gears by means of steam hoists or oxen; from here they are transported to the different mines by mule or ox transport, the price, delivered on the mine, being:—9 inch 4s. 6d. each, 8 inch 4s. 2d., 7 inch 2s. 6d., 6 inch 2s. 2d., 5 inch 1s. 1d., and 4 inch 1s. The poles last from one to three years, according to the condition they are in when first used and the work they are utilized for, but they may have to be replaced much more often if subject to undue pressure or other adverse conditions. In the main roads it has been found necessary to replace them every two years.

As an experiment, poles of the black wattle were tried for sets in the main drives, and some of these have been standing for five years, but are showing signs of decay and will have to be replaced at an early date; but so far they have proved to be the best mining poles obtained locally. Blue gums have been tried but are found to be practically useless as props or legs of sets if subject to any great pressure, as they sheer their entire length in a very short time. If put in green they serve as a makeshift, but when once sun cracks have developed they are useless. There are two styles of notching used in framing the sets, which are locally known as the "Welsh" and "Cornish" notches respectively. As can be seen from Figures 4 and 5, the "Cornish" notch is cut

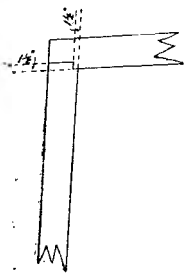

FIG. IV.—Cornish Notch.

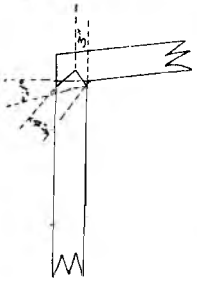

FIG. V.—Welsh Notch.

square whilst the "Welsh" is bevelled. The "Cornish" has been found to be the best suited to withstand the movements of the strata, and consequently is superseding the "Welsh." As the greatest pressure on the sets, &c., is from the top, the notches in the legs of a set to fit the cap piece are cut in the butt end of the pole, thus the pole—as the leg of a set—is erected in the opposite position to that in which it grew, or, in other words, is put in upside down. The first parts of a set to show signs of weakening are the notches.

Poles from six inches to nine inches are used for sets in the main drives and crosscuts, those from five inches to seven inches for the auxiliary crosscuts and stope drives, and the four inches and five inches for lagging; for props any of these sizes are utilized as occasion demands.

The upkeep of the timbering in the mines is a very large item in the working costs, but this again is somewhat counterbalanced by the low consumption of explosives as compared with the Rand mines.

Haulage.—The underground haulage is accomplished by means of boys, mules, and electric hoists. Boys push the trucks along the easy gradients, but where more than two boys are required a mule is used.

Unless the working faces are near the outcrop—in which case the trucks are taken out direct—the reef is trammed to a central station, from whence it is hauled in sets of three or more trucks to the surface by mules or electric hoists. The type of mining truck used is the V-shaped side-tipping of 16 or 11 cubic feet capacity, whilst, as already stated, a truck of 6 cubic feet is used in the stopes. After leaving the mine, the ore is dumped into a big bin, from whence it is taken to the mill in sets of 11 to 14 trucks of 30 cwt. capacity each.

On the Clewer Mine an aerial gear is employed to convey the ore from the top levels to the main bin, but self-acting inclines are more frequently used. The full sets are run to the mill by gravitation or electric motors. From the Clewer Mine the sets, after being started by the motor, are run by gravitation to the junction with the Jubilee Mine (4,000 yards). The speed is regulated by brakes on the trucks and by the motor. From the Jubilee Mine the ore leaves the mine bins in sets of six trucks of 16 cubic feet by means of two self-acting inclines, 660 and 500 yards long respectively, to a station on the top of the main incline. From this point another self-acting incline lowers it to the bins at the foot of the mountain, 1,100 yards distant, of an average gradient of 1 in 5. From the foot of the mountain it is taken in sets of 10 to the junction with the Clewer Mine, 2,800 yards distant, by gravitation, and from the junction the trucks are hauled by motor to the mill, a distance of 1,000 yards.

From the Theta bins the trucks—in sets of 6 of 16 cubic feet—first pass down a self-acting incline 350 yards long, and are then taken along the contour of the mountain (2,000 yards) by mules to a point immediately overlooking the crusher house at the mill, from whence they are lowered by another self acting incline 430 yards long.

The ore from the Peach Tree Mine is first conveyed down a self-acting incline—155 yards—to the main bins, from whence it is run in sets of 14 trucks of 30 cwt. to the mill by gravitation, 2,580 yards.

The trucks from the Beta Mine, after travelling 100 yards by gravitation, join the Peach Tree line. From the main station at the mill the trucks are hauled up an incline by electric hoist to the crusher house.

Pumping.—There is only one pump in use in the mines of the Tranvaal Gold Mining Estates, Limited. This is at the Clewer Mine. It is a centrifugal with a 3 inch delivery and is electrically driven. Working in spells of 20 minutes duration four times per 24 hours it successfully copes with the water.

Whenever the water is met with elsewhere it is conducted to some fissure in the strata down which it disappears.

In the dry season the mines are comparatively dry, but when the rains are on, the water percolates through the soil and finds its way by means of the fissures, &c., into the mine.

Employees.—The white population on the mines is a very cosmopolitan one, although among the mechanics and staff the Scotch predominate, whilst among the miners the Welsh are in the majority.

The wages ruling are :—

Mechanics	... ...	20s. per shift.
Electrical Shiftmen	...	£25 per month.
Stopers	... from 15s. to 20s.	per shift.
Developers and Timbermen	20s.	,,

In the mine the day shift is from 7 a.m. to 6 p.m. and the night shift from 7 p.m. to 6 a.m., each shift being allowed an hour off for meals.

The white employees are provided with free quarters. At the central works there is a boarding-house for the convenience of the mechanics and mine men, but on the mines the men generally form messes among themselves and employ Kafir cooks. The cost of running a mess ranges from £4 to £6 per month, whilst at the boarding-house £8 is charged.

Owing to the mines being so scattered, there is no recreation hall as on the Rand, but there is a Sports Club which embraces every kind of game, and of which nearly every employee is a member. The average cost of living—outside of food—is much higher than on the Rand, owing, no doubt, to the high rates of transport. Each employee contributes 10s. per month for medical attendance, medicine and hospital. A Medical Officer is in attendance for both white and coloured employees, and there is also a hospital with a full equipment of medical and surgical appliances, including an X-ray apparatus.

Natives.—The natives employed are mostly local boys and Basutos, and, whilst the supply at present is about equal to the demand, it is very erratic, four months being the average length of a boy's contract.

The wages per 30 day month are :—

Boss Timber Boys ...	from	50s. 0d.	to 60s. 0d.
Drill and Pick Boys ...	,,	42s. 6d.	,, 50s. 0d.
Trammer Boys ..	,,	30s. 0d.	,, 45s. 0d.
Picanninis (drill carriers)	,,	7s. 6d.	,, 20s. 0d.

There is the usual allowance of meal and fresh meat per boy.

Wood and iron brick-lined compounds were erected for native accommodation, but they prefer living in grass huts as at their kraals, consequently the land around the mines is dotted with those huts, around which is generally a mealie patch planted by the boys for green mealies.

For cooking purposes the boys are squadded into gangs of from 16 to 25 with a cook to each gang, the food being cooked in the ordinary 3-legged Kafir pot. There being numerous kraals in the vicinity of the mines, the week ends are generally devoted to "tywala" drinking, which is brought in by the wives and sisters of the boys working, and as the mine officials keep the supply within reasonable limits there is no loss of time through drunkenness, and the boys appear to be quite content.

General.—Around Pilgrim's Rest are several small mining propositions which are worthy of note.

The Grasskop Theta Syndicate is working claims 1½ miles S.E. of the township on the hills opposite the Jubilee Mine, whilst adjoining ar several smaller concerns held on tribute and in course of development which are turning out well. N.E., and about 15 miles down the Blyde river, is the Vaalhoek Mine (Transvaal Gold Mining Estates, Limited), on which it is intended to erect a 10-stamp mill, whilst adjoining it the Grasskop Theta is working some claims. These are situated on the Glynn's Horizon—below the Beta Reef. Nearer home and more to the N. is the Frankfort Company. 18 miles S. of Pilgrims' Rest are the Glynn's Lydenburg and the Glynn's Extension Companies, working the Glynn's Reef. Quite recently a new find has been made close to the Sabi Falls in the sandstones and about two miles N. of Glynn's Reef; several small mines are at work here and doing well; these are in the hands of private individuals. To the E. of Pilgrims' Rest is the defunct—as far as working is concerned—Lisbon Berlyn Co., and also the farms London, Kimberley, Belvedere, and several others, on which are very rich leaders from 1 inch to 6 inches thick but very patchy: several of these have been and are being worked. 25 miles W. are the Finsburg and Nooitgedacht mines on the Finsburg, Button and Davidson Reefs, lying in the shales: these are being worked on tribute to the Transvaal Gold Mining Estates, Limited.

In and around Pilgrim's Rest are numerous small propositions worthy the attention of the prospector and small capitalist. The reefs are generally easily worked, and the power (water) is ample for anything up to a 5-stamp proposition. As these small mines are likely to attract more attention in the future, some notes on them may not be amiss. Undeveloped claims fetch about £20 to £25 each and developed claims about £100, the price depending on assay values and economic conditions. Leases are generally let for three years, with an option of another two years, and at 10 per cent. on actual gold won. Claim licences amount to 2s. 6d. per claim on Govern-

ment ground and 5s. per claim on private ground.

The following was the price paid for a 5-stamp mill with a portable engine :—

Cost in Johannesburg ...	£375
Railage	£30
Transport to mine	£60
	£465

The stamps were 750 lbs. weight each. This price does not include any cyanide plant or any erection costs.

The gas engine is highly thought of where water power is not available, and under these conditions the 1-stamp gas mill seems but suited for these fields; the duty is high, up to 25 tons per 24 hours, whilst the cost of fuel, 2s. 6d. a bag for charcoal, is low. A Tangye suction gas engine driving 10 stamps, crusher, and two centrifugal pumps at Barberton, consumed 3½ to 4 bags of coal per 24 hours, equivalent to 1 lb. coal per horse power per hour; when using charcoal the consumption was 7 to 8 bags per 24 hours.

Charcoal has the advantage of leaving no tar and no clinkers are formed in the gas generator. Another Tangye suction gas engine plant belonging to the Messina Development Co., Ltd., rated at 116 B.H.P., but only loaded to 40 or 50 B.H.P., ran during one month 30 days 14 hours, including a non-stop run of 21 days 6 hours The fuel consumption amounted to 443 bags of local charcoal of 65 lbs. weight each and costing 2s. a bag, equal to £44 6s. 14 per cent. of dust, fines, and waste was sieved out, leaving 380 bags actual consumption. On the basis of 443 bags, the consumption was 0·84 lbs. of charcoal per B.H.P., per hour, or a cost of 0·36d. per B.H.P., per hour, or, on a basis of 1,300 tons, 1s. 6d. per ton crushed.

A scheme is on foot to erect an electrical supply plant, guaranteed by the Government, in the vicinity of Pilgrim's Rest, and it is estimated that a 5-stamp proposition can be run by this means at a cost of £30 a month.

The President: I would like to move a hearty vote of thanks to Mr. Phillips for his interesting paper. I think this is the first description we have really had of the Pilgrims' Rest goldfields, and we have had the advantage of having the paper read by one who knows the district, and is able to lay stress on various little points which might otherwise have escaped notice.

The vote of thanks was agreed to.

Mr. W. A. Caldecott (*Past-President*): I should like to offer a comment upon one point in this paper with which the author does not deal very fully. He mentions the gold in the oxidised portion of the ore as being found free, and that nevertheless there is an absence of "visible." In my experience of the ores of that district a good many years ago I also observed this fact, and I think that it is due to the extremely fine state of division of the gold so that when the ore is panned the gold floats in the water without sinking to the bottom. As an illustration of this, I may say that on one occasion I came across a prospector who showed me, with a certain amount of pride, the reef that he had struck. On asking him what he thought its gold contents per ton were, he estimated the value at 10 dwt. The assay value turned out to be 46 dwt., but the 10 dwt. estimate was probably not much, out so far as possible recovery by amalgamation only was concerned. In the days before the introduction of the secondary process of cyaniding, and when exceedingly rich ore was treated in batteries and grinding pans in the Pilgrim's Rest district, the loss of fine float gold in the tailings flowing into the Blyde River must have been very high.

Mr. W. R. Dowling (*Member of Council*) read the following paper :—

NOTES ON PRECIPITATION.

By Mather Smith (Member).

The object of this paper is an attempt to prove, that the generally accepted principle that it is necessary to have one cub. ft. of zinc shavings, for the precipitation of one ton of solution in 24 hours is wrong; that more zinc than is necessary, is being generally used; that an excess of zinc, is both harmful and costly; that precipitation boxes, are often constructed on a wrong principle; and that a comparison of the rate of flow of solution per cubic foot of zinc, which does not also include or consider the difference in area and number of compartments in the precipitation boxes, is misleading. If one cub. ft. of zinc is required for the precipitation of one ton of solution in 24 hours, then, by simple multiplication, 2 cub. ft. of zinc will be necessary for the precipitation of 2 tons of solution, in 24 hours, and this is the rule which appears to have been acted upon by those responsible for the construction of the boxes on some of our latest cyanide plants.

But there are four different possible ways of arranging these two cub. ft. of zinc, as they may be either in series, vertically or horizontally, or in parallel, separately or together. Although in each arrangement the total flow of solution would be the same; with those arranged in series it would flow through each cubic foot of

zinc at the rate of two tons per 24 hours, whereas through those in parallel it would be at the rate of one ton per 24 hours only,* and does it not seem reasonable to expect that those arranged in parallel would give better precipitation than those in series? At any rate, the danger of having gold slimes in suspension carried out of the boxes would, in those in parallel, be reduced to a minimum.

One group of mines started several years ago dividing their boxes of eight compartments, into two boxes of four compartments each, and are now installing, low wide boxes of four or five compartments each. If these are giving good results, why are boxes of seven and eight compartments still being constructed? If four or five compartments are sufficient; what is the necessity for the other two or three?

In my opinion, the superfluous zinc does a great deal of harm, as it must occasion an unnecessary consumption of cyanide, and charge the solutions with an excess of zinc, which, in circulating through the plant, will in a great measure, be deposited as white precipitate at the heads of the boxes. Many will disagree with me in this, but I have personally observed, on two different cyanide plants, on which the sands and slimes solutions were at first run separately, that when the weak solution from the sands plant was used on the slimes plant, there was an immediate increase in the amount of white precipitate in the slimes boxes which I could only attribute to the zinc from the sands solutions. May it not also be possible, that solutions which are comparatively free from zinc, will allow of better precipitation than those which are heavily charged?

Another thing which must still further reduce the area required for the free passage of the solution, is the way in which, on many plants, the zinc is pressed down into the boxes. It is surely unnecessary in an electrical process of precipitation, to endeavour to get each and every molecule of the solution into actual contact with the zinc. The solution gradually forces open channels for its free passage, and I have frequently seen, on more than one plant, the solution rising up through one or more circular holes in the mixture of zinc and white precipitate in the top compartments of the boxes, and making small geysers on the surface. The same thing must be taking place, although not visibly, in the other compartments, and must obviously tend to carry off gold in suspension.

I had long held the opinion that more zinc than was necessary was being generally used, and on taking over the management of the cyanide works on the Worcester G. M Co., Barberton, at the end of April last; I started putting my theories into practice. There were here four precipitation boxes, three for the sands solution, and one for the slimes. The sands boxes had each six compartments full of zinc, and the slimes box had seven. Each compartment measured 36 in. wide, 22 in across, and 24 in. deep to the top of the sieve and all were packed as full as possible with zinc. I took samples of the solution sumps, and had them assayed with the following results:—

		dwt.
Strong sump	...	·6
Medium sump	...	·225
Weak sump	...	·225

I cannot find the record of the slimes sump assay, but it was also very high in gold.

