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Proceedings
AT
Ordinary General Meeting, May 18, 1907.

The Ordinary General Meeting of the Society was held in the Chamber of Mines, on Saturday, May 18th, Mr. E. H. Johnson (President), in the chair. There were also present :—

31 Members: Dr. J. Moir, Prof. J. Yates, Messrs. R. G. Bevington, T. L. Carter, W. R. Dowling, K. L. Graham, A. Heymann, A. McA. Johnston, H. A. White, J. Littlejohn, F. F. Alexander, W. Beaver, E. H. Croghan, G. Goodwin, H. R. Grix, W. H. Jollyman, E. J. Laschinger, Hy. Lea, Jas. Lea, G. A. Lawson, W. D'A. Lloyd, W. P. O. Macqueen, P. T. Morrisby, J. F. Pyles, A. Richardson, O. D. Ross, A. Salkinson, S. H. Steels, H. Taylor, W. H. Winckworth and H. Wiley.

13 Associates and Students: Messrs. S. J. Cameron, W. M. Coulter, C. L. Dewar, J. H. Harris, J. Innes, W. H. Johnston, H. R. Jolly, R. W. Leng, R. W. Maxwell, C. G. J. Moore, C. A. Robinson, C. Toombs and W. Waters.

8 Visitors and Fred. Rowland, Secretary.

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

New Members.

Messrs. Beaver and Goodwin were elected scrutineers, and after their scrutiny of the ballot papers, the President announced that all the candidates for membership had been duly elected, as follows :—

Argall, Philip, 728-732, Majestic Building, Denver, Colo., U.S.A. Consulting Mining Engineer and Metallurgist.

Dunning, Frank George, P. O. Box 2567, Johannesburg. Mining Engineer.

Reynolds, James Percival, Geldenhuis Estate & G. M. Co., Ltd., P. O. Box 5, Cleveland. Assayer.

Rouillard, R. A., Messrs. H. Eckstein & Co., P. O. Box 149, Johannesburg. Mining Engineer.

Welsh, J. O., P. O. Box 1935, Johannesburg. Foreman Timberman.

The Secretary announced that the following gentleman had been admitted an Associate by the Council since the last general meeting.

Ayers, Gilbert F., Dynamite Factory, Modderfontein. Chemist.

Mr. Alexander Aiken was again appointed as Auditor.

The President: In the list of members elected at this meeting it is a pleasure to notice the name of Mr. Philip Argall, one of the leading metallurgists of the United States. Mr. Argall's work in connection with the sulpho-telluride ores of the Cripple Creek district of Colorado has been monumental, and the development of the treatment of complex ores by cyanidation has been given an enormous impetus by Mr. Argall's work. Mr. Argall was one of the earliest —if not the earliest—to draw attention to the importance in the nomenclature of comminution of the determination of the actual screen aperture as distinct from number of screen meshes, a system now adopted by the Sub-Committee of the Transvaal Chamber of Mines on Screen Standardisation. This Society, I am sure, extends a most hearty welcome to Mr. Argall, and we shall look forward to his contributing to our *Journal.*

The following gentlemen were then elected scrutineers for the ballot for Officers and Council for the ensuing year :—

Messrs. S. G. Bartlett, J. Gray, J. A. Jones, Jas. Lea, W. P. O. Macqueen and W. D. Morton.

The President announced that the ballot would be scrutinised in the Secretary's office on the afternoon of Saturday, June 15th. It was further decided, on the motion of Mr. Bevington, that these gentlemen should have power to add to their number or to act by themselves as they should think fit.

General Business.

The President: I have a note here from Mr. Caldecott, who wishes to say good-bye to all the members of the Society. He is leaving to-night for a six months' holiday in England and the States, and I am sure we all heartily wish him a very pleasant and interesting holiday.

The Secretary read the nominations which had been received, for Officers and Council for the ensuing year.

The President: It is very encouraging to find such a number of nominations. It does not look as if there were any fading of the interest taken in the Society.

A FEW NOTES ON THE REFINING OF BASE BULLION.

By C. W. Lee and W. O. Brunton (Members).

There has been a great deal of trouble experienced lately on several mines owing to the baseness of cyanide bullion, and there is considerable benefit to be derived by having it of a fineness of 800 or over, viz., saving the deduction of three points per thousand (extra refining charges) from the assay value of gold below 800 fine, which amounts to 3·06d. per fine ounce on realisation.

Perhaps the following cheap application, based on the experimental results obtained by Dr. Kirke-Rose on the refining of gold bullion by means of the injection of air or of oxygen into the molten metal, may prove of use to some of the members who experience difficulty in obtaining their bullion of the necessary fineness. Moreover its simplicity recommends it in preference to the more laborious method of granulating the bullion and remelting with manganese dioxide, etc.

We have brought cyanide bullion (the analysis of which was as follows:—

Gold, 64·7 per cent.	Silver, 8·8 per cent.
Copper, 16·3 per cent.	Lead, 7·4 per cent.
Zinc, 1·3 per cent.	Nickel, 0·2 per cent

Iron, trace) after two hours' injection up to a fineness of gold = 814 and silver 105 per thousand.

In another case bullion weighing 670 oz. and assaying 761 fine, after three quarters of an hour's injection was brought up to a fineness of 802. The assay of the resulting slags showed that 1·529 oz. of gold had been retained by the same.

In another instance 1·067 oz. assaying 760 fine was after one-and-a-half hours' injection brought up to 815·5 fine.

In another instance 1·360 oz. of bullion assaying as follows:—

Gold, 78·81 per cent.	Silver, 6·26 per cent.
Copper, 7·10 per cent.	Lead, 4·71 per cent.
Nickel, 0·92 per cent.	Zinc, 0·71 per cent.

Cobalt, Iron, Manganese and Sulphur, traces, was brought up to a fineness of 816, and silver 066 per thousand after four hours' air injection.

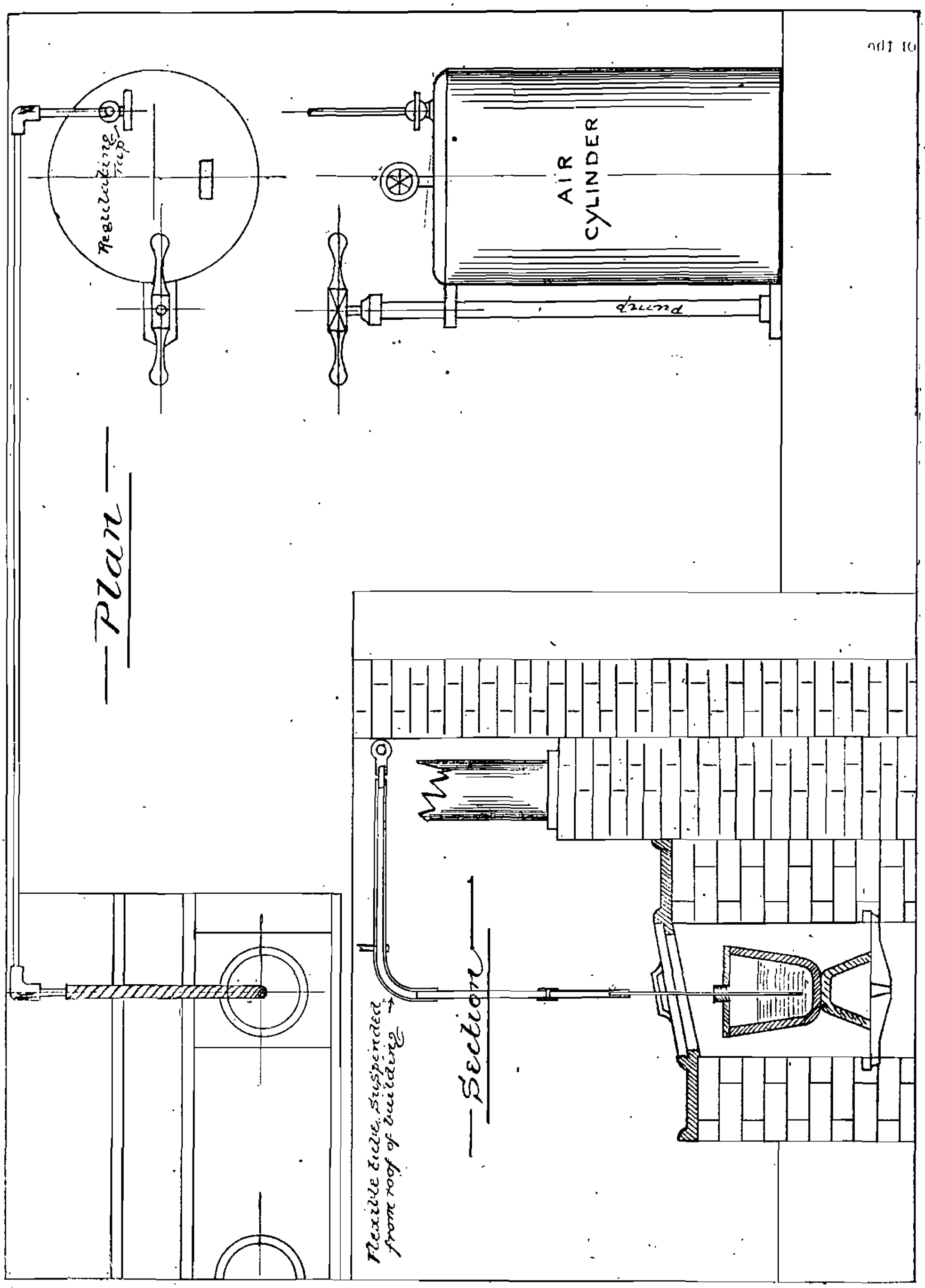
The method of procedure we have adopted is to have a cylinder similar to those used for whitewashing and sprays, etc., fitted with a pressure gauge and pump. A $\frac{1}{2}$ in. iron pipe is led from this cylinder to a point near the furnace and is made a fixture to the wall of the building. Connected to this we have a piece of flexible metallic tubing 5 ft. long, attached to which is a length of $\frac{1}{2}$ in. iron piping, which hangs vertically within 4 ft. of the crucible, when in position in the furnace. For attachment to this we have three 2 ft. lengths of interchangeable $\frac{1}{2}$ in. iron piping, one end of which is expanded in order to admit about 3 or 4 in. of one of Morgan's 24 in. by $\frac{1}{8}$ in. bore porcelain chlorine tubes. The idea of having three tubes, is in the event of one or more breaking during the refining operations. The short length of piping with the porcelain tubes fitted ready is easily affixed to the vertical pipe by the coupling shown on the rough sketch on the following page.

The making of the joint between the iron tube and the chlorine tube requires to be carefully done, and unless firmly made, the somewhat violent vibration, caused by the air bubbling through the molten bullion, will soon cause leakage. We find by placing the porcelain tube 3 or 4 in. inside the expanded end of the piping and filling the intervening space with very moist plaster of paris, that a good air tight joint is formed. In order to strengthen this joint we place three pieces of $\frac{1}{8}$ in. iron wire in the form of splints, extending about 4 in. up the iron pipe and 4 in. over the porcelain tube, and bind this over with a covering of $\frac{1}{8}$ in. asbestos cord soaked in liquid plaster of paris.

At the point where the flexible tube joins the vertical iron pipe a rope is attached running through a pulley fixed in the ceiling. By this means it is easy to adjust the height of the tube in the crucible. The fact that the pipe is supported in this non-rigid manner lessens the danger of breaking the porcelain tubes through the vibration caused by the bubbling of the air through the bullion.

The bullion is melted in a suitable size of clay-lined (salamander) crucible, standing on an inverted crucible in the furnace. This obviates a constant adjustment of the injection tube. It is best to select a pot giving the largest vertical body of metal, so that the maximum contact with the passing air may be obtained.

In the case of treating about 500 oz. base bullion, 1·5 lb. of borax and 3 lb. of fine tube-mill sand is added. The pot is covered with an ordinary plumbago cover which has a $1\frac{1}{4}$ in. hole bored through the centre for the admission of the porcelain tube. When ready, let a gentle current of air pass through the tube and gradually lower it into the molten metal, insert within an inch of the bottom of the pot and regulate the current of air so that there is a gentle ebullition of the metal. If this is done properly, there will be no projection of the metal. If after half an hour's



blowing the slag becomes very liquid, which may be observed through the hole in the cover without the removal of the tube, dry sand may be added at intervals through the hole in the cover.

We have found that the removal of the tenacious slag, consisting of boro-silicates formed from the oxides of the base metals oxidised off, is much more easily effected by skimming than by removal after pouring with the metal.

In order to ascertain the progress of refinement, dip samples can of course be taken at any stage and a rough assay made.

In conclusion, we should like to thank Mr.

E. H. Croghan for kindly assisting us by making the check analyses of the base bullion.

The President: The paper of Messrs. Lee and Brunton has one serious defect; it is too brief. It is an extremely interesting application of the method which Dr. Rose gave us two years ago.

I should like to ask Mr. Lee one question on the loss of silver they found. As far as I recollect Dr. Rose found a good deal of silver passed into the slag.

Mr. C. W. Lee: So far we have been able to account for the whole of the silver from the original assay value, and practically no loss has occurred from volatilisation.

The President: Did you find that silver passed into the slag disproportionately to gold?

Mr. C. W. Lee: On the contrary, we found that the silver contents have been normal. The base metals have naturally been the first to come away.

The President: I should like to thank Mr. Lee for that information. As far as I can recollect Dr. Rose found some loss of silver, which had passed into the slag towards the end of the refining when only a small amount of base metal was present. It is interesting to find that that can be avoided. I think Messrs. Lee and Brunton deserve a very hearty vote of thanks for their very interesting paper.

Mr. T. Lane Carter: Messrs. Brunton and Lee have given us an interesting paper. It seems to me a simple way of refining bullion up to a certain point. The bullion we get from the Tavener process averages about 870, and we have even had it as high as 900. Will Mr. Lee tell us why they get such base bullion? Is the Tavener process in use?

Mr. C. W. Lee: I must admit that in our particular district there has been an epidemic of low grade bullion from the ordinary Crosse method, and ours is simply bullion obtained from the ordinary reverberatory process. I throw out a hint as to this which may appeal to mine samplers in that particular district. Where do we get all this copper from? We have bullions running up to $16\frac{1}{2}$ per cent. copper. I do not think it is due to stripping of copper in the mill from using a very coarse mesh screen as we do in our tube mill practice, but from some extraneous circumstance. Perhaps some of the mining men may throw some light on the point.

THE SCREEN ASSAY ON THE MEYER AND CHARLTON G. M. UNDER "THE NEW METALLURGY."

(*Read at March Meeting, 1907.*)

By Chris. Toombs (Associate).

DISCUSSION.

Mr. A. Whitby: The subject of Mr. Toombs' paper is one of exceptional interest to assayers on these fields, not merely in its connection with Meyer and Charlton difficulties, but also to those of other mines. In conversation with many assayers I have found much unanimity as to the frequent occurrence of differences in duplicate screen assays, and I thought that perhaps a few observations of my own might not be out of place, even if I were unable to throw any fresh light on a very vexed question.

The first point which occurs to me is the difficulty of obtaining a sample which is really and truly representative of the material passing the screens. Here, I note that Mr. Toombs quotes a figure of 797 oz., say, 50 lb., for a 24 hours' run of 70 stamps, and that only three samples were taken per shift. It appears to me that some modification is wanted here. The size of the individual sample is quite immaterial so long as it represents the entire stream, but the number of times it is taken is a highly important factor as the grade passing the feeders is continually varying. Practice recommends, therefore, the taking of more frequent samples, and at the same time reducing the bulk taken at any one time to the smallest quantity compatible, so that the bulk sample, whilst being entirely representative, shall present less difficulty in handling. With the type of sampler in use at the Robinson Deep and taking samples every hour. I think I should be safe in saying that the total weight rarely exceeds 50 lb. for 300 stamps. With regard to its after treatment in such cases as the Meyer and Charlton and George Goch plants, opinions may vary as to the course to be pursued. I do not like the method of drying first and then washing out the soluble gold, for I am inclined to the opinion, expressed by Mr. Toombs, that the gold dissolves at an accelerated rate during drying. There must, however, I think, be a balancing of errors, since I do not believe that all the soluble gold is recoverable from the drying vessel or that it can be completely washed out again after drying.

One is naturally placed in some difficulty in discussing the aspects of screen sampling and assaying presented by the Meyer and Charlton plant without having the special experience of dealing with these, but at the same time I think there is a tendency to overestimate the difficulties of assaying such material and to saddle the responsibility of discrepancies on the dissolved gold. Such variations in duplicate assays, as are mentioned by the author, are by no means confined to the Denny plants. In a paper I read before this Society on "Routine Assaying" I mentioned the method in use at the Robinson Deep of not taking the sample to complete dryness before quartering down. Mr. Toombs does not say whether the tub samples from each shift are mixed or separately assayed. This is important, as the method followed, if mixing is adopted, bears largely on the representative value of the assay, and if the shifts are assayed separately in quadruplicate, the number of results obtained tends to lessen the errors. In the former case the greatest attention must be paid to taking representative portions, my experience going to show that it is quite easy to go astray with a general tendency to high results, and I found that this tendency was to a great extent checked by working with moist material. Probably methylated spirits would be the ideal liquid for quartering down, but a proportion of moisture about equal to that of a well-drained sand-vat material is near enough.

The sum of my conclusions is that the errors in dealing with unwieldy samples, quartering and mixing, have often more practical bearing on the industrial value of the screen assay than errors introduced by such special conditions as obtain on the Meyer and Charlton, the general conditions being the same there as elsewhere along the reef wherever the gold is unevenly distributed and coarse.

I agree with Mr. Toombs that 2 A.T. charges are better than 4 and for the same reasons. Certainly 2 A.T. in quadruplicate will give more reliable data than 4 A.T. in duplicate.

Mr. H. A. White: The claim made by Messrs. G. A. and H. S. Denny for superior accuracy in the valuation of the ore going to the battery, on mines where the new metallurgy was installed, is not borne out by the valuable paper Mr. Toombs has presented to this Society. This adverse factor was one of those generally foreseen, and attention was called to it during the discussion on the paper read before the South African Association of Engineers, though it was not claimed that the objection was fatal. In fact, it was only necessary to instal suitable appliances for sampling the ore from the mill shoots, before any contamination with cyanide solutions takes place, to entirely obviate this objection.

There are some few points in connection with the screen sample method as practised on the Meyer and Charlton and the George Goch which Mr. Toomb's paper suggests on a careful perusal.

In the first place, is that sample properly taken?

The method described on p. 273 of the June, 1906, number of the *Journal of the South African Association of Engineers*, is faulty in one important respect. The vessel employed for taking the sample is said to be 3 in. deep, and no mention is made of a front plate of the height of the screen which should be employed to prevent loss of the lighter portion of the pulp which is generally of greater than average value and is accompanied by a greater proportion of solution. The low solution ratio shown may thus be merely due to errors of sampling and not to any real economy—as has been claimed elsewhere.