I am not trying to prove that the large amount of zinc in the boxes had anything to do with the bad precipitation, but I am sure that many cyanide managers have experienced the same difficulty which I then had, and that was, to know what to do with the zinc in the boxes so that I could replace some of it with fresh zinc. There was no room for any at the heads of the boxes, and I had to run down a lot of it with sulphuric acid, in order to make room for fresh. Under my present system, there is always plenty of room at the heads of the boxes, so that should anything go wrong with the precipitation, and I have known that to happen on many plants, the boxes can be dressed up without any trouble.

To continue, each box, at the end of April, contained, on an under-estimate, 60 cub. ft. of zinc, and the solution during that month had averaged, in the sands boxes 47 tons per box per day, or ·78 tons solution per cubic foot of zinc per 24 hours. I started reducing the amount of zinc in the boxes, and now, instead of putting in 24 in. of zinc into the bottom compartments, when dressing a box, I put in 12 in. only. This afterwards settles down to about 9 in. It took about three months to gradually reduce the quantity of zinc to its present state, namely, five compartments in each sands box, with an average depth of zinc in each compartment of less than 9 in. As the sands tonnage treated has considerably increased since then, and the charges are also getting more solution, the sands boxes are now running at an average rate of 63 tons per box per 24 hours, or over 3 tons of solution per cubic foot of zinc per 24 hours. The weak box does more than the average. I had the slimes box made into two boxes, the one of four and the other of three compartments. The one contains, on an over-estimate, 16 cub. ft. of zinc, and the other 12. The usual flow of solution is at the rate of at least 65 tons per box per 24 hours, which works out at 4 tons of solution per cubic foot of zinc

* The rate of flow per sq. ft. of cross section being reduced by half.

per 24 hours, in the one box, and at 5·4 in the other.

I will now explain my method of dressing the boxes. When the strong or medium boxes require more zinc, the zinc from the top compartments of the slimes or weak boxes is lifted out and shifted over into them. The second compartment of the weak or slimes box is then lifted slowly up by the handles attached to the tray and put into the top compartment, tray and all, without any handling or disturbance of the zinc, and so on to the last compartment, which is dressed with new zinc.

This is usually done every fifth day, so that no zinc, in the one slimes box, is ever more than fifteen and in the other more than twenty days old. In the weak box, also, no zinc is ever more than twenty days old, as two compartments are frequently shifted out.

New zinc is put into the bottom compartments of the strong and medium boxes twice a month. No free cyanide drip is used, and the zinc, not having been disturbed, starts precipitating immediately, and, were it not for occasional traces, I might say that there is an entire absence of white precipitate.

The zinc in the boxes being shallow and loose, the gold slimes do not clog up the boxes, but fall to the bottom of the tray and leave the zinc clean and fresh.

My reason for continuing to fill four compartments with zinc in the one slimes box, when the other, with three compartments and running at the same rate, gives equally good results, I am unable to give, and can only attribute it to prejudice and years of training in the "more zinc better precipitation" theory.

I find that five compartments are necessary in the strong and medium boxes, as the zinc in these boxes, having been longer in use, appears to have lost some of its efficiency as a precipitant.

A sample of 100 c.c. of the solution leaving the slimes boxes is taken every 8 hours, and 20 A T.'s of the mixture are evaporated for assay at the end of each week, and since 9th August nothing higher than a trace has been returned from the assay office. Owing to the pressure of work in the assay office, the solutions from the sands boxes have not been sent up regularly, but so far, the highest assay from solution leaving the boxes has been one of ·01 dwt. from the strong box. The weak and medium boxes have been "nil" and "trace."

It has been suggested to me that the solutions here may allow of quicker precipitation than those on the Rand. In that case, why were the sumps here full of gold at the end of April last, although the boxes were full of zinc? It has also been suggested that my system of dressing the boxes would not be suitable on a large plant, as it would mean much extra work. I fail to see this, as it would simply require stronger trays and a proportionate increase in the number of assistants. I can dress my five boxes, with two boys to help me, in a little over an hour.

I attribute the good precipitation, to the freshness of the zinc, to the freedom of the solutions from excessive zinc in solution, to the freedom with which the solutions can pass up, without making channels through the zinc, and also to the fact that the hydrogen evolved has free egress and does not clog up the zinc with bubbles.

My actual extraction from the sands for the past three months, in spite of an increased tonnage treated, and a shorter treatment, has been 4% better than the average of the eighteen previous months of which I can find a record, and I strongly hold to the opinion, that the freedom of the solutions from zinc has a lot to do with the improved extraction. My experience here has convinced me, that if precipitation boxes were constructed to allow of one square foot, in area, for the precipitation of one ton of solution in twenty-four hours, with a maximum height of twenty-four inches from the top of the zinc frames to the top of the solution, and with a maximum of five compartments in each box, there would then be no possibility of any mistake being made in the construction of precipitation boxes.

I append the following table showing the consumption of zinc and acid, and the value of the gold slimes, for October and November, 1908.

	Oct.	Nov.
Pounds zinc used per ton treated	·19	·25
Pounds zinc used per ounce fine gold ...	1·2	1·8
Pounds sulphuric acid used per ton treated ...	·09	·07
Pounds sulphuric acid used per ounce fine gold	·55	·5
Percentage fine gold in gold slimes ...	33·8	35·2

The zinc used includes waste, which, as we have a very poor lathe, amounts to a good deal.

Since writing this paper, I have had two high assays from solutions leaving the boxes, the one of ·2 dwts. from the slimes boxes, and the other of ·075 dwts. from the strong, but as these were sent up during "clean-up," and the next assay from the slimes boxes was again "trace," I am sure that they were "salted." I have often had trouble here with salted samples taken at "clean-up," as the smelting room and extractor house are all in one.

The President: On behalf of the Society I beg to thank Mr. Mather-Smith for his interesting paper.

NOTE ON A PROBLEM DURING SHAFT SINKING.

(Read at September Meeting, 1908.)

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

REPLY TO DISCUSSION.

I regret that no members have seen fit to bring forward any problems that must have been met with in some of the shafts of the Rand during recent sinking operations; surely the work cannot have all been plain sailing.

At the risk of wearying you (and jumping in where wise men fear to tread), I will briefly describe a second and somewhat similar though more difficult and dangerous problem, that faced us in the Turf shaft, and our method of overcoming it.

For three months after settling the trouble described in my last note, excellent sinking was done, an average of 159 ft. for that period was maintained; then at 3,100 ft., or 540 ft. below, that bugbear water was struck in the north-east corner of the shaft, and then we sank 74 ft. only. At 3,140 ft. the formation became disturbed, and a fault appeared in the same corner, with a dip of 70°—80° south-west. At 3,170 ft., a narrow dyke with a dip of 73° south-west and parallel to the faults was cut, which had 6 in. to 18 in. of soft mud on the footwall, along which the water was flowing. The water now increased to some 30,000 gallons per 24 hours. No one has any idea what a drawback to sinking, even a small quantity of water, is when on the shaft bottom. At 3,195 ft., the position was as per Fig. I.

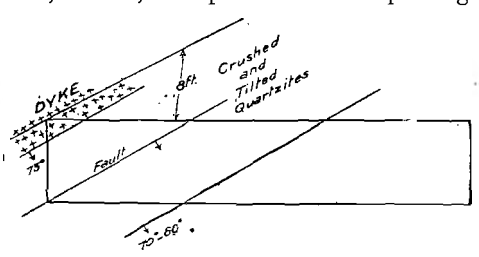

Fig. I.—At 3,195 ft.

The soft mud was being washed away by the water, thus undercutting the formation on the east side, which was already broken up by the faults. At this point the north-east side was breaking 8 ft. wider than necessary, the ground coming off to the dyke. At 3,171 ft., bearers were placed under the north end piece in good deep hitches, and at 3,177 ft., bearers were put under No. 1 divider, but after sinking further, the hitch on the east side became so weak, although cut in 6 ft. from the timbers, that it was found necessary to put in a vertical leg 30 ft. long, and 12 in. × 12 in., under the bearer, with the foot let into the solid quartzite through the dyke and cemented in. This leg was secured to the bearers with steel straps. At 3,190 ft., the bearers under No. 3 divider, where the wall plates are jointed (I may here mention that the wall plates are sent down in two halves, and butt-jointed, the No. 3 divider with studdle binding the joint), were also found to have a very insecure hitch on the east side, although cut in 12 ft. beyond the shaft timbers, and as the ground above was very heavy (because of the water washing out the decomposed dyke) and getting heavier every hour, and the formation below the hitch being broken, a diagonal bearer 30 in. deep was put in with a strong hitch on the west side and cemented. The wall plates were strapped to this diagonal and to the horizontal bearers, with steel angle iron, and heavy blocks were firmly wedged in as stiffeners, between the diagonal under No. 3 divider, vertical leg under No. 1 divider, and north end of shaft, so that any movement was impossible.

On referring to Fig. II., which is a vertical section north and south of the shaft, looking east, it will be seen how, as sinking was continued, the support or toe of the broken ground between the dyke and faults was being cut away and, with the water, making the east side very dangerous; thus when the shaft got to 3,226 ft. the problem became exceedingly ugly.

On the east side, "A to B" was the dangerous area, and the west side, "C D" was broken up, and tending to fall away from the dyke into the shaft, and between these points no solid ground could be found on which to place bearers; if solid on one side, it was weak on the other. All sinking was stopped, and the following method adopted, to overcome the difficulty.

Under the north end piece, good hitches were cut in each corner, and 36 in. bearers put in. Under the No. 3 divider, where the wall plate joint comes, a good solid hitch was cut on the east side, and a long 5 ft. hitch, which was not over good, on the west side, and 36 in. bearers put in. These bearers are east and west across the shaft. On these again were laid, on the west side from north end piece bearers to No. 3 divider bearers a beam 14 in. × 14 in. and 24 ft. long, and into this beam were dovetailed timbers, 16 in. deep, under north end piece and under No. 1

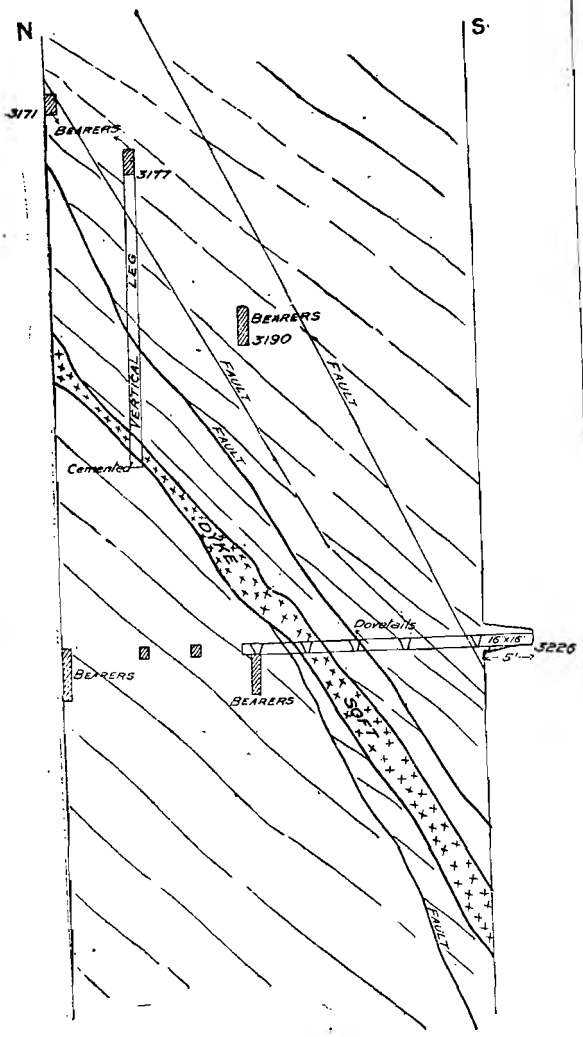

FIG. II.

2 and 3 dividers; each of these timbers was cemented into solid hitches on the east side. From No. 3 dividers into a hitch cut 5 ft. into the south-east corner, was placed a beam 16 in. × 16 in. and 28 ft. long, and into this beam was dovetailed 16 in. deep timbers under south end piece, and under Nos. 3, 4, 5 and 6 dividers, and each of these timbers was cemented into good hitches on the west side. (See Fig. III.)

Thus a solid framework was built in with supports, whenever sound ground could be formed, so framed that both vertical and side pressure would be resisted. A strong close bottom was put on this, the 3 ft. centre setts were close lagged, and the empty space behind, filled in with rock from the bottom, as previously described. Sinking was then continued with great care so as not to disturb the job with too

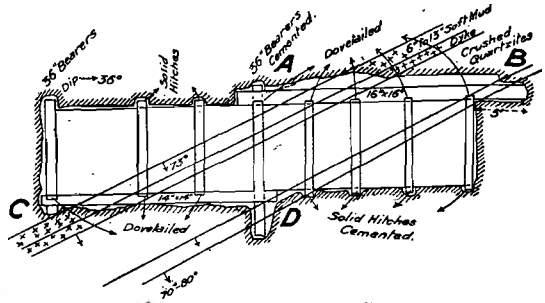

FIG. III.—At 3,220-30 ft.

heavy blasting. The ground on the west side on further sinking became bad, and the hitch under No. 3 divider bearers at 3,226 ft. became under-cut because of the faults and dyke. (See Fig. IV.)

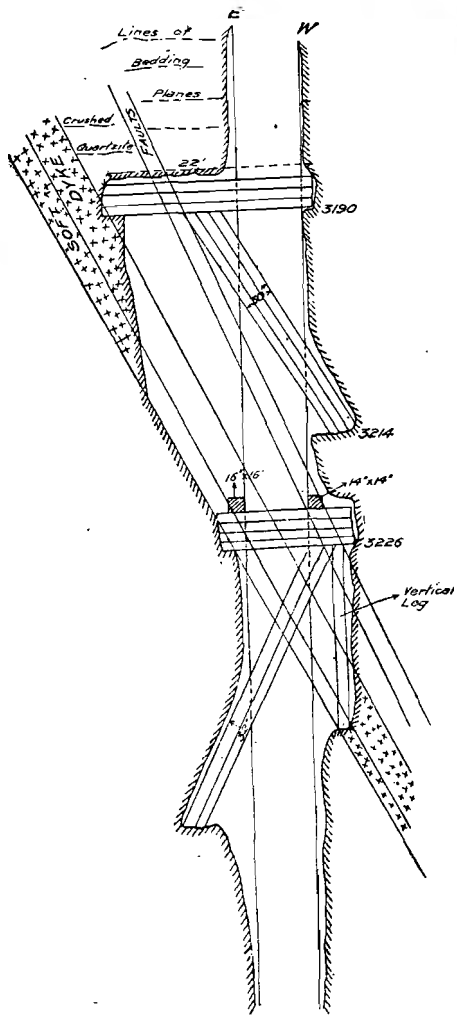

FIG. IV.

A vertical leg was put under, to take the place of the hitch, but as no satisfactory foothold could

be got for this leg, a 32 in. deep diagonal was put in under the 3,226 ft. bearers on the west side, and into a good hitch on the east side. Thus the shaft was secured with a lattice work of diagonals through the disturbed area. At 3,294 ft., when once more in solid regular formation, a complete sett of heavy bearers was put in across the shaft as a final security.

I may mention that during sinking over 3,600 ft. in this shaft, seven distinct dykes have been passed through. When hard, but bringing in no water, progress is slow, the timber is damaged with blasting, but the sides are usually fairly good and ordinary care only is required. But when soft and wet dykes, either separately or both together, are met with, work is very dangerous and complicated, and I can say here with pride, that in spite of the several troubles encountered, no life, black or white, has been lost in overcoming them; and this I attribute to the care exercised by all concerned.

The President: I am sure we must thank Mr. Saner very much for bringing these extra notes before us. He seems to have encountered great difficulties in sinking, and to have shown a great deal of ingenuity in overcoming them.

WHITE LABOUR IN MINING.

(*Read at January Meeting, 1909.*)

By Tom Johnson (Member).

DISCUSSION.