Mr. Chapman has opened the question of the cases where the value of solution is occasionally greater before than after the boxes, and pertinently asks what procedure is then adopted. I understand that in such cases an average is substituted for the negative difference. This method cannot be defended on statistical or metallurgical grounds. *The loss in value may be real and due to a temporary excess of ferrous sulphate in the ore* being crushed or merely apparent due to non-coincidence of variations in two different places of sampling. In either case the value of the ore must be reduced to correspond with the difference found in the solution values or the plant may be debited with the same gold twice over. Another point of great theoretical interest is the variation shown in duplicate assays. This defect is never entirely absent in screen samples, even when crushing with water and differences of 1 to 1½ dwt., though not frequent, are by no means unknown. A common explanation is the presence of a small quantity of gold much coarser than usual. That this explanation does not fit the present case is shown by the alternative method of thoroughly washing the sample when the results become consistent. We are, therefore, dealing with a special case, and the drying down of material with fairly coarse pyritic particles in the presence of rich gold bearing solution is the clue to the results observed.

It is clear that unequal redistribution of the gold may readily result under these conditions and possibly the presence or absence, in the sample actually fused, of a piece of specially enriched pyrites may seriously affect the result. The remedy proposed is two-fold. In the first place the drying down should be done under a vacuum at a fairly low temperature. In the second place the dried sample should be reground to pass a much finer sieve than usual and a look-out kept for metallics.

It would be interesting to members if Mr. Toombs would give us a comparison between the value of rock as determined by calculation from stope assays and tonnage measurements and the figures shown by his screen assays.

If the figures were given both before and after the introduction of cyanide into the mill water on this mine it would at once be seen what difference is required to balance the discrepancies incidental to a mistaken method of determining that important figure, the "Screen Assay Value."

Mr. E. J. Laschinger: Although not an assayer, I read this short paper with interest.

It is easily seen that the first formula and method given for obtaining the assay value of the screen sample cannot be correct, as it assumes that the solution left with the ore after decanting off the bulk of the solution has the same assay value as the decanted solution. This could only be true if, before decantation, the whole sample had been agitated until the solution had dissolved all the gold it could. This would probably be an unsatisfactory method to pursue.

Under the conditions of procedure given in the paper the correct screen assay value is given by the formula

$$z = q + \frac{W_1 x - W y}{w}$$

when z = the screen assay value sought
q = assay value of unwashed pulp
x = assay value of tub solution
y = assay value of hose solution
w = weight of dry ore sample
W_1 = weight of solution decanted off
W = weight of total solution.

This formula gives the same result as the corrected method proposed by Mr. Toombs, but seems to the writer simpler in form and involves less arithmetical work.

It is to be noted that the screen assay if conducted as indicated by Mr. Toombs is affected by more elements of uncertainty than the direct assay of ordinary screen samples. Even with the ore and water screen sample one can only hope to obtain a fair representative value by the exercise of care, by regularity in routine sampling and precautions against abnormal conditions at sampling time. The results of the assay of ore crushed in cyanide solution involves (if conducted as at the Meyer and Charlton) three assay determinations and three separate weighings to find the relative weight of ore, "tub solution" and "hose solution," so that the possible error in the final result is enhanced more than threefold above that of the ordinary assay.

The erratic assay results of the unwashed ore sample also indicate that the method of evaporating the solution is not good, the probable reason being irregular distribution of the soluble gold in the sample. In this particular the sample, even if carefully and thoroughly mixed, presents features analogous to an ore sample that has been "salted" by the injection of auric chloride solution. There are various methods of dealing with the problem of assaying samples containing soluble gold, which should present no great difficulties to expert assayers.

It is, of course, a question where crushing is done with cyanide solution whether proper arrangements for sampling the ore before coming into contact with the cyanide would not be much better than the screen sample. The assay is simple and direct, and can be depended on. With a sample much larger than the screen sample and averages taken over periods of, say, a month, the true value of the average ore should be very closely approximated to.

This matter of the reliability of the assay value of the ore milled is obviously of the greatest importance as indicating to the metallurgist the real performance of the plant as a whole. If the original value of the ore milled be given at too low a figure, the plant may apparently give most satisfactory results, and if at too high a figure, then the performance may appear very bad, and the real truth is not known in either case.

Assuming the figures given by Mr. Toombs as to the value of the ore by the two methods of assaying to be representative, the original method giving too low a figure, then instead of the recovery at the Meyer and Charlton being 94·8 per cent. as quoted, it would only be about 90·5 per cent.

MINE SUBSIDENCE.

(Read at March Meeting, 1907.)

By Alex. Richardson, A.I.M.M. (Member).

DISCUSSION.

Mr. T. Lane Carter: There are many interesting points in this paper, and I have only jotted down a few notes, so that I trust you will bear with me if my remarks appear disjointed. In the first place, Mr. Richardson speaks of the suddenness of mine subsidence. There are possible cases where subsidences are very sudden, but in the big subsidence with which I had to deal eighteen months ago and which he mentions in his paper, there was not this suddenness, in fact, for a number of days before the collapse came the pillars had commenced to sheer slightly, and some of the timbers gave warning that a good deal of pressure was on them. Then one morning it seemed as if a cyclone had broken loose in this disturbed area, and the whole zone, as shown in Mr. Richardson's paper, seemed to be moving quickly. Now, I think we did the right thing; if we had done otherwise I am afraid there would have been several men killed. Instead of trying to keep up the hanging wall, which in some instances might have been done, we simply ran away from the disturbed zone. The pillars were much shattered. We put barricades outside the zone of the subsidence, to prevent the ingress of men into the affected area.

The underground storm lasted for nearly a week, and there was a continual falling of rock at intervals. Then almost as suddenly as it came the subsidence seemed to ease off. In a few days we took down the barricades and went in without danger. We were working two reefs, the main reef and the south reef. The south reef pillars had all slacked off. To the inexperienced eye it would have appeared absolutely essential to timber up all the drives, and to either take out the pillars or protect them, but after the storm passed there was no further movement. That was eighteen months ago, and these pillars, which have "slacked off" from the hanging, are still standing.

I want to bring up one point. It is an *utter impossibility* to keep up a large area of ground when it is on the move. No supports in the world will stop a big movement of that kind, and I am sure *that the only thing* to do is to get out and let it come in.

There was another interesting point in the subsidence, namely, *the effect of the large* boundary pillar. We have an agreement with the Champ d'Or mine by which we leave a 10 ft. pillar each side of the boundary instead of the 30 ft. pillar required in the mining regulations. This boundary pillar seemed to act as a fulcrum. One morning the Champ d'Or men were driven out of their mine by a caving in of their stopes, and one life was lost in getting the men out. This boundary pillar acted as a fulcrum and caused a sheer at right angles to the strata from the lowest workings of the Champ d'Or to the surface. Ten or fifteen minutes after the caving had started on the Champ d'Or numerous cracks appeared on the surface. The sheer came along on the north side of the pillar, and none of the cracks extended to the French Rand property. We know this to be so, for when the rainy season came the Champ d'Or got all the surface water while we were not bothered with water at all.

Mr. Richardson also makes an important point—I think he might have dwelt on it a little more—in regard to the mining regulations respecting pillars underground to protect the surface. That law appears to me to be ineffective. The pillar as required by law is in the wrong place, and I would suggest that since there is a Commission just appointed to look into these regulations and report on them, the attention of the Commission should be drawn to the shortcomings of this law, and that the law should be revised so as to be more in keeping with the facts.

I believe we should put in timber in excavations rather frequently, to act as "tell tale" timbers. Any sort of timber will do for this work, and by carefully watching these props one can see what weight is coming on them, and can tell how the area of ground is behaving itself. Another point Mr. Richardson makes is in regard to the support of shafts. If your shafts are damaged by subsidence it is as if the artery in a man's body was cut. They are the main arteries of a mine, and should receive every possible protection and care. I agree with Mr. Richardson that our incline shafts should be in the foot wall. All mines cannot have their shafts in the foot wall all the way on account of faults, dykes, etc., which disturb the reefs. He believes in leaving pillars over the shaft and on each side when the shaft is in the foot wall. Provided the shaft is 40 ft. or so under the reef, I believe it a better policy to take out the pillars overhead altogether. I have seen a shaft which was under the reef and two large pillars were left, one at the bottom and one at the top of the stope, just over the shaft. A tremendous subsidence came and these two pillars gradually took all the weight of the disturbed zone. The weight was so great that the rock was broken right through to the shaft. If these pillars had been elsewhere, although the stope would probably have caved in, the shaft would have escaped without any damage.

There is another interesting point which was brought up by one of the critics at the last meeting, namely, in regard to keeping our mine "houses" in order, so that in years to come our descendants may return and take out the reef which is unpayable to-day and which is being left in the stopes. To leave a mine so that it will stand for years requires many more pillars. If the hanging wall is to be supported for a few months only, until the payable ore is extracted, not many pillars or timbers are required. If we are to leave our stopes so that men can come back in 100 years' time to mine what is now unpayable, it is going to be very expensive for us now. My view is that to-day is the appointed time to get this low grade stuff out, and hence the absolute necessity of getting working costs down so as to bring a larger tonnage into the pay zone. I notice Prof. Yates disagrees with Prof. Callon and others in the theory that subsidences are transmitted to the surface without sensible diminution in amount, irrespective of depth. I agree with Prof. Yates, and our experience on the Rand goes to prove that the reasoning of Prof. Callon and others is not applicable here. Of course, in different places we get different kinds of hanging wall, and I have seen stopes where the country rock is flexible, and such stopes move gradually for years, but there is scarcely any subsidence. The hanging wall comes down very slowly, crushing out the pillars, until the hanging wall and the foot wall practically meet. The subsidence was so gradual that it was never noticed at all, and there was no collapse. Much depends upon the character of the country rock.