Mr. Ralph Stokes (*Member*): The author pleads with such obvious sincerity for the employment of more white men underground, that it is unfortunate he does not support his case with a few more decisive facts in place of general assertions, highly theoretical estimates, and an appeal to sentiment. At times one may even suspect that he desires to involve the Society in a debate on social politics, and that his next theme will be women's suffrage, or the licensing bill. It is surely not within the province of this Society, wide though it be, to advocate the cause of white labour, save on purely economic grounds So far as the business side of his paper is concerned, the author leaves his first proposition—that more whites be employed on shovelling and tramming—in a state of such obscurity that it is fairly safe from criticism

The arguments advanced are, too shadowy for attack. One might as well go shooting protoplasm with an air gun However, the author does commit himself to the vague statement that the Chinese were better than the natives, but "did not get anywhere near" what he has "seen white men do." Such general opinions as these, do not help us to see how the whites are to be handled, and formed into a steady working force, to compete successfully underground with Kafirs, and at the same time earn a decent living wage. I hope he will touch upon this trivial question in his reply. When the smooth working of a "white" lashing and tramming system in other countries is cited, it is necessary to bear in mind, that this class receives as a rule, only about 30% less money than the competent machine man.

The author declares that we are too much in the hands of the natives, and that "stranger things than a strike amongst Kafirs have happened;"—true enough, stranger things have happened, but a strike amongst white shovellers is not one of them.

The paper contains repeated references to how the unskilled whites on the Rand are "looked down upon," by the "aristocratic skilled men" who are said to forget, how short a time has elapsed since they themselves were "unskilled." His vivid picture of their contempt, is most astonishing. Few imagine the Rand miner to be such a snob. But the author's diagnosis is, I think, entirely at fault. The skilled man's antipathy is not a system of contempt, but is due on the contrary, to a full realisation of the narrow margin mentioned, and to the consequent dread of greater competition.

Turning to the second part of the paper, *re* "more whites on machines," it is satisfactory to find definite estimates given. For the purpose of his comparison, the author discreetly takes low mining efficiencies upon which to improve. For his representative case, he assumes an average fathomage per machine shift of 5—the best men breaking ·8, and others down to ·3. As the average for five mines, for which I happen to have the figures, is ·7, the author seems to have a good deal of lee-way to make up before he reaches present standards under well run normal systems. We find him taking his ·8 men off their machines, by which they were probably making £2 a shift, and making them professors of mining to the lower ranks, at 19s. per shift. This, of course, would put the expert miners in a proper spirit of ardour, for the success of the scheme. The ten drillers supervised, would likewise welcome the minimised prospect of making big contract cheques, to which they might have risen with greater experience by putting their brains into their work, under the good influence of personal responsibility.

As an auxiliary and apparently not essential part of his scheme, the author advocates the introduction of a rigging and blasting gang The experiment has already been tried, I believe

with promising results, and as those responsible, may perhaps favour the Society with their conclusions, the idea need not be theoretically dealt with. Taking the first proposal on its own basis, we may compare costs as follows :—

Under normal conditions, an average of ·7 of a fathom per machine shift, can be maintained under keen and helpful supervision. Let this be reduced to ·6. Then to break 74 fathoms, 123 machines are required, and a labour force of 41 miners and 287 natives. With gelatine at 16 lb. per fathom, costs appear as follows :—

41 miners at 30s. a shift	...	£61 10
287 natives at 3s.	...	43 0
Gelatine at 47s. per case	...	55 10
Fuse, detonators, etc.	...	7 0
Total	...	£167 0

Cost per fathom, 45s.

This figure is not merely an estimate, but practically coincides with the actual cost on five representative mines for six months.

Now, in the author's case, he first of all assumes the low average of ·5 fathom per machine shift. By taking off his ·8 men, this average is automatically reduced to about ·45, which he has to raise to ·7 by special supervision. Even allowing this unreasonable improvement of 55%, the possibility of which predicates an abnormally poor prior standard of mining and supervision, and also allowing the men to do their own charging and blasting, his costs compare most unfavourably.

74 fathoms would require 106 machines, with 106 white drillers, 10 bosses, and 141 boys ($1\frac{1}{3}$ per machine). Then

116 whites at 19s.	...	£110 0
141 natives at 3s.	...	21 0
Gelatine at 47s. per case	...	55 10
Fuse, detonators, etc.	...	7 0
Total	...	£193 10

Cost per fathom, 52s. 6d.

This means an increase of 17% in cost to the mine and a 33% decrease in average earnings to the men.

If we work out the two schemes on a more rational basis, giving the expert men their former earnings and granting, say, a 30% increase in efficiency, the loss, of course, is greatly increased. But the author's estimate is given the benefit of every doubt. Furthermore, it is assumed that suitable instructors can be found to effect the 55% increase in efficiency, and to benevolently give away the hard-won secrets of their trade to men who would become their competitors when the scheme collapsed. After a year or so of this costly tuition, the men would be individually well nigh as efficient as the boss, and then to give his policy support on grounds of economy, the author would have to manipulate his estimates with some ingenuity.

THE GASES RESULTING FROM THE USE OF HIGH EXPLOSIVES.

(Read at November Meeting, 1908.)

By Wm. Cullen (Past-President).

DISCUSSION.

Dr. J. Moir (*Vice-President*): I should like to call the author's attention to the fact that neither I nor he is the pioneer in showing that explosives give off CO on firing. In Sanford's* book I find (p. 142) an analysis of gelignite fumes (apparently obtained by firing in a bomb) giving 7% of CO along with 25% of CO_2 and 67% N_2. This analysis is sixteen years old, and is the first in which CO appears.

The President: I am sorry there is no discussion on so many of the papers before us. Some time ago we had a plethora of discussion and but few papers, now we have plenty of papers and no discussion. The main object of contributors in giving us these papers is to have discussion upon them. I think our members should take this to heart and see if they cannot manage to raise some discussion on these papers. It is disappointing to the authors, and the discussion is often far more valuable than the original paper.

The meeting then closed.

THE ESTIMATION OF CARBON MONOXIDE IN MINE GASES.

By E. H. Weiskopf, Ph.D., F.C.S. (Member).

Appendix.

Figures and data involved in analysis of gases given off by gelignite.

Determination of Amount of Sample taken.—The sample bottle after connecting with apparatus showed a temperature of 23·5° C. and 630.7 mill barometric reading; an internal pressure of 16·0 mill above the barometer. During the course of the analysis 12,209 gm. of mercury (S at 24·5° C. = 13·5350) equal to 902 c.c. had been introduced into the same bottle. At the end of the estimation the temperature had risen to 24·5° C., the barometric pressure remaining at 630·7 mill. The original volume of the sample bottle was decreased by 902 c.c., and therefore, provided that the remaining gas was brought to the same temperature and pressure conditions as existed originally in the sample

* Analysis by W. J. Orsman, F.I.C., in 1893: quoted in Sanford's "Nitro-Explosives." (Crosby, Lockwood & Son.)

bottle, this decrease corresponds to the volume of gas displaced and used. It therefore remained only, to bring the gas left in the bottle by means of the aspirator, to that pressure which at the changed temperature corresponds to the original condition. The formula is :—

$$P_1 = P\frac{273 + t_1}{273 + t}$$

In our determination, $P_1 = 648{\cdot}9$ mill; and the pressure inside the bottle was consequently adjusted to that figure. The volume of gas taken from the sample bottle, now corresponds to the volume of mercury introduced, *i.e.*, 902 c.c. at 23·5° C. and 646·7 mill. pressure, which equals 707 c.c. N.T.P.

In cases where the gas samples are in a moist condition, the influence of the vapour tension due to the change in temperature must be allowed for, by adding or subtracting this difference to or from, the results obtained by the above formula.

ESTIMATION OF CO BY TITRATION OF IODINE SOLUTION WITH THIOSULPHATE OF SODIUM.

A freshly standardised solution of sodium thiosulphate was prepared for each analysis, containing about 5 gm. of this salt per litre. For the standardisation, freshly sublimed iodine was taken, starch being used as indicator for this, as well as the actual estimations.

·1674 gm. iodine dissolved in potassium iodide solution made up to 250 c.c.

20 c.c. of this solution equal to ·013392 gm. iodine required 48·90 c.c.

Hypo solution (mean of 3 titrations).

The iodine solutions from the analysis required :—

Before blasting	...	3·80 c.c.
After blasting	...	71·80 c.c.

From the equation

$$I_2O_5 + 5CO = I_2 + 5CO_2$$

1 gm. of iodine is liberated by ·551833 gm. or 441·4663 c.c. of CO.

Therefore we obtained :—

Before blast : 3·80 c.c. = 0·45337 c.c. = ·028 vol. per cent. CO.

After blast : 71·80 c.c. = 8·6803 c.c. = 1·228 vol. per cent. CO.

GAS-VOLUMETRIC ESTIMATION IN MODIFIED ORSAT APPARATUS.

Gas taken	...	139·85 c.c. at 21·5° C. = 139·85 c.c.
After KOH absorption	...	131·85 c.c. „ 21·4° C. = 131·18 c.c.
After fuming H_2SO_4	...	131·12 c.c. „ 21·4° C. = 131·17 c.c.
With palladium tube affixed before burning	...	131·15 c.c. „ 21·4° C. = 131·20 c.c.
After burning	...	130·19 c.c. „ 21·5° C. = 130·19 c.c.
After absorption of CO_2 formed	...	128·38 c.c. „ 21·5° C. = 128·38 c.c.
After absorption of oxygen	...	104·23 c.c. „ 21·6° C. = 104·20 c.c.

Pressure constant all the time.

CO_2 = 139·85 c.c. − 131·18 c.c. = 6·2 vol. %

Olefines = —

$$CO \;\&\; H = 131{\cdot}19 - 128{\cdot}38 - \frac{2{\cdot}80}{3} = 1{\cdot}344 \text{ vol. \%}$$

CO	from iodine estimation	= 1·228 vol. %
H	therefore	= ·116 vol. %
O	= 128·38 − 104·20 + ·93	= 17·960 vol. %

Contributions and Correspondence.

SILVER ORE TREATMENT IN MEXICO.

(Additional note to contribution published in *Journal* of August, 1908.)

Some little time ago I received a request for a description of the preliminary treatment referred to in my contribution which appeared in the August, 1908, *Journal*, in discussion of Mr. Caldecott's paper on "Silver Ore Treatment in Mexico."

After making a large number of tests and applying every method that appeared to have any application whatever to the problem, I found only one means of unlocking the silver in the manganiferous ore so as to make it soluble in KCy solutions, and that was by preliminary chloridising. By treating with potassium chlorate and hydrochloric acid, or roasting with salt and sulphur, and following this with treatment by weak KCy solutions in the usual manner, a uniformly high rate of extraction was obtained. I desired however to hit upon a method that would be more feasible to develop on a practical working scale than either of these appeared to be, and it occurred to me to apply the common laboratory method of preparing chlorine, using salt, manganese dioxide and sulphuric acid according to the reaction :—

$$2NaCl + MnO_2 + 3H_2SO_4 = MnSO_4 + 2HNaSO_4 + 2H_2O + Cl_2.$$

The salt was mixed with the ore, crushed through 100 mesh screen. The sulphuric acid was added in weak solution (5% H_2SO_4) and allowed to stand with occasional agitation for 24 hours. The solution was then filtered off, the residues washed and treated with ·2% KCy solution. The silver extraction varied from 60

to 94%. The method has the advantage of low cost of materials; commercial sulphuric acid and salt are cheap, and the manganese dioxide is already present in the ore. I regret that I have not had the opportunity of going further and working out the practical application of the method used in these laboratory tests. The results were so satisfactory that I consider the method worthy of the attention of operators who have large enough bodies of refractory manganiferous silver ores to justify the expense connected with working out a new process.

ROBERT LINTON.

Los Angeles, Cal.,
25th January, 1909.

FURTHER NOTES ON THE UTILISATION OF WASTE HEAT IN SLIMES TREATMENT.

By A. SALKINSON (Member)

Since the publication of this paper in June, 1907, a number of mines have adopted the scheme with successful results; to those which are still considering the matter, the following notes may be of interest.

To begin with, the most suitable source of heat, though by no means the only one, is the mill engine; this has been very well utilised, amongst others, on the Knights Deep-Simmer East plant, where a solution heater has been erected between the exhaust of the mill engine and the condenser. The heater consists of a shell containing about 400 steel tubes, through which the solution is circulated from the sumps and so arranged that it has to pass four times through the heater before being returned. The plan adopted is to circulate the solutions alternately, so that only one heater is in use on each plant; every two months the tubes are cleaned, which is done without stopping the steam, the latter passing through the body of the heater to the condenser.

The principal advantage in utilising steam from the mill engine is that a large and constant source of heat is available; it also helps the engineering department, enabling the latter to obtain either a greater vacuum or to lessen the amount of water pumped through the condenser and cooling tower. Generally the principal objections met with are that the mill engine or other available source of heat is too far away from the slimes plant, and also that there is no waste heat. This latter astounding statement is not at all infrequent; however with the exception of one mine, which has electrical installation throughout, supplied with power from a distant source, it is of course groundless. Cyanide solution acts quite as well as a condenser of exhaust steam as water, and the results are far more satisfactory; but there is often a strong feeling against interfering with the existing plant.

As regards the distance from the available source of heat, this adds only a trifle to the cost of power, provided that the piping is not too small.

As will be seen from the figures below, the Simmer East has gained one-third in treatment capacity and one-fifth in extractor box capacity, as well as better precipitation. Nor must this be assumed as the limit, as the solutions could be still further heated, the end point being the temperature at which cyanide solution begins appreciably to decompose; solutions of a temperature of 130° to 140° F. show no signs of decomposition. A simple and cheap way of utilising exhaust steam from a non-condensing engine is to lead the steam directly into the solution sump. On the Wit Deep, a drum 4 ft. deep open at the top, and with an outlet just below the top has been attached to the side of the precipitated solution sump and the exhaust steam from a fan engine is led nearly to the bottom of the drum. The outlet pipes from the extractor boxes deliver into the drum and the solution overflows through the outlet, so that the steam has to pass through about 4 ft. of cold solution, which effectually condenses it. Hence the height of the solution in the sump is immaterial. Special attention may be drawn to the gain shown in the extractor boxes in the figures below, 20% more solution being precipitated per unit of zinc, and the percentage precipitation values having risen from 91 to 94·5%. Mr. Walford Dowling, reduction manager of the Knights Deep-Simmer East plant has informed me that since these figures were made up, the amount of solution precipitated has risen to a little over 2 tons per cub. ft. of zinc, or a gain of 30%.

It must not be forgotten that this would apply also to sands solutions, and a decrease of one-fifth to one third in extractor boxes and zinc is not to be despised. Another point of interest noticed on the Wit Deep is in connection with last drainings; after a sand charge has had its quota of weak solution, it is drained dry, and hot precipitated slimes solution is pumped on as a last wash. On comparing the last drainings, it was found that those from slimes solution were, contrary to expectations, almost invariably a little higher than from the cold weak solution. This can only be explained by the fact that hot solutions exert a quicker dissolving effect. It is hardly necessary to state that the original values of the two precipitated solutions were duly ascertained. On the Glencairn, heating of

sands solutions has been carried out for some months and, I am informed, resulted in an appreciable lowering of the residues. The Village Main Reef successfully used the process on portion of their plant during last winter for a mixture of accumulated and current slimes and obtained a considerable saving in settling time.

I am indebted for the following figures to Mr. J. W. Craig, manager of the Knights Deep and Simmer East, Ltd., and they are published by his kind permission. The gain in the collecting tanks is not shown, as the temperature of the mill water had already been raised before the installation in the treatment tanks. The distance of the solution sumps from the heater is about 600 ft.

SIMMER & JACK EAST, LTD.—SLIME TREATMENT, WITHOUT AND WITH HEATED SOLUTION.

		Without August, 1908.	With Sept. 9, to Oct. 8, 1908.
No. of vats in use	Collecting ...	4	4
	1st settlement	5	3
	2nd ,,	4	3
Total tons of slimes treated ...		9,900	10,620
Tons of slimes per day ...		320	354
Average depth of charge in collecting vat		3′ 5·08″	3′ 8·61″
Average tons per charge at 47·5% moisture...		165	179
Average temperature in Fah. degrees	Pulp during collection	70·3	73·7
	On completion of 1st transfer	58·9	86·4
	On completion of 2nd transfer	57·4	86·6
Average settling time, hours, 1st settlement		46·6	30·1
Average settling time, hours, 2nd settlement		44·6	28·2
Average time under treatment, hours		102·4	65·5
Average circuit in hours	Collecting ...	60	54
	1st settlement ...	75	41
	2nd ,, ...	60	41
Per cent. of charges	One settlement only ...	3·3	3·2
	Not completely settled		
	In 1st settlement ...	1·6	40·3
	In 2nd settlement ...	57·6	12·9
Estimated tons 2nd wash solution lost through incomplete settlement		787	158
Assay value of slime charges ...		1 486	1·501
Assay value of residues ...		0·237	0 245
Percentage extraction ...		84·0	83·7
Precipitation	Temperature entering extractors ...	57·3	77·3
	Tons solution per c.f.Zn. per 24 hours ...	1·555	1·89
	Assay value before ..	0·357	0·382
	Assay value after ...	0·032	0·021
	Percentage extraction ...	91·0	94·5

SIMMER & JACK EAST, LTD., SLIMES PLANT.

Notes on comparison of treatment without and with heated solutions :—

As no extra heat was added to the slime pulp at collection, the number of collecting vats remains the same in each case. The extraction in each case is practically the same; 720 tons more slime was treated with heated solution, in one day less, than was treated without heated solution, or 354 tons per day against 320.

Six vats did more work with heated solution than nine did without heat. This means 33·33% saving of plant without allowing for extra work done. Calculating the saving of plant from the settling time figures, the result is 36%, and on time under treatment figures also 36%. These latter figures are not quite reliable on account of the difficulty of keeping the solution low enough in decanting to see when the slime is settled. The more conservative figure had better be adopted. The incompletely settled charges for both cases were very near complete settlement, the difference averaging less than 3 in. per charge. The circuit hours calculation was based on a vat holding 200 tons of dry slime. The improved precipitation is very marked, more and better work being done with less zinc. Finally, it is safe to say that more heat would do quicker work.

Erratum.

ANALYSIS OF LEAD AND SILVER IN ORES.—Dr. J. Loevy writes pointing out a printer's error in the November *Journal* under the above heading.

Journal November, 1908, page 170, column 1, line 19, for "poor" read "pure."—Ed. Com.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

ANALYSIS OF ALUMINIUM POWDER.—Aluminium powder can readily be analysed by heating with an excess of ferric sulphate in an atmosphere of carbon dioxide, and determining the amount of ferrous salt formed by means of permanganate. The equation is $3Fe_2(SO_4)_3 + 2Al = Al_2(SO_4)_3 + 6FeSO_4$. In two analyses the author has found that nearly 6 % of oxide is present, but he cannot positively affirm that this oxide is aluminium oxide."—E. KOHN-ABREST, *Chemical News.—Forey Sciences*, January 15, 1909, p. 35. (E. H. C.)

THE CONTROL OF DISINFECTANTS.—In the *Lancet* of Sept. 19th, p. 902, Dr. S. Rideal and Mr. J. T. A. Walker draw attention to the necessity which exists for the examination of disinfectants after they have been allowed to stand after dilution for 24 hours. The subject is one of considerable importance to users of disinfectants, as frequently a disinfectant solution is made up and allowed to stand for several days before it is completely used, so if the dilution necessary is calculated upon a freshly made emulsion which afterwards separates and which in this way has a lower efficiency than that calculated, it is quite conceivable that considerable damage might be done. Some results which I recently obtained on a comparatively recent disinfectant with a guaranteed coefficient of 17 will be of interest. The sample was obtained from bulk which had been supplied to a large sanitary authority. The co-efficient on the freshly made emulsion was only 14, but when a 1 % emulsion had been allowed to stand for 24 hours the co-efficient had dropped to nine. This instance is only one of several which could be quoted, and I trust that those interested will adopt the suggestion of Dr. Rideal and Mr. Walker.—T. H. LLOYD, F.C.S.—*Lancet*, Oct. 17, 1908. (G. H. S.)

REPORT OF THE INTERNATIONAL COMMITTEE ON ATOMIC WEIGHTS, 1909.—The most important determinations which have been made during the year may be summarised thus:

Hydrogen. 1·00787. W. A. Noyes. By complete syntheses of water.

Chlorine. 35·457. Noyes and Weber. By syntheses of HCl; 35·468. Edgar. By syntheses of HCl.

Sulphur. 32·070. Baumé and Perrot. By determination of density of H_2S.

Lead. 207·19. Baxter and Wilson. From analyses of the chloride.

Tellurium. 127·609. Baker and Bennett. By heating TeO_2 with S; 127·54. Baker & Bennett. By direct conversion of Te to $TeBr_4$; 126·65—126·94. Marckwald. By careful dehydration of telluric acid.

Rhodium. 102·9. Hüttinger. Reduction of pentamine chloride with hydrogen.

Palladium. 106·708. Wœrule. Analysis of palladosamine chloride; 106·75. Haas. Reduction of palladosamine bromide; 106·434 Kemmerer. Reduction of palladosamine chloride in hydrogen.

Europium. 152·03. Jantsch. Analysis of octohydrated sulphate.

Erbium (Neo). 167·43. Hofmann and Burger. Synthesis of sulphate obtained from oxide by repeated fractionations.

Ytterbium. 170·6—174·02. Urbain - Welsbach. The old element has been proved to be a mixture of two elements, called neoytterbium and lutecium, of approximate atomic weights 170 and 174 by Urbain and Welsbach working independently.

Columbium (Niobium). 93·5. Balke and Smith.

Radium. 226·4. Mme. Curie and Thorpe. Analysis of $RaCl_2$.

In their report for 1908 this committee recognised the fact that a general revision of the atomic weight table was desirable, and such a revision has now been made. Modern investigations have shown that the fundamental values required modification, and through them many other atomic weights are affected, although the changes thus brought about are less important than they were generally supposed to be. Many atomic weights remain practically unaltered, and in few instances are the changes large, as a comparison of the new table with its predecessors will show. A careful scrutiny of all the evidence was, however, none the less necessary, and the table now offered gives the results thus obtained.

The fundamental atomic weights, the standards of reference employed in the calculations, are as follows, when O = 16.

H	...	1·008	Ag	...	107·880
C	...	12·000	K	...	39·095
N	...	14·007	S	...	32·070
Cl	...	35·460			
Br	...	79·916			

The value for silver is possibly a trifle too low, by from three to five units in the third decimal place. A combination of the best measurements gives Ag = 107·883. In this case, and in others as well, the second place of decimals is given in the table, the third place being uncertain. Thus we have K 39·10, N 14·01, Br 79·92, etc. Only with hydrogen is the third place retained.

	Symbol.	Atomic Weight.		Symbol.	Atomic Weight.
Aluminium	Al	27·1	Molybdenum	Mo	96·0
Antimony	Sb	120·2	Neodymium	Nd	144·3
Argon	A	39·9	Neon	Ne	20·0
Arsenic	As	75·0	Nickel	Ni	58·68
Barium	Ba	137·37	Nitrogen	N	14·01
Bismuth	Bi	208·0	Osmium	Os	190·9
Boron	B	11·0	Oxygen	O	16·00
Bromine	Br	79·92	Palladium	Pd	106·7
Cadmium	Cd	112·40	Phosphorus	P	31·0
Caesium	Cs	132·81	Platinum	Pt	195·0
Calcium	Ca	40·09	Potassium	K	39·10
Carbon	C	12·00	Praseodymium	Pr	140·6
Cerium	Ce	140·25	Radium	Ra	226·4
Chlorine	Cl	35·46	Rhodium	Rh	102·9
Chromium	Cr	52·1	Rubidium	Rb	85·45
Cobalt	Co	58·97	Ruthenium	Ru	101·7
Columbium	Cb	93·5	Samarium	Sa	150·4
Copper	Cu	63·57	Scandium	Sc	44·1
Dysprosium	Dy	162·5	Selenium	Se	79·2
Erbium	Er	167·4	Silicon	Si	28·3
Europium	Eu	152·0	Silver	Ag	107·88
Fluorine	F	19·0	Sodium	Na	23·00
Gadolinium	Gd	157·3	Strontium	Sr	87·62
Gallium	Ga	69·9	Sulphur	S	32·07
Germanium	Ge	72·5	Tantalum	Ta	181·0
Glucinum	Gl.	9·1	Tellurium	Te	127·5
Gold	Au	197·2	Terbium	Tb	159·2
Helium	He	4·0	Thallium	Tl	204·0
Hydrogen	H	1·008	Thorium	Th	232·42
Indium	In	114·8	Thulium	Tm	168·5
Iodine	I	126·92	Tin	Sn	119·0
Iridium	Ir	193·1	Titanium	Ti	48·1
Iron	Fe	55·85	Tungsten	W	184·0
Krypton	Kr	81·8	Uranium	U	238·5
Lanthanum	La	139·0	Vanadium	V	51·2
Lead	Pb	207·10	Xenon	Xe	128·0
Lithium	Li	7·00	Ytterbium		
Lutecium	Lu	174·0	(Neoytterbium)	Yb	172·0
Magnesium	Mg	24·32	Yttrium	Y	89·0
Manganese	Mn	54·93	Zinc	Zn	65·7
Mercury	Hg	200·0	Zirconium	Zr	90·6

In adjusting the other atomic weights the determinations by Richards and his colleagues have generally been given preference. They are certainly entitled to the highest weight, but probably not to exclusive consideration. The work of Guye and his associates at Geneva, and the recent direct measurements of the chlorine-hydrogen ratio are also of very great importance. It is to work of this order that we must look for ultimate precision. Important investi-

gations upon atomic weights are now being carried on in several laboratories, and our knowledge of these constants will doubtless become much more exact within the near future."

F. W. CLARKE, W. OSTWALD, T. E. THORPE, G. URBAIN.—*Journal of the American Chemical Society*, Jan., 1909, xxxi, p. 1. (J. A. W.)

TEST FOR PURITY OF MINE AIR.—"Dr. Angus Smith, in a report to Lord Kinnaird's commission, gives it as his opinion that the quantity of CO_2 in the air of mines should never exceed ¼, or ·25%.

Dr. Smith gives a very simple test by which determination can readily be made to determine whether mine air contains as much, or more, or less quantity of CO_2 than ¼%.

The necessary apparatus consists of: One 8-oz. bottle and cork; one 5-oz. bottle and cork; one bottle of limewater with excess of lime; one pipette or measure holding ½ oz.; four ½-oz. bottles corked; one ¼-oz. bottle, containing an alcoholic solution of phenolphthalein; one piece of india-rubber about 1 ft. long.

The method of preparing for and making the test is as follows: Measure ½ oz. of clear limewater and pour into a 5-oz. bottle, and fill up with distilled water, giving a solution of 10% strength. Add a drop or two of the phenolphthalein solution; the limewater assumes a pink colour. Now fill the 8-oz. bottle with mine air from a sample flask which has been obtained from the mine, add ½ oz. of the coloured solution and shake. If the pink colour disappears quickly, the air contains more than ¼% of CO_2. If the colour fades only after repeated re-shaking, the quantity of CO_2 in the air is approximately ¼%."—*The Chemical Engineer*, Dec. 1908, p. 269. (W. A. C.)

METHODS OF TESTING THE EFFICIENCY OF VENTILATION.—"A general consensus of opinion among recent investigators points to recorded temperature as the most practicable means of controlling ventilation under modern conditions. The temperature of any given room may increase by any one or any combination of the following causes: (1) Heat given off by the bodies of the occupants and by their breath; (2) Heat given off by the lights; (3) Rise of outside temperature; and (4) An excess of heat from the heating plant.

The last of these causes should be practically eliminated in a modern, well-constructed, and well-managed public structure. In all such places the heating system is, or should be, designed to deliver a constant amount of heat for any given period of time.

Contrary to the old theory that a heating plant should "keep a room at a given temperature," and so automatically supply less and less heat as the heat derived from the occupants increases, the new idea is rather that the plant deliver a constant predetermined amount and that the ventilating plant keep the room at the desired temperature. This idea is really carried out in most of the buildings with which the ventilating engineer or inspector would have to deal. This being the case, any rise of temperature in a room would be caused by the animal heat of the occupants, effect of the lights, or by rise of the outside temperature. This last cause is slight in any instance and may usually be neglected.

Then, since in giving off heat the body and the lights (with the exception of electric incandescent) give off both CO_2 and moisture in proportion, we have in this increase in temperature an almost ideal index of the quality of the air. A recording thermometer with a large dial face, hung or placed so as to be as readily seen as a clock, is the best known gauge of an occupied room.

If the air current through an occupied space is sufficient to carry away the increased moisture and to keep the temperature even, it will lower the carbon dioxide in the same proportion This may be accepted as true without chemical examination (there are occasionally other sources of CO_2). The humidity is therefore an excellent indicator of the degree of ventilation, and a recording hygrometer is most desirable to place alongside the recording thermometer.

We feel safe in stating that probably the best engineering practice in ventilation to-day is in the use of these instruments, combined with training in the detection of odors. Only in cases of control or inspection will the determination of CO_2 be necessary when these instruments are at hand and when time for observing them is available. But there are frequent circumstances in which CO_2 is an equally good indicator and the tests for it more convenient of application, but they should partake of the rapidity and comparative accuracy of the instruments, and should not require the engineer to be burdened with cumbersome apparatus. For these purposes a sufficient degree of accuracy may be defined as within two-tenths to three-tenths of a part per 10,000, with a possible maximum error of five-tenths

Interpretation of Results.—If in a crowded hall the temperature is less than 70° F., if the wet and dry bulb thermometers are 7° apart (less than 70 per cent. humidity), with no appreciable odour, and if the carbon dioxide is less than seven parts per 10,000, the air may be said to be satisfactory; but if the temperature is 80°, the wet and dry bulb only 4° apart (humidity 83%), and the carbon dioxide eight parts, with a close odour, then measures should be taken at once to remedy the condition.

The ideal condition for constant occupancy is: thermometer 68°, humidity 68%, or wet and dry bulb 6° to 7° apart, with carbon dioxide less than six parts per 10,000 of air.

If these general results may be obtained, the smaller differences in readings may be discarded until the public is educated to this extent."—RICHARDS, WADE, GILBERT, HANSON and TALBOT.—*Technology Quarterly*, Sept., 1908, p. 321-331. (E. H. C.)

HARD CHEMICAL PRODUCTS.—"The diamond, so long the hardest known substance, now has two rivals, the silicide and the boride of titanium—products of Prof. Henri Moissan's electric furnace—each being claimed to be as hard as the diamond."—*Mining World*, London.—*The New Zealand Mines Record*, Dec. 16, 1908, p. 221. (A. R.)

ZINC IN ZINC DUST.—"In determining the amount of Zn in zinc dust by the volume of H evolved on treating with acid, it has been found that the use of rubber connections in the apparatus must be avoided, otherwise the results are too low. The phenomenon seems to be due to diffusion."—DE KONINCK, *Bull. Soc. Chim. Belge*, xxii., 113. —*School of Mines Quarterly*, Nov. 1908, p. 52. (H. A. W.)

COLORIMETRIC DETERMINATION OF DISSOLVED OXYGEN IN WATER.—This method depends on the change in the colour of cuprous ammonium chloride

when brought into contact with oxygen. The salt is difficult to prepare quite colourless, and hence it was advisable to construct an apparatus in which a solution of the pure colourless salt could be prepared and kept at all times ready for use. This was finally done in an apparatus where the rubber connectors were protected from air by an atmosphere of hydrogen, but even then the reagent could not be kept colourless for any length of time and finally an apparatus made entirely of glass was used. As soon as the water containing oxygen was brought into contact with the reagent the blue colour appeared and the comparison was made with a standard solution of cupric chloride of such strength that 1 c.c. of the solution was equivalent to 1 c.c. of oxygen in a liter of water when the quantity of of water taken for analysis was 100 c.c. This was prepared by dissolving 1·1364 grams of pure copper in aqua regia, evaporating off the excess of acid, dissolving in water and making up to 1 liter. 1 c.c. of this solution is equivalent to 0·1 c.c. of oxygen as indicated thus

$$Cu_2Cl_2 + 2HCl + O = 2\,Cu\,Cl_2 + H_2O.$$

The apparatus requires some skill to use it but when the necessary experience has been obtained accurate determinations of oxygen may be easily and rapidly made.—G. B. FRANKFORTER, G. W. WALKER and A. D. WILHOIT.—*Journal of the American Chemical Society*, January, 1909, xxxi. 35. (J. A. W.)