In one big colliery in France they are now sending culm mixed with water into the old workings, and this becomes almost as solid as the original coal. The water runs out through an adit. In this way the hanging wall is protected to a great extent. We cannot use this scheme here, but if we had some method of being able to send sand and water down into the mine and handling the water cheaply afterwards, there might be something in the idea. My last remark is in regard to resueing. I think there is a lot to be said in favour of resueing in all mines. Very often it is the best method of working the present day stopes. Of course, it sends up your working costs, but I think what we are aiming at is maximum profit rather than lowest working costs.

Prof. J. Yates: You mentioned one case where you had a shaft in the foot wall. Did I understand you to say that the two pillars were forced through into the shaft when the subsidence happened?

Mr. T. L. Carter: Yes, practically, these pillars were the cause of the caving of the shaft.

Prof. J. Yates: You also mentioned the practice of sending culm underground. As a matter of fact, not only is culm used but sand is sent down ocassionally in some parts of Australia.

Mr. W. Reid Bell, M.Inst.C.E. (visitor): On reading Mr. Richardson's valuable paper, it strikes one whether the time has not come for such matters of general interest to be investigated systematically, that is to say, a small standing committee should be put in charge of the collection and arrangement of information relating to subsidence on the banket mines of the Rand. The mines might be invited to furnish through their surveyors accurate particulars of such matters, a survey of the subsidence and of all faults and throws in connection therewith, statistics as to the age of the workings affected, the strength and nature of the rock, and the history of the subsidence, and so on. In this way would it be possible to perfect our knowledge of the strength of the strata on the Rand, and to arrive at the best methods of attacking the deeper levels and the poorer ore bodies.

Prof. Yates has covered the ground for discussion so well that one can only offer remarks in amplification of the points touched by him.

The formulæ given in the paper appear to apply only to one class of rock, as it stands to reason that the angle and consequently the height of the dome of subsidence must depend on the strength of rock and character of the stratification as well on the dip.

In the case of some rocks of great strength when they settle, the beds and joints open, and their strength enables them to support the superincumbent weight for an indefinite time with only a partial bearing. The settlement of such rocks at a certain depth might not affect the surface at all.

On the other hand, soft rocks, after settling down, will in process of time yield to the superincumbent pressure and, the interstices closing up, the settlement at length reaches the surface.

It must be remembered that any movement of rocks that are solid must result in them occupying more space when the movement finishes than before it began, and the bearing surface of each bed upon the other must be diminished. If then the crushing strength of the rock be sufficiently great, the beds may be able to support the weight on the reduced bearing surface, and the disturbance might never reach the surface. If, on the other hand, the rock be weak, it must in its new position be crushed until the weight is sufficiently evenly distributed. In this case the disturbance would be felt in course of time at the surface, but in no case would the surface depression equal the original thickness of excavation.

Mr. Richardson, citing the case of the Geldenhuis Deep, states that the caving was sudden and argues that the disturbance would not reach the surface, but I am inclined to think that a sudden caving would be more likely to reach the surface than a gradual settlement in the case of these rocks. Had the subsidence given much warning, I would conclude after a certain lapse of time that the disturbance would never reach the surface, but with *sudden* caving it would argue that an empty cavern was developed in some part of the depth, and it would be impossible to say when the strata above this would drop in and let down the surface. On the other hand, the gradual subsidences in hard rock described by Mr. Richardson would certainly point to the probability that at a reasonable depth the effect of settlement will fail to reach the surface

I hardly think it fair to the Jumpers to describe as "sorting out" the process by which 20 per cent. of the rock broken down is obtained for pack, because "sorting out" implies that it is a sorting of rock broken out for the mill, and sorting of poor grade reef and stowing it underground means that there is no possibility of checking the sorting by re-sampling the waste, nor will the rock ever be available for re-treatment in the future. I know of one old waste dump within a mile of this meeting which assays 4 dwt., but if this were buried in the stopes underground, there would be no chance of recovering anything from it.

The notion of longwall working both for banket beds and for our thick coal seams is a very alluring one, and it should never be lost sight of, however far off it appears just now.

The difficulty that would be met with in the banket is that these Archæan rocks are not like the coal measures comparatively undisturbed, but bear the scars of age in their faults, joints and backs. The irregularity of these joints in conjunction with the density and tenacity of the measures would make the weight in a longwall come on very irregularly, and to this difficulty must be added the intensity of pressure, on the working face (if the working be to the dip) occasioned by the steepness of the dip. This pressure would act in opposition to the draw of the strata, but if the working were to the rise, the draw and pressure due to the dip would be additive and might assist the winning of the rock as well as the bringing down of the roof in rear of the face. Possibly rise workings would be the only way of working a longwall in banket.

Longwall working has been adapted to so many different classes of seams and minerals that there appears to be hardly any mines to which it cannot in some form be applied. The length of face must be very carefully adjusted to the nature of the seam, the roof and the floor, due regard being had to the condition in which the mineral is required to be won. The main thing in starting a longwall is to get the first weight on, and sometimes this is a phenomenon little short of terrifying to miners accustomed only to other methods of working, but once the shaft pillars and the face have been secured and the roof comes down nicely in rear of the face there is less risk of irregular surface disturbance than with other methods, especially if the face progresses steadily.

One of the difficulties in regard to the introduction of longwall in the thick coals in the Transvaal is the cost of timber. Timber can be employed in the thin banket seams, but in the coal mines the quantity required and the low market price of the product prohibits the use of timber to any great extent, and the consequence is that extravagant methods of working have perforce to be perpetuated, although it is no reason why colliery engineers should relax their efforts to improve upon the methods in vogue, and here I would remind young engineers that where there's a will there's a way to be found some day.

It is at present an object with engineers mining the Rand reefs to keep up the roof as far as possible, and to leave pillars for that purpose when packs cannot be made, but in the first place the hanging wall cannot be permanently and uniformly supported in this manner, and secondly, as the depth becomes greater, it will be found impracticable to work the poorer ore bodies unless the getting of the rock has been assisted by the settlement. That is to say, in working the poorer reefs at depth, it will be necessary to cause settlement of the strata in the preliminary workings.

The discussion on this paper so far emphasises the fact that pillars and packs are too often the cause of irregular settlement, and one of the great points in longwall methods is that it acknowledges the difficulty of preventing settlement, and instead of resisting settlement its success depends upon bringing down the strata as the mineral is worked out. If then the matter of irregular settlement be taken into account in working superposed beds in which the value of the mineral forbids the abandonment of pillars of excessive size we are compelled to fall back upon longwall working. In this country, as stated before, the great hindrance to the employment of longwall systems is the scarcity of pit-timber, but stone packs and steel props are two alternatives, while necessity may perhaps show us others. For instance, the existence of two or more beds of mineral near to one another should offer, where timber is not available, an ideal case for the employment of longwall, while after the upper bed and its hanging wall above it have settled down on the stopes below, the getting of the upper bed should be a comparatively cheap proposition and the weight should come on the new workings with great regularity and certainty. Whether the second workings should follow the first quickly or not is a matter which experience would determine.

The arguments regarding the influence of the strength of the rocks upon the method adopted in mining refer to the question of keeping the workings open by leaving pillars. If, however, it is not attempted to keep the roof up, but, in accordance with the best practice of mining (where applicable), the roof be brought down, the influence of the strength of the strata assumes a different aspect. Here we meet with another question which is a serious one in connection with longwall working in beds of varying value. The system of longwork adopted in banket mines will have to take into account the existence of poor shoots (not chutes nor shutes, if you please) of ore which may be unworkable at the time the main ore body is taken out.

Now let us consider what we mean by crushing strength of rock. A cube of rock under pressure with free faces will yield and the pressure gradient, let us call it, being too steep from the mass of the cube to the free face, the cohesion of the particles is overcome and instead of flowing like lead or a liquid the cube is disintegrated and, as sand, occupies a greater space than before.

But where the rock is confined on all sides it is obvious that the pressure will not result in the rock taking up more space, nor can the pressure gradient be so steep. If the rock be subjected to pressures sufficiently great, and if there be any variation of pressure it will result in a flow of the rock which will be shown in certain mineralogical features well known to geologists.

Now certain rocks, let us say sandstones and quartzites, are known to have been subjected to pressures of many thousands of feet of strata in certain localities and to have been subjected to even pressures and to varying pressures, and have afterwards been raised to our view.

If then the pressures which the rock had sustained were approaching the point at which the mineralogical structure, as we know it, were threatened with breakdown, we would expect the phenomena of flow to be very common indeed in these rocks.

But are these phenomena common in all rocks which have been subjected to such conditions as above described? I would look to our geologists to throw some light upon the subject, and I think it will be found that the limitation to mining at depth will be due to other causes and not to the breakdown of the rock structure. And further, if we could reach such critical depths the result of mining operations would probably result in the generation of such heat in the rock mass that the workings would have to be abandoned from that cause alone. Of course, these considerations apply to cold rocks, such as the main reef series. In coal, critical depths have been reached, as in one case I know of in the South Lancashire field, where the pillars under a railway, when in process of being robbed, fired under a pressure of some 4,000 ft., and the whole was lost. In this case, however, it is obvious that the mere working of the seam did not produce the effect, seeing that the mine was safely opened up and worked; the abnormal heating was, no doubt, due to the extra weight thrown upon the pillars and to the time that elapsed between the opening of the mine and the robbing of the pillars. At Home we hear a great deal about iniquitous mining royalties, but there is no doubt that owners of royalties exercise a most useful function in checking wasteful methods of mining, and some such check is sorely needed in the Transvaal coal mines.