COBALT TIN ALLOYS.—A thermal and micrographic examination of alloys of cobalt and tin showed that these metals form two compounds, CoSn and Co_2Sn, both of which are non-magnetic and are considerably harder than pure cobalt. The compound, Co_2Sn, melts at 1171° C, but the compound, CoSn, decomposes at 927° C. before it melts. —S. SHEMTSCHUSHNY and S. BIELYNSKI, *Z. anorg. Chem.*, 1908,*59*, 364—370. *Chem. Zentr.*, 1908, *2*, 1498.—*Journal of the Society of Chemical Industry*, Dec. 15, 1908.

METALLURGY.

CLASSIFICATION FOR TUBE-MILLING.—"The common practice on these fields for separation from the tailings pulp of coarse sand for regrinding has not varied from the form of hydraulic classification supported by the writer some years ago, in a paper contributed to the proceedings of another technical Society.* Other methods, which have been advocated, consist in separating the coarser grains for regrinding by means of a screen, as, for instance, of the Callow endless travelling belt type, or by Wilfley tables as on two mines on the Eastern Rand, or by continuously removing the heavier product settling in a shallow box by means of an inclined scraper conveyor.† This last method has the advantage of not requiring fall, and appears suitable for small installations; it is used in the United States in the form of the Dorr classifier,‡ which is fitted with reciprocating rakes.

As is well known, classification prior to tube-milling is here commonly effected by leading the underflow from several pyramidal or conical spitzlutten, by means of a launder, into a dewatering spitzkasten placed above and near the tube-mill inlet, which receives the finally thickened pulp underflow. This system is fairly satisfactory, but has the objection of requiring considerable elevation of pulp and a certain consumption of mill service water to assist clearance of the spitzlutte, whilst there are for each tube-mill a number of classifiers requiring adjustment, and each liable to clogging of the comparatively small nozzle outlet; further, constant attention is required to ensure a regular feed to the tube-mill of a pulp containing the desired percentage of moisture.

To overcome these difficulties there has lately been developed on the Simmer and Jack plant a method which is now in regular operation, and which offers considerable advantages. Briefly, it consists in combining the spitzlutten and dewaterer in one classifier, which receives the pulp direct from the tailings launder, and delivers its coarse sand underflow into the tube-mill. This classifier is of the conical type, with peripheral launder for receiving the final pulp overflowing to the cyanide plant. In its operation, a sufficient quantity of pulp, including return from tube-mills, after elevation is delivered from the main tailings launder by means of a by-pass and vertical inflow pipe, fitted with a horizontal baffle, slightly below the surface of the pulp in the centre of the classifier. Whilst smaller dimensions can be employed, the classifier at present in use is 6 ft. diameter at top and 9 ft. deep. This classifier is kept nearly filled to the top with sand, and an essential patented feature in its satisfactory operation consists of an internal serrated or notched horizontal disc diaphragm near the bottom. This ensures a steady underflow of thick pulp, containing very little moisture, which can be regulated in amount up to 440 tons of solids per 24 hours. An illustration given shows a classifier, with one of the various forms of diaphragm. Owing to the thick consistency and slow velocity of the pulp, a large underflow opening (2⅜ inches in diameter) is used, which wears but little and can be exactly regulated by an adjustable horizontal cut-off gate. To thin down the pulp for satisfactory tube-milling, turbid water is withdrawn from the upper portion of the tailings stream and mixed with the underflow entering the tube-mill, whilst a further supply from the same source is employed to further thin down the tube-mill outflow prior to amalgamation on shaking tables. By regulating the size of underflow opening, the tonnage of solids issuing can be altered without changing its percentage of moisture, which last is regulated for tube-milling purposes by the independent supply of turbid water. Should the volume of tailings pulp vary, the classifier automatically adjusts itself to the altered conditions, by the increased or decreased volume of pulp flowing upward to the surface of the classifier, sorting out and carrying away in the overflow larger or smaller grains corresponding to its changed velocity. The diaphragm, by supporting a considerable depth of settled sand, prevents any change in consistency or breaking away of the thick, sluggish underflow, whilst the large size of the opening precludes choking In practice on the Simmer and Jack about 440 tons of solids containing 65% + 60 (0·01 in.) grade are delivered per 24 hours from the classifier into the tube-mill as an underflow pulp containing 26% to 28% of moisture. The classifier overflow of final pulp at present contains about 10% + 60 (0·01 in.) grade, but of course varies with tonnage of ore milled.

The advantages of the system described are:—(*a*) Much less elevation of pulp for the tube-mill circuit so that the installation of an elevating

* See *Trans. Inst. Min. and Met.*, p. 55, vol. xiv., 1904.
† See MacDermott and Duffield's "Amalgamation of Gold Ores," p. 32, 1908.
‡ See *Mines and Minerals* June 1908, p. 541.

system to this height only is contemplated, with subsequent elevation of the final pulp alone to the necessary height for delivery to the cyanide plant; (*b*) considerable economy in initial cost of construction, owing to several classifiers with fittings and accessory launders being replaced by a single classifier for each tube-mill; (*c*) simplicity of operation of the single classifier in place of several; (*d*) the dispensing with a supply of mill service water and its pumping cost; and (*e*) non-liability to derangement by choking, owing to large size of underflow outlet.

The use of deep conical hydraulic classifiers with diaphragms had been originally developed in connection with pulp classification for another purpose, but their successful application to tube-milling practice is largely due to observant attention and skilful modification by Mr. G. O. Smart, Reduction Works Manager of the Simmer and Jack Proprietary Mines, Ltd."—W. A. CALDECOTT.—*Journal of South African Association of Engineers*, Dec., 1908, p. 101. (W. R. D.)

THE CALLOW SCREEN.—"In most concentration mills the sizing or classification of ore in some manner is essential. The general custom has been to use either revolving or shaking screens for the coarser sizes, and water classification for the finer particles. With the ordinary revolving screen, or trommel, the limit of practical economical work is about 2·5 mm., or, say, one-tenth of an inch screen opening; finer than this, water classifiers are used. But water classifiers are generally admitted as a very poor substitute for screens, since they are quite different in effect, giving grades of equal falling particles, which is an entirely different thing to grades of equal-sized particles, and as a consequence water classification is not a proper preparation for concentration, while it is, on the other hand, most objectionable as introducing great excess of water into the slimes. Many and varied attempts have been made to invent a practical method of fine screening wet, and the failure so far resulting has been due to excessive wear, imperfect screening, clogging of mesh, or small capacity, with undue water consumption. The Callow travelling belt screen, it is claimed, is free from the objections named, and is one of those simple inventions which appeal to the practical man at first sight. The repeated attempts to make a practical fine wet screen are clear confirmatory evidence of the recognised large field of utility.

Water classification tends usually to make the fine slimes poorer than a similar grading (if possible) by screens would do, but it never makes the slimes poor enough for throwing to waste, and it increases the difficulty of their treatment by the introduction of a great volume of extra water. On the other hand, it is possible, on certain ores, to use Callow screens so fine that what passes through them is actually an enriched product, owing to the percentage of brittle mineral broken down fine in the crushing. For example: a copper slime which from water classifiers would assay 3% of metal, may by its production from a screen of 150 mesh assay 7½%, and thus become a smelting product added to the other mill concentrates, and therefore not subject to the great loss which all millmen know will follow the concentration of the 3% slime.

In the grading of material for recrushing—as for instance for tube mills following stamps—screening in certain cases offers great advantages over water classification; because in water the particles of heavy mineral which settle are much smaller than the particles of gangue rock, and thus much of the mineral already fine enough is subjected to regrinding, and it may be unnecessary, or even objectionable, in subsequent treatment, that the heavy mineral should be reduced much finer than the gangue accompanying.

The fundamental principle of this machine is a travelling band or belt of screen cloth, over which the ore and its carrying water is spread by means of distributing aprons. The band or belt of screen cloth passes over head and tail rollers, and the belt is caused to travel continuously in a horizontal direction at a speed varying from 25 to 125 ft. per minute, according to the nature and quality of the material to be screened. These aprons play a most important part in the operation of the machine, and its successful and proper operation depends in a great measure upon this feature and its proper construction and adjustment. It is with this that a preliminary sorting or sizing is done, for as the particles which are of all sorts and sizes fall from the lip, the coarsest and largest pieces, having the greatest trajectory, strike the screen at a point far ahead of the smaller ones, and they are there deposited, leaving a space behind free and unincumbered. The pores of the cloth in the rear being, therefore, still uncovered and open, permit the free passage of the fines and water. Simultaneously with this action the cloth is being moved forward and the deposit of oversize is continuously and unintermittently removed from the separating or screening zone, and thus the machine continues to perform its function so long as it is kept moving and is supplied with feed.

After leaving the screening zone, the deposit of oversize is carried forward and passes under a shaking spray of clear water, where any remaining traces of slime or of fine adhering particles are washed, and pass through with the water from the spray into the undersize hopper beneath. Continuing on, the oversize still clinging to the screen cloth, passes in front of a small impinging spray, which is conveniently situated somewhere about the mid-diameter of the front roller, and the oversize is there washed off the screen cloth into the oversize hopper below. The 24 in. Duplex machine is in two parts, each half being the duplicate of the other; the two belts are independent of one another, but operate from a common driving mechanism. The speed of the two belts can be varied by means of cone pulleys, and the driving shafts are arranged so that either side can be thrown in or out by suitable friction clutches adjusted to each driving roller. This permits of one side of the machine being shut down for changing or the repair of the screen cloth, all the feed being turned on the running side pro tem., it only being necessary to speed up the belt sufficiently to carry the increased load.

The rollers revolve on overhung shafts, so there are no out-board bearings to prevent the quick renewal of worn-out cloths, the new ones being slipped on (after being made endless) over the ends of the rollers. The screen belts are inserted on each side, between the plies of a 2-in. two-ply rubber belt; these rubber edges serve as a belt to drive the screens, and the flanged edge serves to confine the material to be screened to the screen belts, and prevents unscreened material escaping over into the undersize hoppers.

The shafts which carry the rollers are carried in long hubs or bosses attached to each side of a central casting or girder; to this casting are also attached

the undersize hoppers. This casting is hollow, and forms, besides a support for the moving and all other parts of the machine, a central gutter or launder into which the combined undersize from each half of the machine discharges."—*Transvaal Leader*, Feb. 2, 1909. (A. R.)

NON-EXPANSIVE ALLOY.—"A new alloy which has been brought out in France, and is know as 'Invar,' has the valuable property of having practically no expansion, and is thus adapted for various purposes. M. Guillaume succeeded in forming such an alloy in connection with the Commentry-Decazenville metallurgical firm. Nickel steel is the base of the alloy, and up to 20% of nickel, the expansion is about the ordinary. It then increases up to a value of 24%, reaching a minimum value at 36% of nickel, then rising and taking the normal value for a 50% alloy. It is the 36% metal which has the curious properties brought out by M. Guillaume, and its expansion is almost zero, being 17 times less than that of steel and about the same as for melted quartz. The new steel is not magnetic, and it does not rust. Owing to these properties, it is already used at Paris and elsewhere in the construction of spiral springs, pendulums, gradnated scales for instruments. Its field of usefulness is extensive, and it can receive many applications where these properties are needed. One valuable application is in the formation of standard measures, such as the metre, and different standard scales have been made at Paris by the International Bureau of Weights and Measures. Such scales or gauges are remarkably exact, owing to the non-expansion of the metal. They are used in the artillery works at present, and very precise work can be carried out with the gauges and calipers, diminishing the errors which were found previous to their use. According to the tests made upon standard scales, these hold their value within a remarkably small percentage. By varying the amount of nickel in the combination, a second alloy can be formed which is valuable as having the same coefficient of expansion as glass, and it can thus be used to replace platinum in the leading-in wires of an incandescent lamp, thus reducing the cost of these wires to a comparatively low figure. It can also be used to replace the platinum wires of eudiometers and Geissler tubes for the same reason."—*Scientific American Supplement.—Mining Science*, Dec. 31, 1908, p. 523. (A. R.)

DANGERS OF WHITE ARSENIC.—"At a meeting of the Cape Chemical Society, Professor B. de St. J. van der Riet, of the Victoria College, moved that in the opinion of the Society the present law regulating the sale of poisons should be so amended as to restrict the sale of uncoloured arsenical preparations to the public. There was, he said, a law regulating the sale of poisons, but it did not seem to prevent white arsenic or arsenical preparations coming into the hands of ignorant and careless persons. At present, also, criminals obtained pounds of white arsenic with the greatest ease, and people were in the habit of leaving the substance lying about houses in the most careless manner imaginable. He thought that regrettable accidents and crime could be prevented by the Colony adopting the English law on the subject, which compelled the colouring of all arsenical preparations sold to the public, so as to take away their unsuspicious appearance.

Dr. C. F. Juritz stated the English law compelled the addition to arsenic and arsenical preparations of at least six per cent. of soot or three per cent. of indigo.

The motion was unanimously agreed to, and it was further decided that the Council of the Society should communicate the resolution to the Government."—*Rand Daily Mail.* (G. H. S.)

TOOL STEEL.—"It was, of course, well known that to subject ordinary carbon tool steel to such temperatures as 2,000° F., would mean its absolute ruination, and it was, therefore, evident that carbon, the most important constituent in ordinary tool steel, was only of secondary consideration in self-hardening steel, and that it was by virtue of some other special constituent that the latter could retain its hardness at high temperatures.

Further experiments on these lines soon made it clear that it was the proportion of tungsten, and in a lesser degree, that of chromium, that makes the difference between the ordinary self-hardening steel and high-speed steel. By working on this knowledge, and making many patient and costly experiments, steel manufacturers have now placed on the market a high-speed tool steel which gives results that are nothing short of marvellous, when compared with the results obtained from ordinary carbon and self-hardening tool steels before the introduction of their formidable rival.

The following analyses show clearly the difference in constitution of the three different classes of tool steel that have been referred to :—

	Ordinary Carbon Tool Steel. %	Self-Hardening Tool Steel. %	High-Speed Steel. %
Carbon ...	1·04	2·4	·67
Tungsten ...	—	5·62	18 90
Chromium ...	—	·49	5·47
Manganese ...	·25	1·90	·11
Silicon ...	·145	·71	·034
Sulphur ...	·012	·031	·029
Phosphorus ...	·022	·035	·013

A great drawback to the high-speed steel of the early days was the cumbersome method of hardening the tools made of it. To obtain the best results it was necessary to heat the tool to a white heat and quickly reduce it to a dull red, which was accomplished by plunging the nose of the white-hot tool into a vessel containing molten lead. This, coupled with the difficulty of persuading tool smiths of the necessity of bringing the tool to a white heat in the first instance, led to the system of returning tools to the makers when re-forging and re-hardening became necessary. Such difficulties are, happily, no longer a bar to development of the use of high-speed steel. Improvement in manufacture has rendered the handling of a high-speed tool a comparatively simple process, and time has overcome the very natural reluctance of the tool smith to abandon that experience which, prior to the advent of high-speed steel, had taught him that tool steel heated beyond a bright red was absolutely ruined."—R. E. L. MAUNSELL.—*Page's Weekly*, Dec. 18, 1908, p. 1243. (A. R.)

THE ABSORPTION OF WATER BY COKE.—"A series of researches on the power of red-hot coke to absorb water are detailed in the issue of *Stahl und Eisen* of January 6, and the results are set forth in graphic diagrams. The amount of the water again given off at various intervals of time is also shown. It would appear from the results of these tests, which were conducted with small quantities of coke (1 kg.), that red-hot coke plunged into cold water will absorb from three to four times the volume of water as compared with the quantity which it will take

up, if it is placed in hot water when it is cold. The results of ten experiments are also recorded, in which freshly drawn red-hot coke, in quantities of about 100 kg. at a time, was immersed in cold water and allowed to remain therein for half an hour. It was then withdrawn and placed in casks with a perforated bottom, out of which the water was allowed to drip for an hour. Subsequently the weight of the coke was ascertained at intervals of 24, 48 and 72 hours—that is to say, the gradual evolution of water was ascertained. Ultimately the whole of the coke was stamped to powder and carefully dried, so as to ascertain the weight of the coke alone. The test proves that slightly damped coke will give off water in a railway wagon at the same rate as that which has been more drenched. Samples, both of foundry coke and of blast furnace coke, were tested."—*Times Engineering Supplement*, Jan. 13, 1909. (J. A. W.)

SPARKS FROM ALLOYS.—"Auer von Welsbach has recently discovered that certain alloys of iron with cerium, lanthanum and other rare metals obtained from monazite, emit brilliant sparks when scratched with a knife or file. When strongly pressed with a piece of steel the alloy gives forth a brilliant flame that is accompanied by little heat or smoke. The sparks readily ignite gases, tinder, alcohol wicks, and explosives, such as are used in blasting and gunnery. The flame might also be made serviceable in flashlight photography, dispensing with the disagreeable odor of magnesium powders, and another possible use suggested is for military signalling."—*Mining Science*, Dec. 10, 1908, p. 463. (A. R.)

MINING.

MOUNT MORGAN INQUIRY REPORT.—"The report of the special board appointed to inquire into the disasters in the Mount Morgan mine on September 5 and in November was tabled in the Legislative Assembly by the Minister for Mines recently. The board makes the following recommendations :—

(1) The abandonment in the copper chambers of the pigstye system of timbering, as carried out in the past.

(2) The adoption of a sectional system of mining in all such parts of the mines as are already known, or may reasonably be suspected to be intersected by dykes or 'heads.' It was added that the system proposed by the mine manager (Mr. James White) appeared to be such as would meet the requirements in parts of the mine adapted to its use, it being a system that would lend itself more generally to the mining of ore bodies, such as are at present exposed in the copper chambers; and inasmuch as by this method a greater extent of ore could be worked at one time.

(3) Whilst recognising that no one system could be recommended for working the whole of the ore bodies, the board advised the careful selection of suitable methods of working different parts, with a view to securing a maximum of safety, a matter which, owing to the sudden variations from a hard compact ore to that of a treacherous nature, must of necessity be left to the judgment of the mining engineer.

(4) The exercise of the greatest precaution in the inauguration of any method of mining adopted in the future, as they recognised that the process of changing from one system to another would take some time, and would be attended with considerable risk.

(5) That in all work done in treacherous ground in the future reliance should be placed on close filling rather than on timber as a support. The filling available was of very good quality, and thoroughly suitable for this purpose.

(6) That more attention should be paid to the mullocking and filling of depleted stopes, and that a rigid course of filling those at present open should be initiated from and including the sill floor; this system of filling should be carried on concurrently with the work and kept up to within a workable distance of the face.

(7) That using wedges for blocking timber to the back, walls, or face, more attention should be paid to keeping the wedges tight; in the past there appeared to have been some misconception as to the use of wedges for taking up shrinkage and carrying weight, as distinguished from their use as a means of showing pressure.

(8) The Mining Act of 1898 did not make compulsory the reports of falls of ground unless these were attended with serious injury to any person. The reporting of all falls and movements of ground was necessary, and could be provided for by amending legislation upon the lines laid down in the evidence of Mr. Inspector Ward, a course which the board recommended.

(9) The board was satisfied that the Inspector of Mines had been attentive to his duties, and carried them out satisfactorily. They did not consider necessary the appointment of a resident inspector of mines at Mount Morgan.

(10) The replies to certain questions pointed to the fact that the duties previously carried out by three officers have of late devolved upon the registered mine manager, and they recommended that the attention of the company be called to the matter.

(11) That the workings of any flat-backed stope with two faces advancing towards each other be discontinued, unless the intervening ground is mullocked close up to the back.

(12) That in adjoining sections of the mine, where square set work is adopted, the timber should be laid out in its relatively correct position, so that it may ultimately be connected, and may thus develop its full strength.

(13) The board recommended a continuance of the use of ironbark, spotted gum, or other approved hardwood timber for square sets. As regards brigalow timber, in spite of the objections of a number of witnesses the board took no exception to its use except for the purposes for which it has been used in the mine. In conclusion, the board acknowledged the assistance rendered to them throughout the course of the inquiry by the management, shift bosses, and miners."—*Australian Mining Standard*, Dec. 30, 1908, p. 747. (A. R.)

AMOUNT OF COAL DUST IN AIRWAYS.—"In a paper read before the Institution of Mining Engineers, in Edinburgh, Henry M. Hall, H.M. Inspector of Mines, gave an account of the various investigations in regard to the explosive properties of coal dust and the findings of the Commission on Explosives on Coal Dust in Mines. Continuing, he said :

Before any useful suggestions could be made as to the treatment of coal dust, so as to render it innocuous, it was absolutely necessary to obtain some idea as to the rate at which dust is deposited on underground roadways. Having gained this infomation, at any rate, approximately, it then becomes possible to judge fairly accurately of the practicability of any suggested remedy.

The writer believes that hitherto nothing has been done to obtain this information. Dusty roads, even

to the extent of coverings several inches thick, are familiar to all mining engineers, but over what period of time these coverings have been accumulating no evidence is forthcoming; in some cases it may have taken only days, in others months, or even years. Aided by the willing help of others, the writer has set himself the task of solving this question, at any rate so far as to obtain a working idea of the rate of the accumulations in distinct parts of the mine and under ordinary mining conditions. With this object, aspirators were fixed in a return air road at intervals of about 75 ft.

The action of the aspirator is by means of cotton-wool; the air is drawn through it and the dust deposited in transit, the cotton-wool being carefully weighed before and after the operation. A bottle on a stool is filled with a measured quantity of water, which is then siphoned out into a bottle on the floor, thus allowing the air to pass through a tube which contains the cotton-wool, and deposit the dust.

As a result of these experiments, it was shown that something like 11½ lb. of coal dust were carried in eight hours past the point where the measurement was taken. The velocity of the current of air carrying this dust was 95 cub. ft. per minute. The measurements for dust in the air were made with a bellows aspirator, and it was found that in 3 cub. ft. of air the amount was ·002, ·001, ·002, and ·0005 gm., depending on the point of measurement.

These results show that there was gradual diminution of the dust as it travelled outbye.

A third method of measurement was adopted, with the object of arriving at the quantity of dust deposits on the floor, sides, etc., as it was being carried along by the ventilating current. Flat porcelain dishes of an area of 1·12 sq. ft. were used, two of which were placed at points on the main intake and two on the main return. These were in operation for periods varying from 8 to 10 hours on two separate days. The dust passing the two points on the main intake must have come partly from the screens and partly from the tubs passing up and down the shaft (about 600 tons of coal passed up the shaft each day).

The results show that at the shaft bottom, where the area of the road is 150 sq. ft., something like 11·47 gm. of dust were passing in the air each minute when the velocity was 225 ft., that is, 18 lb. in 6 hours. So that at least this quantity is being deposited each working day on to the floor and ledges of the main road between the bottom of the shaft and the bottom of the tunnels, 3,000 ft. inbye. If it be assumed that this road averages 12 ft. in width and that ½ lb. of dust per linear foot of such a road is required to make an explosive mixture of dust and air, starting with a clean sheet, it would require 83 ordinary working days to render this 3,000 ft. of road absolutely dangerous. (Note —The deposit, of course, will be thicker near the shaft than farther in.)

An interesting experiment was made at Garswood Hall colliery. The screens at this colliery are near the downcast shaft, and a considerable quantity of dust is made in the process of screening, some of which finds its way back into the mine along with the ventilation. To obviate this, water sprays have been fitted up at the bottom of the shaft, and a frame has also been put in the main airway a little distance from the shaft; arranged around this frame are several sprays, the object of which is to wash the dust from the air as it passes through the frame. Quite a considerable amount of dust is thus washed out. With 40,000 cub. ft. of air passing through, 7½ lb. (weight after drying) of dry dust was recovered by means of a settling tank in 14 hours, and probably as much more escaped collection, which means that the sprays washed down 14 or 15 lb. in the 14 hours. This dust would originate partly at the screens and partly be blown from the trams in the pit shaft. The total quantity of dust finding its way back on the road-ways must be very large, when with only one frame so much can be recovered.

These experiments, made with a view of getting an idea of the rate of accumulation of dust in mines under ordinary working conditions, should be carried much further; but, so far as they go, it seems clear that the common estimate of the quantity of dust deposited day by day far exceeds the actual facts; and it follows that when once the roads have been made thoroughly clean, to keep them so ought not to present any insuperable difficulty. Allowing dust to get back into the workings from the shafts and screens amounts to poisoning the ventilation before it starts on its duty."—HENRY M. HALL.—*Mines and Minerals*, Dec., 1908, p. 223. (A. R.)

EUCALYPTUS FOR MINE TIMBERS.—"The woods of the eucalyptus are heavy, only a few species floating in water. According to McClatchie, there are 22 species cultivated in the United States that are suitable for railroad ties and underground work. Selection of species with reference to use as timber must be considered in relation to soil and climatic requirements.

To the average person, the term 'eucalyptus' has meant nothing but the most widely planted species, *Eucalyptus globulus*, or 'blue gum.' In Australia several species are known as 'blue gums' and several as 'red gums'; and the same species may be designated as by several different common names. Australian common names that have been generally adopted in America are 'blue gum tree' for *Eucalyptus globulus*, and 'red gum tree' for *Eucalyptus rostrata* and several other species indiscriminately. The entire genus *Eucalyptus* should not be condemned because the wood of one species, according to Mr. T. L. Carter, has been found inferior to other timbers for mining purposes. The principal drawback to eucalyptus timber has been its tendency to check, or open fissures and shakes, and its hardness, with consequent difficulty in spiking and working. The better understanding of methods for seasoning and handling this timber is doing much for its advancement. Mr. Carter has considered only *Eucalyptus globulus*, and has hardly given that widely known species its due. Its timber is hard, heavy, strong, and fairly durable. In America, however, it is not considered equal in timber properties to the ironbarks, to the sugar gum, or to the red gums. Its fast-growing characteristic has been its chief claim for prominence, and has led to planting in different parts of the Southwest for windbreaks and fuel,

For mine-timbering there are other species that are superior to the blue gum. There are about 17 species of eucalyptus that could be grown by mining companies and would furnish satisfactory material for timbering. According to the report of the State Board of Forestry, blue and sugar gum timber have been used with success in timbering mines near Escondido that are flooded during a part of each year.

The sugar gum (*Eucalyptus corynocalyx*), the red gum (*Eucalyptus rostrata*), and the narrow-leaved ironbark (*Eucalyptus crebra*) are eucalypts that should recommend themselves to the mining industry.

The durability, hardness, and strength of the timber obtained from these species make them eminently fit for the mines. The sugar gum furnishes a wood that is very durable as railway ties, as posts, and for underground use. Baron von Mueller states that posts set in the ground 15 years showed no signs of decay. The wood warps but little in drying, and when dry is very hard. In the interior valleys of southern California and Arizona it withstands both the intense heat of summer and the frost of winter. For desert country this is one of the most eligible among timber eucalypts. It does not grow quite as rapidly as the blue gum, but furnishes a more valuable timber. In New South Wales the ironbark is considered the king of hardwoods. The narrow-leaved ironbark endures a greater variety of climatic conditions than do other ironbarks. At Fresno, California, it grows vigorously, and young trees have grown well near Phoenix, Arizona. It is said to be content with poor soil, and, judging by experience with it thus far, it ought to grow in most valley and hillside situations in the Southwest.

An important point in considering the value of commercial plantations of eucalyptus has been determined by the United States Forest Service, from strength tests made by them, namely, that the fastest growing species are also the strongest. The bending tests showed the modulus of rupture in pounds per square inch to be:

	Age in Years.	Pounds.
Sugar gum	15	25,334
Blue gum	30	23,265
Blue gum	15	16,900
Red gum	15	14,380

The crushing strength in pounds per square inch, the compression being parallel to the grain, was:

	Age in Years.	Pounds.
Sugar gum	15	11,290
Blue gum	30	12,310
Blue gum	15	8,190
Red gum	15	7,723

The crushing strength of some American woods, in pounds per square inch (from Kidder):

	Pounds.
Oregon pine	4,500
California redwood	3,000
White pine	3,500
White oak	4,000

Comparison with Forest Service tests on hickory shows that 30-year blue gum is stronger than XXX hickory, and that 15-year sugar gum is nearly as strong as second-growth hickory. These tests would seem to disprove Mr. Carter's statement that the blue gum wood is of poor quality, and not strong enough for props in heavy ground. Perhaps the timber was not properly seasoned, or from immature trees. We see from the above tests the difference in strength between 15 and 30-year blue gums.

I believe mining companies would do well to plant eucalyptus for their future supply of mine timbers, not necessarily at the mine, but in the nearest locality available for a commercial plantation. A careful selection of species suitable for this work and a thorough investigation of local climatic conditions, by someone familiar with the requirements and cultivation of eucalyptus, may save money and avoid failures."—L. W. SYMMES.—*Mining and Scientific Press*, Dec. 26, 1908, p. 870. (W. R D.)

BLASTING GELATINE. — "Considerable interest has been aroused, not only in Colorado but throughout the country, by the rapid advance recently made by Contractor A. E. Carlton in driving the Cripple Creek drainage tunnel.

In February, 1908, when Mr. Carlton took up this work, the 'double-shift' system was inaugurated, but the great difficulty in blasting the round quickly and completely made it almost impossible to drill down, fire, and throw back the muck twice in 24 hours. This was due principally to Mr. Carlton's inability to secure an explosive quick and powerful enough to bring out the 10-ft. cut hole without firing them many times, the rock, a close tough granite, offering great resistance at the points of these cut holes, which were started 2½ ft. on either side of the centre of the tunnel face and drilled so that their points almost came together on the centre line of the tunnel.

Heavy charges of 60% gelatine dynamite were used for some months in these cut holes, but without very satisfactory results, the best month's work up to September carrying the face back 301 ft. only. At this time it was decided to make a trial of Du Pont blasting gelatine, and accordingly a small quantity was secured and the cut-holes loaded with the same amount of this explosive as would have been used of 60% gelatine dynamite. The result of the first blast was terrific. Every pane of glass in the engine house and smith shop outside of the tunnel entrance and more than 3,000 ft. from the face being shattered to bits. The 24-in. ventilator pipe and 6 in. air line were blown down 500 ft. out from the face, and things mixed up generally. However, Mr. James A. McIlwee, the superintendent, immediately profited by his first experience with the powerful explosive, and after a few more trials succeeded in adjusting the charges satisfactorily by loading from four to seven 1¼ in. × 8 in. cartridges of blasting gelatine in the points of the cut holes, and six or seven 1½ in. × 8 in. cartridges of 60% gelatine on top with tamping to the face—the detonator, a quintuple or preferably a No. 20 blasting cap, being placed in the top or outside cartridge of the blasting gelatine and behind the gelatine dynamite. With the use of the blasting gelatine, Mr. McIlwee was able to bring out his cut to the back of the holes in two shots, the first shot blowing out about half of the cut and breaking back and chambering the cut holes so that a pocket was formed where the two holes came together, large enough to take from fifteen to twenty cartridges of 1½ in. × 8 in. 60% gelatine dynamite for the second shot. As this second charge could be concentrated directly back to the centre of the cut which was already relieved of at least one-half of its resistance, it never failed to bring out all of the remainder of the cut.

Occasionally one cartridge of blasting gelatine was also used in the point of each of the 'relief-cut' holes but usually a charge of 60% gelatine dynamite was all that these holes required. The remainder of the round, including side or rib, top and bottom of lifter holes, was charged with 50% gelatine dynamite.

When the tunnel was measured up at the end of September, Mr. Carlton had to his credit a drive of 353 ft. for 25 working days, or a gain over his best previous month of almost 17% in returns with practically no increase in outlay, as the slightly greater cost of the blasting gelatine over that of gelatine dynamite was offset by the reduction in quantity used."—*Mines and Minerals*, Jan., 1909, p. 282. (A. R.)