The subsidence in the Cheshire salt districts has been touched upon, and as it is an interesting phenomenon a personal reminiscence of it may not be out of place, though my acquaintance with the district is of thirty years ago. There are two beds of rock salt worked in Cheshire, the lower bed mined in the usual manner while the upper bed is not mined, but the drainage of the geological basin runs over the surface of the salt bed and is pumped up through bore wells and evaporated in large iron pans. The pumping of the underground brine causes the water to flow in definite channels over the underground surface of the rock salt, and the subsidence of the strata over these channels causes the surface of the ground to sink directly over the underground channel. These sinkings of the surface are called "brine runs" and they appear for all the world as the natural bed of streams, a foot or perhaps two deep and 10 to 20 ft. wide or so. Their course can be traced for long distances across roads, streets, fields and under houses, one half of which sinks down and the other stands in its original position, or the whole house may lean several feet out of plumb against its neighbour. Besides these "runs" the whole area whence the brine is being pumped is sinking so much so that streets, houses, bridges, etc., have constantly to be raised and underpinned. Further, in connection with the river Weaver are large lakes or meres many acres in extent, called "flashes," some of which have been brought into existence in a night by sudden subsidence, and the beds, or at least the shores, of all of them are constantly subsiding.

Mr. J. F. Whitton: There are various points of interest but briefly touched on by Mr. Richardson in his interesting and instructive paper, and opening up as it does, a large field for additional information, future contributions may, no doubt, be looked for. Many of these points have not in the past received the attention which they merit, owing, no doubt, to their not having interfered to any appreciable extent with the regular working of the mines, and to the prevailing disinclination to introduce any change which might increase expenses. The importance of keeping stope footwalls as dry as possible cannot be over estimated, as observation has shown than when water is present the hanging walls of underlying stopes are in time affected in a very marked degree. These hanging walls may to all appearances seem perfectly sound and may show no signs of water, but it is persistently working its way into the bedding planes, gradually destroying cohesion and eventually causing collapse. I have seen many instances of this in outcrop mines in which much water is usually met with and where, therefore, disintegration and subsidence are not uncommon. In such cases careful examination of the hanging walls is imperative, and stack timbering with waste filling will insure comparative safety not only to that particular stope but also to the other workings in the vicinity.

The possibility of increased grade at the battery through underground sorting and resueing has not, up till now, been properly investigated. It would seem as if the penny-wise-and-pound-foolish policy of low working costs, through the heavens fall, were more of a real bogey than a fancied one, as it has often stood in the way of a trial of methods without which many of our pressing problems can never be solved. Where either or both of these methods are adopted, a safe increase of profit per ton can be secured in spite of an increased cost. At the Jumpers, where extensive resueing and underground sorting is practised, the beneficial results are clearly illustrated by the following figures :—

With surface sorting only.

Sorting	18 per cent.
Recovery per ton ...	32s. 11d.
Total working cost per ton	23s. 9d.
Profit per ton ..	8s. 2d.

With surface sorting, underground sorting and resueing.

Surface sorting ...	18 per cent.
Underground sorting ...	20 per cent.
Recovery per ton ...	44s. 7d.
Total working cost per ton	30s. 9d.
Profit per ton ...	13s. 10d.

So that although the working costs are increased by 7s. per ton by the adoption of underground sorting and resueing the ultimate profit is increased by 5s. 8d. per ton. In resueing the common impression is that much of the finer portion of broken reef is lost by becoming mixed with the sorted out and stowed away waste. That some is lost is undeniable, but where a stope is worked by men skilled in this class of work the loss is inappreciable, and practically nothing but pure reef is shovelled and trammed away.

Resueing can only be practised to any advantage in hammer stopes, and the reef must be carried either on the hanging or foot, in order that the waste necessary to make stoping width may be broken out clean. In a 4 ft. stope with a well defined bedding plane between reef and waste a reef width up to 2 ft. 6 in. can be well resued, beyond that, however, the holes in the waste are apt to break into the reef. Much depends on the position of the reef, a reef on the foot being overcut further and better ; such overcutting being limited only by the amount of waste to be cut out and the facilities for getting at the face. A great advantage lies in all the reef being taken out as the waste filling takes the place of pillars ; in stopes of high grade this is a very important feature, and where the low grade of a stope prohibits much timber, waste sorting and filling, leaving clean reef to be taken to the mill, will do much to increase its payability. Excellent ventilation is secured, for with waste packed to within 4 or 5 ft. of the face a passage is formed and the resulting brisk current of air clears out the smoke and gas after blasting, in a few minutes.

Back stoping has been found most suitable where the dip exceeds 40 deg., while with flatter dips breast stoping is preferred, as packing and shovelling is more easily dealt with. Where resueing is not practised and only sorting carried out the waste is packed not only inside stacks but also between them, thus forming a continuous support which allows of a very few pillars being left *in situ*, and then only when the hanging is bad ; with a strong hanging, pillars may be dispensed with altogether. Owing to its cost, stack timbering on the Rand has not been given much attention, but I feel sure should the working of a stope, considering its grade, be carefully studied, stack timbering and waste filling, where practicable, would be generally adopted instead of the present practice of leaving numerous solid pillars and shovelling out all the waste with the reef.

The cost of a composite pack or pigsty composed of round timber 6 in. to 8 in. diameter, 6 ft. by 6 ft. outside and 5 ft. by 5 ft. inside dimensions, including the cost of timber, labour in building, filling in with waste, etc., figures out at about 10s. per foot of height. Suppose for the sake of argument that in a 4 ft. stope going 10 dwt. over the stoping width one 12 by 12 is replaced by four 6 by 6 stacks. The value of the pillar is about £95 or, say, £75 when broken and trammed, the four stacks would cost £8, and many cases have shown that pillars left *in situ* and considered as so much reef to be won some day are through subsidence completely buried up and cease to be an asset ; for this reason the complete mining of the reef from the outset would appear to be the best and most economical method

NOTES ON SOME RECENT IMPROVEMENTS IN TUBE MILL PRACTICE.

By KENNETH L. GRAHAM, M.I.M.M. (Member).

DISCUSSION.

Mr. R. G. Bevington: *Efficiency of Pebbles v. Banket.*—My experience shows that there is practically nothing to choose between the two, taking as a basis the grading of pulp passing on to the cyanide works, the screen used in the mill being the same, viz., 300 mesh, having an average aperture of ·037 in.

The following tables will be of interest with regard to the above:—

	Proportions of Pebbles to Banket		Pulp passing on to Cy		Works per cent.
	Pebbles	Banket	+60	+90	−90
Dec., 06	$\frac{1}{4}$	$\frac{3}{4}$	3·13	9·67	86·62
Jan., 07	$\frac{1}{9}$	$\frac{8}{9}$	1·85	10·21	87·30
Feb.	$\frac{1}{24}$	$\frac{23}{24}$	2·66	13·77	83·08
March	$\frac{1}{28}$	$\frac{27}{28}$	1·82	11·96	85·81
April	$\frac{1}{101}$	$\frac{100}{101}$	3·00	16·01	80·60
May to 15	nil	banket at rate of 23 tons per day.	3·76	12·27	83·97

There is also the additional advantage of the gold recovered from the absolutely clean sorted reef which is used, the expense of sorting, tramming and feeding which is, as Mr. Graham states, but small as against the enormous cost of pebbles of an absolutely barren nature, as far as gold contents are concerned.

I think with Mr. Graham that $1\frac{1}{8}$ in. is the most suitable diameter of aperture of nozzle for feed. I find that this, using in the mill a 300 mesh screen having apertures of ·037 in. passes 240 tons of pulp per day of a consistency of solids 53·2 per cent., liquid 46·8 per cent.

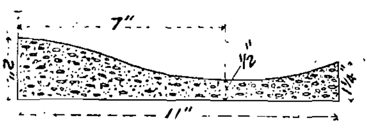
I agree with Mr. Graham that the banket does not appear to be so destructive to silex lining as pebbles.

Two mills were started almost at the same time, both being lined with silex blocks 6 in. deep. One mill ran 140·73 days of 24 hours and the other 143·88 days of 24 hours. These mills were fed during this time with pebbles and banket in the following proportion, viz., $\frac{1}{6}$ pebbles, $\frac{5}{6}$ banket. The run of these mills corroborates Mr. Graham's experience.

I am now, on account of the undoubted fact that the lining at about 2 ft. 6 in. to 3 ft. from the feed end of the mill wears out much sooner than the remainder of the mill, lining the mills with silex 8 in. thick for 7 ft. from feed end: then $7\frac{1}{2}$ in. for 1 ft.: and 7 in. for 1 ft. 6 in. for the remainder. As no mill lined under these conditions has yet worn out, I cannot state the life of liners, but shall be pleased to communicate same as soon as I have the necessary data, but I look for the whole of the lining of the mill to be worn out at practically the same time, thus avoiding a good deal of the waste which must occur when using a lining of uniform thickness throughout.

With regard to lining other than silex I may mention that I tried a lining of local chert which appeared a very good hard stone, but on account of the very rough manner in which the stone was dressed it was impossible to make good joints between the setts, and at the end of a run of 55·89 days of 24 hours it was found that a portion of the lining had fallen out, rendering it impossible to continue the running of the mill any longer. Had the setts been well dressed so as to key properly, judging from the thickness of those remaining in the mill, I think this lining should have run for some 90 days.