CAISSON DISEASE.—"A commission of Belgian medical experts examined 166 caisson workers before and after their work, the shift lasting from 8 to 12

hours, and found (1) that the blood-making function, as shown by the hæmoglobin contents, was actually increased during their work; (2) that so long as the pressure does not increase beyond 3 atmospheres (45 pounds) the men feel perfectly well and perform their labour with more ease and even less fatigue than under normal atmospheric pressure; (3) that men of temperate habits, with a sound heart, lungs, and nervous system, suffer no injurious effects, and none others should be employed; (4) the real injury is done by a sudden removal of atmospheric pressure in a hasty "locking-out" process, for which the workmen are often to blame,

The general rule in "locking out" should be to allow at least one minute for each 6 pounds of pressure within the chamber.

The symptons of the so-called caisson disease are rarely observed until the pressure equals 20 pounds, and usually do not appear for some minutes or hours after emerging. In addition to the symptoms already mentioned, there may be hemorrhage from the nose, mouth, and ears; headache, dizziness, rapid pulse, sweating, severe pain in the back, extremities, or region of the stomach, and vomiting. Partial deafness and symptoms of motor paralysis, more or less general, but most frequently confined to the lower extremities, are frequently observed. Cases with pronounced head and spinal symptoms usually prove fatal. The milder cases, as a rule, recover sooner or later, although the muscular pains and paralytic symptons may persist weeks or even longer."—*Bulletin 75, U. S. Department of Labour.—Mines and Minerals*, Jan., 1909, p. 274d. (W. R. D.)

Lubricating Ropes and Guides.—In *Glückauf* for January 2, Mr. Dobbelstein describes certain devices recently introduced into German mining practice for greasing cage ropes and guides, on account of the waste and imperfect manner in which the work is done by hand. At the Hugo colliery of the Harpener company, a hinged wrought iron casing is arranged in connection with a compressed air pipe. The rope passes through a central bore of the casing, and engine oil (which possesses the necessary fluidity and prevents rusting) is forced through a number of small orifices at an angle of 45° with the rope. The bore of the casing is eased at the top and bottom, to prevent loose ends of wire from being torn out of the rope; and the casing is mounted on rollers, so as to follow any deviations of the rope from the centre of the winding compartment. The consumption of oil is only one-sixth that of the solid grease previously used; and there is also a considerable saving in labour, one man being able to put the apparatus in position for use, whereas hand greasing takes three men a full day. Experiments made with a view to obtaining an efficient, non-slipping, anti-corrosive lubricant for underground haulage ropes showed that the best results for this purpose are obtained by a mixture of tar and driers, this possessing the requisite tenacity and elasticity. The problem of greasing cage guides is somewhat difficult, oil or grease being prohibited, on account of the danger of fire, whilst soft soap soon washes off in wet shafts, and is also wasteful in application by hand. A thin mixture of soap and graphite was eventually prepared, which greatly lessens the friction of the cage shoes on the guides, and can be applied by compressed air. For this purpose a closed vessel, divided into two compartments, for the grease and air respectively, is mounted on tub wheels, and charged with compressed air from the main until a constant pressure is indicated by the gauge. The mounted vessel is run into the cage, and a branched delivery pipe leading from the oil chamber is fitted with two pairs of nozzles that pass through holes in the side of the cage and discharge the grease on to the inner corners of the cage guides as the cage moves down the shaft, the compressed air being admitted to the oil chamber through a reducing valve. A saving of about £20 per annum is effected by the use of this device."—*Mining Journal*, Jan. 16, 1909, p. 79. (A. R.)

Stresses on Winding and Conducting Ropes.—At a recent meeting of the Manchester Geological and Mining Society Mr. J. Hindley read a paper by himself and Mr. J. Stone on the above subject.

By means of an indicator which they had designed and constructed for ascertaining the stresses on winding ropes, a number of experiments had been conducted and diagrams obtained showing the stresses during winding. As a result of their experiments they concluded that the prevalent idea that the greatest stress on a winding rope occurred at the beginning of a wind was shown to be entirely wrong. In every case the application of the stress was shown to be very gradual. The greatest stresses on the winding rope were due to the vibrations of the cage. The vibrations were caused by (1) the pulsations of the engine, (2) the sudden checking of speed by shutting off the steam, or by the application of the brakes.—*Mining Engineering*, Jan. 1909. (C. B. S.)

Bending Stresses in Wire Ropes.—In a paper read before the Australasian Institute of Mining Engineers, the author criticises the Reuleaux and Hrabak formulæ for the stresses due to bending in a wire rope. From the change of curvature of a single helix whose axis is bent into a circle it is readily deducted that the maximum stress due to bending is $(Ed \cos^2 \alpha \cos^2 \beta)/D$, where E is Young's modulus for the steel of which the wires are made, d is the diameter of each wire, D that of the rope, α the angular pitch of the wires in the strand, β that of the strands in the rope. In direct tension each helix of wire is not perfectly free to behave as a spiral spring in tension; hence, although the elongation of the rope as a whole is greater than it would be if it consisted of a bundle of parallel wires, yet it is not so great as for a number of free helices. Doubtless to some degree the same action occurs in a rope bent over a pulley. Tables give the tensions required to produce the same stress as given by the bending for various ropes and diameters of drum. The results of experiments are given in which pieces of rope were bent by a known force or to a known radius. The formula is found to be in good agreement with the results. By bending in the opposite direction the work absorbed in overcoming internal friction is estimated. This amounts approximately to $Ml/2R$, where M is the bending moment required to bend a length l of the rope to radius R.—*Indian Engineering*, Jan. 16, 1909, p. 45. (A. R.)

To Destroy Explosives.—The best way to destroy ordinary black gunpowder is to throw it into a stream under conditions that prevent any harm coming to human beings or animals through the dissolving of the saltpetre. If no suitable stream is available, the gunpowder may be stirred, with water in tubs, or the dry gunpowder may be poured out on the ground in a long thin line and ignited with a fuse at one end. To destroy dynamite cartridges, the paper wrappings should be carefully removed, the bare cartridges laid in a row with their ends in

contact, and the first cartridge ignited with a fuse without a cap. Even with these precautions a simultaneous explosion of the entire mass may occur, so that it is wise to retire to a safe distance. The row of cartridges should be laid parallel with the wind and ignited at the leeward end, so that the flame will be driven away from the mass. Frozen dynamite should be handled with especial care, as its combustion is peculiarly liable to assume an explosive character. A small quantity of dynamite may be destroyed by throwing it, in very small bits, into an open fire, or the cartridges may be exploded one by one in the open air with fuses and caps. Dynamite should never be thrown into water, as the nitro glycerine which it contains remains undissolved and capable of doing mischief. Other explosives which contain nitro-glycerine should be treated in the same way as dynamite. Ammonium nitrate explosives may be thrown in small fragments into an open fire, or, if they do not contain nitro-glycerine, may be destroyed by means of water. Explosive caps should be exploded singly with pieces of fuse.—*Engineer.—Science and Art of Mining*, Jan. 23, 1909, p. 266. (A. R.)

Blown-Out Shots.—'A blown-out shot may thus be defined: (1) 'A shot insufficiently stemmed whereby the tamping offers less resistance than the force generated by the combustion of the explosive.' So that the rocks in which the charge is inserted do not collapse or shatter, but allow the explosive force to escape through the narrow shot-hole at a tremendous velocity causing the flame and force of explosion to extend a considerable distance, at the same time generating great temperature to such an extent that, though in ordinary conditions the flame of certain explosives is not of sufficient temperature to ignite gas and dust, it will, subject to enormous velocity and explosive force, when coming in contact with the small particles of fine dry coal dust, ignite them. This is illustrated by stones falling from some height and striking another blow, and the force of contact generating a spark. The concussion of the air current in the vicinity of a heavy blown-out shot is sufficient, in my opinion, to ignite the small particles of coal dust, when we remember that each particle is surrounded with a thin envelope of highly inflammable mixture called 'ferrous sulphide' caused by decomposition, which becomes more prominent after the dust is treated by water and has since dried; and again during the drying process of the dust it has been proved that the vapour arising from the dust is nearly as inflammable as petroleum. This vapour, subject to severe concussion, can have sufficient temperature imparted to it to ignite the coal dust, and form the first fruits of a terrible disaster.

(2) A blown-out shot may be defined as an overcharged shot-hole, whereby too much explosive is introduced, hurling the rocks a long distance, at the same time a portion of the flame escapes by way of the shot-hole with tremendous force, often greater than the first instance, and with the greater liability of causing an explosion of coal dust. Also, the flying stones may seriously injure persons who may not have been properly sheltered themselves, and, again, the flying stones may knock timbers out, causing a large fall, and an outburst of fire-damp, which may become ignited by the falling stones, or the flame of the blown-out shot, should the fall occur practically instantaneously, as we have records of explosions caused in this manner.

(3) A blown-out shot may be defined as a shot unskilfully placed, or in a crack or thin layer of coal. Such shot may have been properly stemmed, and with a proper charge of explosive, yet the products of combustion escape by way of the cracks in the rocks, or above or below the thin layer of coal, especially if the collier allows the rippings to hang some distance, which may have loosened sufficiently to cause a thin cavity, often unseen, which at the same time may be full of fire-damp impossible to detect. It can be readily seen that a shot fired under these circumstances is dangerous in two ways, by blowing out and igniting dust, and at the same time igniting the fire-damp concealed in the rippings, causing a serious accident, setting fire to everything inflammable within its reach, which may extend with terrible rapidity and spread and create the horrible disasters we are all too well acquainted with. Again, the heat generated by shots when spreading among cracks in the rocks may be sufficient to ignite other explosives which may yet remain to be exploded by the shot-man, and such explosion may occur just as he is in the act of lighting the fuse or connecting the cable to the detonator wires, or possibly inserting explosives in uncharged shot-holes. Many men have lost their lives in this way. A case similar to this occurred after a blown-out shot at Glyncorrwg Colliery, South Wales, where a shot-man attempted to insert a charge in a hole that had blown out in a hard heading, and there was sufficient temperature in the shot-hole to ignite the second charge, instantly killing the two men who were engaged on the work. At the inquest it was said that even cold water had been splashed into the shot-hole before re-charging, yet the charge fired, proving the great heat generated during blown-out shots.

(4) A blown-out shot may also be defined as one where the charge is inserted, and before being tamped has ignited by the heat generated by previous shots, blowing out with tremendous force; according to the amount of explosive that has been inserted, and its density.

(5) A blown-out shot may also be defined as one where a shot fires prematurely by pressing a charge into a hole of insufficient size, or by using an iron bar, etc.

(6) A blown-out shot is also one caused where the hole is not properly cleaned, and a thin layer of stone or coal dust remains on the sides of the hole, which may allow the products of combustion to escape even after being tamped.

Precautions to prevent Blown-out Shots:—

(1) The colliery manager to comply with the C.M.R. Act by appointing the most skilful and careful shot-men he can find, and give them a short lecture as to their duties, the dangers of shot-firing, explaining how accidents may occur, and testing the knowledge of the shot-men in the general practical requirements, also their knowledge of general rule 12 and its amendments. No man should be appointed a shot-man who does not know general rule 12.

(2) Instruct the shot-man to charge and fire all shots himself, and not to fire more than a specified number, as the manager may deem safe. In our colliery only five shots are allowed to be fired in each district each shift.

(3) Each collier to state the length of each shot-hole with clalk on a piece of board, so that the shot-man can properly proportion the charge required.

(4) The shot-man should properly stem the shot-hole from the charge to the mouth of the shot-hole.

(5) He must not fire shots placed in cracks or thin layers of coal.

(6) He must not stem the hole with coal dust, but only good plastic clay, or tamping plugs.

(7) Do not press a charge into a hole of insufficent size, or tamp the first portion too hard on the charge, and only use a proper tamping rod of copper or wood.

(8) Properly clean out every shot-hole of dust with a copper scraper before inserting the charge.

(9) The heading or coal face should be properly undercut or holed, and see that all unnecessary resistances are picked away and removed as sprays, and pieces of coal or stone.

(10) Shot-holes should not be too short, and should be of sufficient diameter to only just allow the explosive to pass easily.

(11) Where there are high currents of air the shot-holes should be placed with the direction of the air current with the mouth of the shot-hole pointing from the downcast shaft. Thus the shot will not blow against the air. No doubt explosions have been caused in this way, due to the concussion caused by the explosion of the shot against the air and dust which contains occluded gas, or heavy hydro-carbons, which according to Prof. Bedson, are even more readily explosive than fire-damp, and take fire at a lower temperature.

(12) Prohibit gunpowder, and use safety explosives as a substitute to secure greater safety in dusty and gaseous mines where blasting may be necessary.

(13) See that the dust and all parts of the roadway are thoroughly wet for a distance of 20 yards radius of the shot-hole to prevent the extension of fire should a blown-out shot occur.

(14) Comply with the C.M.R. Act strictly in case of the presence of fire-damp; as before stated, dust alone is highly dangerous, but becomes more so when assisted by fire-damp.—DAVID FOSTER.—*Science and Art of Mining*, Jan, 23, 1909, p. 281. (A. R.)

MISCELLANEOUS.

CAPACITY OF PIPES.—"This table shows the number of gallons discharged per minute through pipes for specified sizes and grades.

—	Fall per 100 ft.							
Size of Pipe.	1 in.	2 in.	3 in.	6 in.	9 in.	1 ft.	2 ft.	3 ft.
3 in. ...	13	19	23	32	40	46	64	79
4 in. ...	27	38	47	66	81	93	131	163
6 in. ...	75	105	129	183	224	258	364	450
8 in. ...	153	216	265	375	460	529	750	923
9 in. ...	205	290	355	503	617	711	1006	1240
10 in. ...	267	378	463	655	803	926	1310	1613
12 in. ...	422	596	730	1033	1273	1467	2076	2554
15 in. ...	740	1021	1282	1818	2224	2464	3617	4467
18 in. ...	1168	1651	2022	2860	3508	4045	5704	7047
24 in. ...	2396	3387	4152	5871	7202	8303	11744	14466
30 in. ...	4187	5920	7252	10257	12580	14504	20516	25277

Canadian Engineer.—The New Zealand Mines Record, Dec. 16, 1908, p. 233. (A. R.)

ACTION OF GASES ON BOILER PLATES.—"It is well know that even the best Lowmoor furnace plates have, after years of work, acquired a brittleness which cannot then be removed by annealing, and Mr. M. Longridge has recently published details of a boiler which had been in continuous and satisfactory work for 70 years, but which had become so brittle that holes could be knocked into it with a hammer. Such plates would, of course, not stand renewed manipulations in the boiler shop, and might easily crack if repairs were carried out, but if left undisturbed they are likely to do their duty in resisting ordinary stresses, because their strength has not been diminished; on the contrary, it may have been increased. The cause of this brittleness may be either a molecular change due to age alone, or a chemical change due to absorption of gases. It is not as widely known as it should be that neither solids nor fluids are free from gases, that it is most difficult to remove them, and that absorption is constantly taking place. Glass is by most people looked upon as being impervious to gases, but the gradual change of soaped glasses to an amethyst tinge, which is sometimes very noticeable with windows of old houses, proves not only that these glasses absorb oxygen, but also that the oxygen effects a chemical change of the manganese salts in the glass. It is, therefore, not surprising that the noxious fumes of boiler furnaces enter the plates, and possibly also hydrogen from the water side. It is, at any rate, certain that a very small percentage of hydrogen makes mild steel not only brittle, but so hard that it will scratch glass, and very small traces of nitrogen also impart brittleness to mild steel. Unfortunately, these gases are not easily detected by chemists, and are rarely recorded in their analyses, so that when cases of brittlenes are dealt with the possibility of its being caused by gases cannot be ascertained."—C. E. STROMEYER, Manchester Steam Users Association.—*Page's Weekly*, Jan. 1, 1909, p. 2. (A. R.)

CONVERSION TABLES FOR ASSAY VALUATIONS.—The following tables, or rather portions of them, were computed some years ago to facilitate the reduction of certain assay values to a common basis. They have been extended as far as seems desirable, and are presented to the readers of *Mines and Minerals* in the hope of saving time and labor to some of them.

Bases of Computation.—The gram is taken as 15,4320 grains. The value of a troy ounce of fine gold is assumed as being exactly $20.67, instead of $20.6718346+, resulting in an error of less than one in 10,000. Values in English coin are based on the assumption that an ounce of fine gold is worth 4·25 pounds sterling, or 85 shillings, 1,020 pence; this is too high by about one part in 2,000, the true value being 1,019.45 pence. It is useless to attempt a closer approximation in practical work, for the simple reason that gold bullion assays are rarely reported closer than the nearest half millieme, or to within one part in 2,000. At the values adopted one dollar is equivalent to 4.11224 shillings, and one pound sterling to $4.86353.

Foreign and Obsolete Values.—The adarme (36 grams or $\frac{1}{16}$ of the Spanish ounce), sometimes used by Mexicans colloquially, and especially with reference to placer work, is about 1.8 grams, which in fine gold would be worth $1.20. For practical purposes, however, an adarme of ordinary gold may be taken as equivalent to $1, and this exactly true for gold 830 fine.

Russian reports state values in zolotniks per 100 poods, but for low-grade placer deposits doli per 100 poods are used. As a dola is $\frac{1}{96}$ zolotnik we may take $\frac{1}{100}$ of the values given in the table for the zolotnik, without serious error.

Marcos per cajon were formerly used in some South American countries, one marco per cajon apparently ranging between 100 and 70 parts per million. One oitavo per quintal corresponds practically to 2 ounces per ton.

The loth per centner, used in the older German works, corresponds to one part in 3,200, which is nearly 9 ounces per short ton or 10 ounces per long ton. In some cases there seems to have been a considerable variation from this ratio, the value being sometimes taken as one part in 3,520, or $\frac{10}{11}$ of that used in these tables—the centner being then assumed as 110 instead of 100 pounds. The quentchen was $\frac{1}{4}$, and the denär $\frac{1}{16}$ of the loth."—W. J. SHARWOOD.—*Mines and Minerals*, Jan., 1909, p. 250. (W. R. D.)

FINENESS OF BULLION AND ALLOYS OF PRECIOUS METALS.

Denomination.	Equivalent in Milliemes or parts per Thousand.	
1 carat ...	41·660	24 carats = 1lb. troy (England) ,, 1 mark (Germany, etc.)
1 gr. per marc	·217	4,608 gr. = 1 marc of 8 oz. (France, Spain, etc.)
1 oz. per marc	125·000	8 oz. = 1 marc (France, Spain, etc.)
1 loth (silver)	62·500	16 loth = 1 mark (Germany etc.)

VOLUME AND WEIGHT OF FINE GOLD AND SILVER.

	One Cubic Centimeter	One Cubic Inch.	One Cubic Foot.
Fine Silver:			
Weight: grams ...	10·57	173·21	299307·00
Weight: troy ounces	·339825	5·5687	9622·72
Fine Gold:			
Weight: grams ...	19·3	316·269	546,513
Weight: troy ounces	·6205	10·1680	17,570·39
Value: U.S. dollars	$12·82^{57}	$210·17	$363,180
Value: pounds sterling	£2·647	£43·214	£74,674

ASSAY VALUATIONS.

Values.	One Part In.	Per Cent.	Per Metric Ton.			Per Long Ton of 2,240 Pounds.			Per Short Ton of 2,000 Pounds.	
			Troy Ounces.	U.S. Dollars.	Grams.	Troy Ounces.	Oz. Dwt. Gr.	U.S. Dollars.	Troy Ounces.	U.S. Dollars.
1 per cent. ...	100	1	321·50	6,645·40^{6}	10,000	326·666	326 13 8	6,752·20	291·666	6,028·75
1 gm. per metric ton = one part per million ...	1,000,000	·0001	·03215	·66^{45}	1	·03266	15·68	·67^{5}	·029166	·60^{29}
1 troy oz. per short ton ...	29,166·66	·00342857 $=\frac{24}{7000}$	1·1023	22·78^{45}	34·2857	1·120	1 2 9·6	23·15	1	20·67
1 troy oz. per long ton ...	32,666·66	·0030612 $=\frac{3}{980}$	·9842	20·34^{34}	30·612	1	1	20·67	·892857	18·45^{3}
1 troy oz. per metric ton ...	32,150	·0031104 $=\frac{27}{875}$	1	20·67	31·104	1·016	1 7·7	21·00	·90720	18·75^{2}
1 dollar gold per short ton ...	602,875	·00016587	·0533285	1·1023	1·6587	·054185	1 2	1·12	·048379	1
1 zolotnik per 100 poods ...	384,000	·0002604	·083724	1·73^{06}	2·604	·08507	1 16·83	1·75^{8}	·075954	1·57
1 loth per centner	3,200	·03125	10·046875	207·66	312·5	10·2183	10 4 9	211·21	9·11458	188·14
1 oitavo per quintal ...	16,384	·00610	1·9623	40·56	61	1·9938	1 19 21	41·21^{18}	1·78654	36·80

CONVERSION TABLES—WEIGHTS.

	Grains.	Penny-weights.	Troy Ounces.	Avoirdupois Ounces.	Avoirdupois Pounds.	Grams.	Fine Gold Value.	
							United States.	British.
1 Grain ...	1	·041666	·0020833	·00228571	·000142857	·0648	4·306c.	2·125d.
1 Pennyweight	24	1	·0500	·0548571	·00342857	1·5552	$1·0335	4·25s.
1 Troy ounce ...	480	20	1	1·0971428	·0685714	31·104	$20·67	85s.
1 Troy pound ...	5,760	240	12	13·165714	·822857	373·248	$248·04	£51
1 Avoirdupois oz.	437·50	18·22917	·911458	1	·06250	28·35	$18·84	77·474s.
1 Avoirdupois lb.	7,000	291·666	14·58333	16	1	453·60	$301·4375	£61·97
1 Milligram ...	·015432	·000643	·00003215	·000035274	·0000022046	·0010	·06645c.	·033d.
1 Gram ...	15·432	·643	·03215	·035274	·0022046	1	66·45c.	2·73275s
1 Kilogram ...	15,432	643	32·15	35·274	2·2046	1,000	$664·54	£136·64

THE MICROGRAPHICAL STUDY OF CEMENT.—In view of the advantages which have followed the micrographical examination of metals, the Prussian Royal Laboratory for the Testing of Materials have recently, according to the *Builder*, applied the same method to the study of various cements and in particular to Portland cement. By polishing the surface of Portland cement clinker the investigators were enabled to identify the two principal constituents—described as *alith* and *belith*—which were found to

occupy unequal and unequally distributed areas of the surface under examination. The application of re-agents such as hydrochloric, nitric, acetic, salicylic and iodic acid showed still more clearly the differences between these two constituents. Among other re-agents a weak solution of citric acid violently attacked the polished surface, which was quite unaffected by oxalic acid. Hydrofluoric acid gave a characteristic reaction, revealing by discoloration the presence of slag in some specimens. By heating samples to high temperatures the investigators were enabled to ascertain that the hardest crystals—those of *alith*—were not a simple hydrate of calcium, since they were able to stand without disintegration the temperature of 900°C. Finally, the examination of thin laminæ of cement served to illustrate the optical properties of these crystals and revealed the presence of a third constituent—described as *celith*. So far, however, the research has thrown no light upon the nature of the re-action or molecular re-arrangement responsible for the setting and hardening of Portland cement.—*Indian Engineering*, Jan. 16, 1909, p. 45. (A. R.)

Reviews and New Books.

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

THE ORE DEPOSITS OF SOUTH AFRICA. PART I. Base Metals. By J. P. JOHNSON. 5s. Net. (London: Crosby, Lockwood & Son.)

That there should have been no handbook of reference written on South African economic geology since Wilson-Moore's little volume appearing fifteen years ago is by no means creditable to local geologists, but it is a circumstance that should be of considerable significance in giving Mr. J. P. Johnson's treatise, named above, the circulation it merits. Having so clear a field Mr. Johnson was able to deal with his subject from any standpoint. He has worked on thoroughly scientific lines and if, in consequence, he has limited the range of popularity, he has at least produced a work of more enduring value. The book may be said to form a good companion work to Hatch & Corstorphine's "Geology of South Africa." The claim made for it is that it should meet a demand "for a co-ordinated and condensed account of the ore-deposits at present known in South Africa"—the part issued only covering the base metals. Considering the brevity of the production, this aim has been skilfully achieved. In thirteen chapters running to sixty pages in all, the author covers a remarkably wide field. The first chapter is somewhat ambitious for a volume of the present scope and treats with risky conciseness of the origin and classification of ore deposits. Those who have followed the famous controversy of the "Ascensionists" and their antagonists, excited by Posepny's classic paper on the "Genesis of Ore Deposits" will sympathise with Mr. Johnson in his efforts to present an eight-page condensation of the subject as a guide to prospectors, and, at the same time, commend the measure of his success.

The body of the volume is made up of short accounts of the occurrences in South Africa of the following metalliferous ores: titaniferous and chromiferous iron oxides, nickel, copper, cobalt, tin, molybdenum, tungsten, lead, mercury, antimony and iron. Descriptions are based for the most part on personal observation, but other authorities are frequently quoted and fully acknowledged. As a broad review, these chapters are excellent and while the student of any particular class of product may find the data too scanty, he will generally be put on the track of more detailed references. The treatment of the subject should appeal to many readers in Europe and America, who are anxious to learn at a glance the present stage of South African base metal mining development.

The final chapters comprises "Hints to Prospectors." Mr. Johnson practically states that there are no competent prospectors in the country. It is to be feared, however, that the standard of qualifications he establishes is rather elevated and would bar the claims to recognition held by the host of "prospectors" who by virtue of their numbers rather than an individual knowledge of "magmatic differentation" or "metasomatic replacement," add yearly to the world's wealth by their discoveries in Australasia or America. Mr. Johnson is so zealous a supporter of the genuinely scientific side of his profession that he maintains a higher standard of thought and expression than that commonly met with in handbooks for the prospector's guidance. In the first place, he presupposes a better grounding in the elements of geology than the average prospector in any land possesses; in the second, he does not give those instructions as to field equipment and simple expediencies for the rapid determination of minerals, which usually appear in the popular vade mecum.

If the defects of Mr. Johnson's volume have been discussed at disproportionate length and its signal merits too briefly noticed some excuse may be found in that a second part is to follow and omissions can be simply remedied. An index, at least of place-names, would be serviceable. If Mr. Johnson should later produce a volume on the precious metals, and a third on precious stones and such mineral products as asbestos, lime, mica, graphite and magnesite, etc., he would fill a deplorable blank in South African scientific literature, and, judging by his first publication, would fill it well. (R. S.)

COAL MINING. By D. BURNS and G. L. KERR. Part III. (London: Whittaker & Co., 2, White-Hart Street, Paternoster Square, E.C.)

"The whole of Part III. of this publication is devoted to the sinking and fitting of shafts, a subject of primary importance, both to the mining engineer and colliery owner. The general character of the work as a comprehensive and practical guide on the technical side of coal mining is admirably sustained in this section. The treatment of the subject is exhaustive without being discursive, while a word of special praise is due to the illustrations, which are not only numerous but of so eminently practical a type as greatly to assist the task of the student. Commencing with the forms of shafts, the authors point out the advantages and disadvantages of the circular and rectangular types, and proceed with a series of calculations for determining the number and size and position of the shafts required. The sinking of circular shafts is then described and illustrated in great detail, the respective merits of brick walling and concrete lining being touched upon and full directions given for the work of construction. Not the least valuable feature of this portion of the book is the information given relative to the cost of sinking, not only in this but in foreign countries also. The water problem receives full consideration, and the calculations involved in connection with shaft tubbing are presented with commendable clearness. The raising of the débris and its handling

on the surface are also subjects upon which the authors have much useful information to convey. With the same thoroughness and grasp of the subject the sinking of rectangular shafts is then proceeded with, followed by a section upon the enlarging and deepening of shafts. The different methods adopted in sinking through running sand or mud are fully dealt with, prominence being given to the compressed air and freezing system. Some useful hints on the ventilation and lighting of sinking shafts, and on the best forms of bucket to employ conclude the Part, an appendix setting forth specimens of sinking specifications."—*Iron and Coal Trades Review*, Dec. 4, 1908, p. 2,418. (A. R.)

METALLIC ALLOYS: Their Structure and Constitution. By G. H. GULLIVER, B.Sc., F.R.S.E., with 104 illustrations, 8×5½, xvi.+254 pp. (Charles Griffin &Co., Ltd.), 1908.

"The author is to be congratulated on the success of a resolute attempt to introduce a certain amount of system into a subject which has hitherto consisted rather of a series of disconnected experiments and investigations by various authorities than any intelligible scientific theory, treating of allows as a whole. The modern methods of investigation, by means of the etching of polished surfaces and the employment of micro-photography, may still be said to be somewhat in their infancy. In the case of many industrial alloys of commercial importance, the composition of which has long been known, we rely greatly on rule of thumb methods, but owing to the efforts of the Alloys Research Committee of the Institution of Mechanical Engineers, much light has recently been thrown on matters which have hitherto appeared obscure. The employment of the pyrometer and the study of critical temperatures in regard to certain alloys has opened up new methods of investigation, which have yielded important results, and placed in the hands of the metallurgist the means of determining facts which can be explained by no other means. In the case of certain refractory substances, the adoption of the electric furnace as an aid to research has proved of the utmost value.

The author deals fully with binary and ternary alloys, and discusses the known cases in which evidences are found of the formation of definite chemical compounds. A chapter is devoted to the equilibrium conditions in metallic mixtures with special reference to the phase rule. In the opening chapter a good outline is presented of the preparation of specimens for microscopic examination and of the mode of conducting the various investigations. This will prove a useful and suggestive work for the student, and it covers a field on which much still remains to be accomplished. The author's explanatory diagrams are clear and good, and the work is well illustrated with micro-photographs of the etched surfaces of characteristic alloys."—*The Times Engineering Supplement*, Dec. 9, 1908, p. 20. (W. A. C.)

THE MECHANICAL ENGINEERING OF COLLIERIES. Vol. II. By T. CAMPBELL FUTERS. Published for the Proprietors, The *Colliery Guardian Company*, Limited, by the Chichester Press, Holborn, London, E.C.

"The Volume before us deals with Haulage, Pumping, The Generation and Transmission of Power, Coke Ovens, and in the concluding chapter general information on such subjects as Ventilation, Coal-Cutting Machines, Coal Face Conveyors, Workshops, Brickworks, Stores, Ambulance, etc., is given in a clear and interesting manner. Throughout this work the illustrations are of a high standard, the author evidently recognising the material assistance of careful draughtmanship. There is abundant evidence of systematic arrangement, and the reader will find his progress through the work unattended with those annoying repetitions and "doublings" which go so far to mar many text-books devoted to mechanical engineering. The chapter: The Generation and Transmission of Power is in need of expansion and revision. The treatment of the alternating current motor is cursory. The volume displays careful classification, and will prove eminently entitled to a prominent place on the bookshelf of the colliery manager."—*Science and Art of Mining*, Feb. 6, 1909, p. 297. (A. R.)

AUSTRALIAN MINING AND ENGINEERING REVIEW contains a variety of interesting mining news, district reports and special articles. The "Review" is well illustrated and deals in a thorough manner with such wide subjects as Mine Surveying, Ventilation of Mines, Coal Examinations, and Australian Railways, and a special section is devoted to Electricity; it should have a useful and prosperous career, filling a much felt want. (C. B. S.)

THE ELECTRICAL EQUIPMENT OF COLLIERIES.—By W. G. DUNCAN and D. PENMAN. (London: Messrs. Scott, Greenwood and Son). Price, 10/6 net.

"The opening chapters call for little or no comment. General principles, magnetism, units, dynamos, and motors are described and illustrated, and in a manner similar to that of the usual text-book. In the chapters dealing with the Transmission and Distribution of Power, Prime Movers, and Lighting by Electricity much useful information is given. "Initial outlay and Working costs of Electrical Installations" are really good, but when the authors come to electricity in practical use in the mine, viz., coal-cutting, hauling, winding, pumping, drilling, we follow them with respect. Here they are on their own ground. The information, data, etc., incorporated is worth the price the publishers ask for the book. The subject of shaft signalling is meagrely dealt with. The chapters on Dynamos and Motors and Power Transmission could be extended with advantage to include some more illustrations and full description—preferably diagrammatic—of modern circuit breakers, discriminating apparatus, networks, synchronisers, etc. Producer gas plant merits their observation. Would they have their readers know the value of the apparently valueless refuse of the mine?

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