Local granite setts, 11 in. x 8 in. x 6 in., of a very hard nature of stone, said to be the second hardest granite in the world, were tried. These wore through at about 2 ft. 6 in. from the feed end of the mill in 24·38 days of 24 hours. With regard to the wear of linings in corrugations it was very noticeable that in the above long granite liners the corrugations occurred whether at the joint or not. I have some stones showing the greatest thickness left at the joint itself, and the thinnest point being almost in the centre of the stone, thus



the part at A is some 2 in. thick, while at B it is only $\frac{1}{2}$ in., and the distance from A to B is about 7 in., this appeared to occur with fair regularity throughout this mill, the greatest thickness being at times at a joint and the least thickness being also at times at a joint.

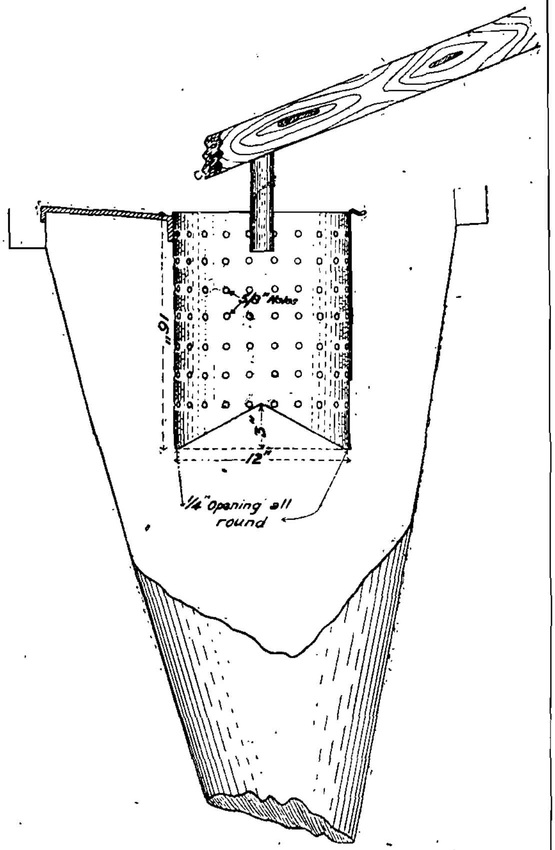
I find that three mills working regularly and taking some 720 tons of pulp per 24 hours from a 160 stamp mill with the before mentioned 300 mesh screening require to be fed with a daily aggregate of 23 tons of banket to keep them full to the most efficient point, that is to say, just to the back flow point.

I quite think with Mr. Graham that the admission of a little water at the bottom of the spitzkasten would give a better product for tubing, it is only rational to presume so, but so far as I can gather the great thing impressed upon a tube mill man hitherto has been, not on any account to use any extra water, in other words, do not use your spitzkasten as a hydraulic classifier.

I may here mention a device which I have had in use from the commencement of tubing on the Village Main Reef for preventing as far as possible the choking of the nozzles of the dewatering cones with small pebbles of quartz and other matter which may inadvertently pass the screens placed before the spitzkasten, as it often happens that a long but narrow shaped piece of rock, or whatever may be, which would not pass this screen if presented at its greatest surface, will slip through if it comes on end wise, and ten to one when it gets to the $1\frac{1}{8}$ nozzle it will perhaps out of sheer perversity present itself broadwise to the opening with the well-known result.

The device is simply as follows:—

At the discharge of the down pipe from the launders into the dewaterer cone I place an inverted cylindrical hood, fashioned thus:—



This I find intercepts a great deal of the matter which goes towards the choking of nozzles, and being fitted at one side with a sliding door, also perforated, reaching to within about 4 in. from the bottom, one can, while all is running, slide up the door, put one's hand in and clear out any matter which may have collected, and slide the door down again.

Mr. F. F. Alexander: I am naturally much interested in Mr. K. L. Graham's paper, since particularly so the improvements alluded to were first made by myself on the Crown Deep in 1906, where I invented a mechanical device for feeding a sufficient quantity of reef into the tube mills, and since September last I have used nothing but reef to crush the pulp from 300 stamps.

There is, no doubt, that selected ore from the sorting tables will do the work quite as efficiently and just a little better than any pebbles I know of, and Mr. Graham's theory, that of impact on the matter is quite a reasonable one. It is reassuring to learn from Mr. Graham's experimental run of 81 days with ore versus pebbles, that the results obtained by him are practically identical with similar experiments I made in September last in one of the mills under my charge (Allis Chalmers). Although my trial run was of shorter duration, I found the all quartz gave slightly better results than all pebbles, the test being made in the same mill, with this difference in my case, that I fed 6·5 tons of ore per day of 24 hours as against Mr. Graham's 2·1 tons and a little over 2 tons of Danish pebbles as against 250 lb. in Mr. Graham's case. After satisfying myself by practical test, which I am pleased to find so thoroughly corroborated in Mr. Graham's paper, I discontinued the use of pebbles and have since October, 1906, to March, 1907, inclusive, crushed 3,400 tons of reef in three tube mills. During the above period I claim the saving effected by the use of reef versus pebbles in three tube mills at the Crown Deep to be over £3,000.

I prefer to leave further computation on this matter to those who care to theorise on this subject, and state the following facts. These mills would have consumed 540 tons of Danish pebbles: they actually crushed 3,400 tons of reef and gave excellent results.

I must disagree with Mr. Graham on the question of silex liners being the best wearing material for lining a tube mill. Our local chert is, in my opinion, quite as good, and I feel confident will ultimately prove better than imported silex, both in cost and wear. As in all new industries, there is always something to learn and few to teach; our stone dressers have not yet learned to select the most perfect blocks and how best to shape and dress them.

The silex blocks are usually well shaped and lie closer when built into the mill than the local product, they are more homogeneous and not quite so brittle, whereas the local chert is of closer grain, very brittle and less homogeneous, particularly near the surface, and unless closely joined will yield to the impact of a heavy load, splinter and crush at the joints. The cement setting, being the softest part of a liner, will, unless the stones are closely laid, wear away and expose the edges to the impact of the load, the mill becomes extremely rough, and the load pitches, more than is necessary for good grinding. This theory is proved in practice on the ends of a mill where the local chert outlives the imported silex. These ends are built up on heavy cast-iron rings bolted to the mill with four $1\frac{1}{4}$ in. bolts, countersunk heads The rings are made in half sections and taper against the mill, the stone is dressed to suit this taper and built up off the ring in two blocks, the top block being wedged against the shell of the mill wooden wedges, with well staggered joints and ordinary cement setting. I have had no trouble with these ends coming loose or falling out, and find they wear better than silex. I am therefore satisfied that the local flint is harder and will stand more wear by attrition, and with close fitting joints should easily take the place of the imported article.

Mr. Graham strikes a note of much importance in his reference to the quantity of pulp and ratio of water fed. This question is governed by two principal factors, first, coarseness of pulp, second, head pressure and nozzle velocity of feed; the first being, of course, governed by the battery screening and nature of rock crushed. With 200 mesh on the battery a tube mill, 22 ft. x 5 ft., will reduce 200 tons of pulp per day to a satisfactory degree of fineness with an equal quantity of water to solids, the +60 being about 65 per cent. and varying as the quantity of pyrites is to coarse sand by specific gravity. The second factor trespasses somewhat on the domain of our mechanical engineer, and is governed by the dewatering cone separator both in size and construction; a separator 6 ft. deep tapering at 60 deg. to $2\frac{1}{2}$ in. with right angle **T** piece and 18 in. to 24 in. of parallel pipe leading to the 1 in. nozzle is a very faulty device for adequately feeding the mill under above conditions, and I am sure considerable improvement can be made at this point to increase both the efficiency and life of our mills, and I would suggest that in place of the present arrangement the dewaterer be either made deeper and larger, or set up closer to the mill, in fact do both, and thereby allow, if required, a feed of 6 to 4 sand to water with a minimum of 250 tons of solids to each mill per day. Pipe friction must be avoided at all cost where we require one of sand to one of water forced through a 1 in. nozzle with only 6 ft. of head pressure behind it. I know that by feeding heavily and thick the liners are preserved from violent impact, and for this reason, if for no other, the peripheral discharge becomes impossible in practice for fine grinding.

With diamondiferous deposits where quantity is most essential, I believe the peripheral discharge is capable of passing a much larger tonnage through the mill than is possible with the central discharge.

Mr. Graham's expeditious relining of a Davidsen mill is indeed a very creditable performance, but I fail to find any economy shown, as against hanging up one mill out of two or three for 96 hours, taking 48 hours to do the work, and 48 hours for ordinary cement to set up fairly well, the cost being about one-quarter that of diamond cement: change the battery screen and calculate the loss as deferred loss of tons milled.

Cyanide men, with tube mills at their disposal, have evidently forgotten the old cry of amalgam in the sand, and I have not seen any mention, in computing the work done by tube mills, of this enemy to cyanide treatment. With one mill at work, practically all loosened pieces of amalgam and gold contaminated with mercury are easily caught on the shaking tables, the sickened mercury becomes lively in passing through a tube mill in which so much friction takes place as to raise the temperature of pulp from about 70 deg. to 90 deg. F., and amalgamation under above conditions appears to be most perfect. I mention this point as of some importance where one or more mills are in commission.

Mr. E. J. Laschinger: I notice that Mr. Graham says that Mr. Laschinger and others *recommended* the peripheral discharge as a means of increasing efficiency and reducing wear and tear of liners.

Now, I was purposely careful when discussing Mr. Dowling's paper on tube mill practice as to the wording of the supposed recommendation of peripheral discharge. What was said (p. 371, June, 1906, *Journal*) is as follows:—

"There would, therefore, be two ways in which the efficiency of wet grinding might be increased and the life of liners prolonged. (1) By introducing as little water as possible with the feed; and (2) by allowing any surplus water introduced into the mill to drain out quickly, thus leaving the particles to be ground as dry as possible. This second method of increasing efficiency might be attained by having the discharge plate or screen of the tube mill in the shape of a disc and perforated all over its area, so that the water could drain out as thoroughly and quickly as possible. I believe that an advantage has been claimed for the peripheral discharge as against the central discharge opening. The mechanical details for such a discharge plate, as I have indicated, should present no great difficulties."

It is evident that these words give a suggestion but do not constitute a claim or a recommendation as to the superiority of peripheral over central discharge.

I also do not know that it has been conclusively proven that the central discharge is superior to the peripheral or whole area discharge, and it would require carefully conducted experiments, unhindered by factors which often limit their scope in an existing working plant, to settle this point."

Mr. H. A. White: "I have no hesitation in accepting the responsibility for recommending the peripheral discharge, but I quite agree with Mr. Laschinger that the question has not yet been thrashed out, though it is clear theoretically that there should be a great advantage in it.

The meeting then closed.

Contributions and Correspondence.

NITROMETER WITH WATER JACKET.

A nitrometer fitted with a water jacket, through which a constant stream of water, whose temperature can be ascertained by an immersed thermometer, is allowed to flow, and which can be finally adjusted by means of a vertical screw attached to the clamp, possesses the following advantages when used for daily volumetric work:—

(*a*) Time saved, as the nitrometer and contents assume the temperature of the water in the space of 20 minutes.

(*b*) That once the temperature of the water has been assumed, it remains constant. The temperature of the gas is taken as that of the water.

(*c*) That thereby all errors arising from draughts of air (hot or cold always so prevalent in laboratories) are obviated, and that on this account the final adjustment by means of the bubble becomes much simplified, no sudden contraction or expansion of the gas being possible.

(*d*) That the apparatus can be handled with impunity, and without fear of raising the temperature by touching with the hands, by the heat of the body, or the warmth of the breath.

(*e*) That being measured in the same vessel as it is generated in, the volume of gas is not liable to the error of loss or gain by being transferred to any other vessel for measurement.

(*f*) That the whole experiment can be conducted successfully and accurately within the space of half an hour.

(*g*) That the apparatus is light, strong, durable, easily handled and quite inexpensive.

G. F. Ayers.

Dynamite Factory,
Modderfontein.

Obituary.

The death is recorded with much regret of Mr. W. L. Bain (Member), who died suddenly of apoplexy off Aden, when on a trip Home.

Mr. Bain was elected a member in 1904.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

New Method for the Preparation of Hydrogen Sulphide.—"The common method for the preparation or hydrogen sulphide by the interaction of hydrochloric acid and ferrous sulphide is open to several objections, the most serious of which is the liability of the gas to contain the hydrides of arsenic and phosphorus, due to the contamination of the iron sulphide with phosphide and arsenide. The method described ensures freedom from these impurities, as well as the absence of most of the unpleasant accompaniments of the ordinary process of preparation. If a current of hydrogen sulphide is passed over an alkali hydroxide, for example, sodium or calcium hydroxides, a hydrosulphide is formed, heat being evolved and a change in colour from white to yellow taking place. If a current of carbon dioxide be now passed over the hydrosulphide, the compound is at once decomposed, a carbonate being formed and hydrogen sulphide set free. In the case of calcium hydroxide, the yellow substance has the composition $Ca(HS)(OH)$, and the action of carbon dioxide is given by the equation:— $Ca(HS)(OH) + CO_2 = CaCO_3 + H_2S$. A drying tower is filled either with granular soda-lime or an intimate mixture of calcium hydroxide and moist sawdust. A stream of hydrogen sulphide is passed into the tower until the contents are saturated with the gas, the course of the action being easily determined by the change in colour and by the movement of a heated zone upwards through the tower. After the action has proceeded for some time, the gas which escapes from the exit tube smells strongly of hydrogen phosphide, and is inflammable, doubtless consisting of hydrogen mixed with all the other gaseous impurities evolved by the action of acid on the ferrous sulphide. When the hydroxide has reacted with as much hydrogen sulphide as possible, a carbon dioxide generator is substituted for that supplying hydrogen sulphide, and the apparatus is ready to yield as slow or rapid a supply of the latter gas as may be desired. The supply of hydrogen sulphide ceases at once when the stream of carbon dioxide is cut off, and if the exit tube from the tower be closed, a partial vacuum will soon be produced, owing to the combination of the gases in the tower with unaltered hydroxide. Owing to this absorption of the gas in the tower, it is only necessary to cut off the supply of carbon dioxide, in order to prevent further escape of hydrogen sulphide. The rate of supply of the gas may be nicely regulated by introducing a short length of capillary glass tubing between the carbon dioxide apparatus and the generating tower. The danger of contamination of the hydrogen sulphide with carbon dioxide is very small, and may be entirely eliminated by using two towers. As soon as the first shows signs of exhaustion (indicated when calcium hydroxide is used, by the change in colour) it is replaced by the second tower; a new one is put in in place of the latter, and the supply of gas taken from this. Prof. Meldola referred to the known laboratory process of preparing hydrogen sulphide by heating sulphur with certain hydrocarbons, such as solid paraffin."—F. R. L. Wilson.—*Journal of the Society of Chemical Industry*, March 15, 1907, p. 224. (A. W.)

Analysis of Minerals containing Zinc.—"The following process has proved successful for the estimation of arsenic, antimony, lead, copper, cadmium, iron, aluminium, manganese, zinc, calcium, and magnesium in zinc ores, such as blendes and calamines, although not applicable if much phosphorus or fluorine be present. The hydrochloric acid solution is precipitated hot by sulphuretted hydrogen and the precipitate extracted with sodium sulphide.

Sodium Sulphide Filtrate.—This is precipitated by dilute hydrochloric acid, and the precipitate washed with water, alcohol, and then a mixture of equal parts of alcohol, ether and carbon bisulphide to got rid of sulphur. It is washed off the filter with alcohol, evaporated to dryness, and treated with nitric acid (sp. gr., 1·4) in a covered dish. The residue is extracted successively with water and hot aqua regia, and then to the whole solution are added tartaric acid and ammonia in excess to precipitate the arsenic as ammonium-magnesium arsenate, which is weighed on a tared filter, after drying over sulphuric acid. The mother-liquor contains about 1 mgrm. of the double arsenate per 20 c.c. The antimony is precipitated in acid solution by sulphuretted hydrogen, and weighed as sulphide, the trace of arsenious sulphide present being allowed for.

Residue.—This may contain lead, copper, and cadmium. It is treated with nitric acid, excess of the acid removed by heat, and the lead separated by a little sulphuric acid and alcohol. The filtrate is evaporated to complete dryness, the residue taken up by a little nitric acid, and diluted so as to contain 3·4 per cent. of free nitric acid; from this solution copper is separated by electrolysis. The liquid is then evaporated to dryness with a little sulphuric acid, and the residual cadmium sulphate, which may contain traces of lead, copper, and zinc, is weighed.

Acid Filtrate.—Sulphuretted hydrogen is removed by boiling, and then iron, manganese, and aluminium are precipitated by hydrogen peroxide and ammonia. The precipitate is dissolved in hydrochloric acid, the solution boiled, and iron and aluminium are thrown down with ammonium succinate and sodium acetate from a hot and nearly neutral solution. After twenty hours the whole is reheated to 90 deg. C., cooled, filtered, and the precipitate washed with water and with dilute ammonia. The precipitate is ignited and weighed, and the aluminium separated by fusion with caustic potash in a silver crucible, or the iron is estimated volumetrically. The manganese is weighed as Mn_3O_4 after precipitation by hydrogen peroxide and ammonia. After separating the three preceding metals, the liquid is boiled to drive off the excess of ammonia and precipitate the last traces of aluminium. After cooling, a little ammonia is added, and the zinc precipitated by sulphuretted hydrogen, and weighed as sulphide. Calcium is then weighed as sulphate, after precipitation as oxalate, and magnesium as the pyrophosphate in the usual way.—J. A. Muller.—*Journal of the Society of Chemical Industry,* March 15, 1907, p. 205. (A. W.)

Volumetric Determination of Chromium in Steel.—"Either 1·667 or 3·334 gm. of steel turnings are dissolved in 25—30 c.c. of hydrochloric acid (sp. gr. = 1·124) in a 150 c.c. glass beaker. When solution is complete, 20—30 c.c. of an 8 per cent. solution of potassium chlorate are added for the purpose of oxidising the iron. The solution is then boiled until the last trace of free chlorine has disappeared. After the solution has been neutralised in a large Erlenmeyer flask with a 40 per cent. solution of sodium carbonate, the chromium is oxidised with 10—20 c.c. of a 4 per cent. solution of potassium permanganate. The entire solution is boiled for ten to fifteen minutes, with constant stirring. If the liquid is rose-coloured, from 2—5 c.c. of ethyl alcohol are added; and the solution is again boiled for ten minutes. When perfectly cool, the contents of the flask are transferred to a measuring flask, and are diluted to exactly 500 c.c. with cold distilled water, being frequently shaken during the operation. The solution is now poured upon a dry filter. For the determination of chromium, 300 c.c. of the filtrate (corresponding to 1 gm. or to 2 gm., according to the amount of steel originally weighed out) are employed. This sample is divided into six equal parts of 50 c.c. each, which are slightly acidified with dilute hydrochloric acid. To each part is added 5—10 c.c. of a 10 per cent. solution of potassium iodide, then, after shaking, 10—20 c.c. (carefully measured) of $N/10$ sodium thiosulphate solution, and, finally, 5 c.c. of a solution of starch paste. A $N/10$ solution of iodine is now added drop by drop from a burette until the faintest permanent blue colour is obtained. From the number of c.c. of thiosulphate solution taken is substracted the number of c.c. of iodine solution employed. The percentage of chromium is calculated by multiplying this difference by the factor, 0·17362, and multiplying the product by 6. This result must be divided by 2 in case 3·334 gm. instead of 1·667 gm. of steel were taken at the outset."—G. V. Bianchetti.—*Journal of the Society of Chemical Industry*, March 15, p. 205. (A. W.)

Assay of Silver Bullion by Volhard's Method.—"The Volhard method for the assay of silver bullion has been used by the author for several years past for some 200 assays per diem, with entire satisfaction, certain modifications being introduced which add to the accuracy attainable. An ammonium thiocyanate solution is prepared of such strength that 100 c.c. are equivalent to about 1·0003 gm. to 1·0005 gm. of silver. To a solution containing 1 gm. of pure silver, deprived of all traces of nitrous acid, and after addition of the ferric indicator solution (iron alum decolorised by nitric acid), 100 c.c. of the thiocyanate solution are added; when the precipitate that forms are settled, a clear, pale-red coloured solution (due to the presence of ferric thiocyanate) is left, which becomes now the standard of comparison to which all assays are worked. The addition of 0·5 c.c. of $N/10$ silver nitrate solution should render the solution colourless. The standard or check being thus obtained, in assaying silver bullion a solution is formed in like manner from about 1 gm. of silver, and after precipitation (in presence of the ferric indicator), by the normal thiocyanate solution, $N/10$ thiocyanate solution is added until the colour approaches that of the check, and after clearing, the solution is added by drops, until the colour of the check is exactly reached. One drop of the $N/10$ solution (0·05 c.c.) is equivalent to 0·05 mgm. of silver, and makes a perceptible difference in colour. Thus, it is the extreme delicacy of the reaction between the thiocyanate and the ferric salt which is taken advantage of in the described modified process, making it possible to see the end of the assay more accurately than when the assay is finished in the usual way. The assays are best compared with the check by placing the bottles so that they catch the reflected light of a white surface. In assaying silver-copper alloys, a pure silver check is used, with addition of copper in proportion approximately equivalent to that in the alloy."—E. A. SMITH.—*Journal of the Society of Chemical Industry*, March 15, 1907, p. 205. (A. W.)

DENSITIES OF THE ALKALI METALS. — "The densities and atomic volumes of the five alkali metals at 20 deg. were found as follows, many preparations being used, and great care being exercised in all cases except that of caesium.

	Densities.	Atomic weights.	Atomic volumes (Ag=107·93).
Lithium ...	0·534	7·0	13·1
Sodium ...	0·9712	23·008	23·70
Potassium ...	0·8621	39·114	45·38
Rubidium ...	1·532	85·48	55·8
Caesium ...	1·87	133·8	71·0

The density of an alloy of 39·9 per cent. of potassium and 60·1 per cent. of sodium was found to be 0·919."—T. W. RICHARDS and F. N. BRINK.—*Journal of the American Chemical Society*, Vol. xxix., No. 2, p. 127, Feb., 1907. (J. A. W.)

A SYSTEM OF QUALITATIVE ANALYSIS FOR THE COMMON ELEMENTS.—"This is the first of a series of investigations from the Chemical Laboratory of the celebrated Massachusetts Institute of Technology, and the outline of the investigation is given below in the author's own words. For fuller details the article itself, which extends to 62 pp. of the *American Chemical Journal*, must be consulted. It is the most important work which has appeared on this subject in recent years, and the present contribution deals with. I. The preparation of the solution. II. Analysis of the silver, copper and tin groups.

This investigation was undertaken for the purpose of studying the processes of qualitative analysis in general use, with reference to their effectiveness in enabling small quantities of each of the more common elements or element-groups to be detected in the presence of any of the other elements, and for the purpose of selecting the methods best adapted to this end and of improving them wherever desirable and practicable. In spite of the vast number of text-books on qualitative analysis in which more or less different processes are recommended, there seems to have been made in recent years no exact systematic study of the relative effectiveness of the separations or of the delicacy of the final tests. It has seemed to us, therefore, worth while to make a detailed investigation of this kind; and as a result of it to develop, if possible, a system of analysis which will enable a quantity of any element as small as one or two milligrams to be detected in any mixture. Effort will also be made to employ only such processes as enable a rough estimate to be made of the quantity of the various elements present; for this is usually a more important function of a good qualitative analysis than the mere determination of their presence or absence. To this end tests of excessive delicacy, like certain colour reactions, will be avoided, and each element will be obtained in the solid form as a precipitate or residue, as far as possible.

Especial attention will also be devoted to the chemical reactions involved in the process and to the explanations of them furnished by the Mass-Action Law and Ionic Theory. Since the first systematic application of modern theories to analytical chemistry was made by Ostwald, many physico-chemical researches have been published which make it possible to give such explanations with certainty in a greater number of cases. References to the more important articles relating to the chemistry of the processes and to their theoretical interpretation will be given.

The elements and element-groups included in this system of analysis are primarily those which, with remarkable unanimity, are treated of in the ordinary schemes of qualitative analysis; but we have also introduced the more important of the so-called rare elements, in so far as this has seemed practicable without seriously complicating or modifying the process best adapted to the more common elements. An independent 'System of Qualitative Analysis including nearly all the Metallic Elements' is also being worked out in this laboratory; and parts of it have already been published under this title. The latter scheme differs from the one here presented in that the detection of the rare elements forms the essential feature, instead of a merely incidental one.

The results will be presented in the form that seems best adapted to the purposes of the analyst. The 'System of Analysis' is for convenience in publication primarily divided into a series of parts, each of which treats of one of the large divisions of the subject. Under each part is first presented a 'General Discussion,' in which are given the reasons for the adoption of the process employed. This is followed by a 'Tabular Outline,' which gives a survey of the important steps and the chemical reactions involved in the process of analysis. Then comes the chapter entitled, 'Procedures and Notes,' in which the details of the process are presented and discussed. Next are given the 'Test Analyses,' which were made with known mixtures in order to test the efficiency of the process. Finally are presented 'Confirmatory Experiments and References,' which serve to substantiate the statements made in the notes and to justify the details of the procedures."—A. A. NOYES and W. C. BRAY.—*Journal of the American Chemical Society*, vol. xxix., No. 2, Feb., 1907, p. 137. (J. A. W.)

A STUDY OF THE FERROCYANIDE METHOD FOR THE DETERMINATION OF ZINC.*—"*Effect of Acid.*—A certain amount of free HCl is necessary to sharpen the end reaction. An excess of from 1 to 10 c.c. does not affect the results sensibly. Beyond these limits increase of acid increases consumption of ferrocyanide, particularly when the titration is conducted at a high temperature.

Effect of $KClO_3$.—The effect is to evolve Cl_2 which decomposes the ferrocyanide. A high temperature increases this decomposition and also increase of free acid. If more than 1 gm. is employed, any remainder should be decomposed before titrating, otherwise the end point is uncertain.

Effects of Increase of Acid with Constant Amounts of Chlorate.—This increases the consumption of ferrocyanide especially when titrating at a high temperature.

Effect of NH_4Cl.—This helps to settle the precipitate and contributes to sharpen the end reaction. There is no apparent evil effect.

Effect of Metallic Salts.—The influence of iron, cadmium and copper has long been known, as well as that of manganese. The bad influence of lead has

* See this *Journal*, p. 56, vol. vi., Aug., 1905; also p. 89, vol. vii., Sept., 1906.

been generally overlooked, owing, no doubt, to the fact that 1 per cent. of lead increases the apparent amount of zinc by only about 0·32 per cent., as, owing to its higher atomic weight 1 c.c. of ferrocyanide will precipitate about three times as much lead as zinc. As lead is frequently a constituent of zinc ores, those assayers who have not been removing it in determining zinc have been getting high results; varying also, since frequently more or less of the lead would be rendered insoluble, and all of it would not follow into the liquid to be titrated. Low is the only one, to my knowledge, who has sounded a warning concerning lead, and he has not been as emphatic as now seems necessary. It should be noted that aluminium, in the form of chloride, does not affect the results and it naturally suggested that aluminium might be used to remove copper. A set of experiments was made, using 0·25 gm. of zinc oxide, corresponding to 0·200875 of metallic zinc, 5 gm. ammonium chloride, 5 c.c. excess hydrochloric acid and varying amounts of copper, removing the copper with metallic lead.

Weight of Copper Added.	C.C. of Ferrocyanide Consumed.
0·	40·8
0·02046 gm.	42·4
0·05850 ,,	43·5
0·10326 ,,	45·7

The disappearance of colour indicated the complete precipitation of copper, lead going into solution corresponding atomically to the weight of copper removed, increasing the percentage of zinc but not to the same extent that an equal weight of copper would have done had it been allowed to remain.

Influences of Varying Amounts of Zinc.—Differences of from two-tenths to as much as 4 per cent. demonstrate that in control and umpire work the operator must use the factor nearest corresponding to the percentage of the ore. All determinations depending upon an end colour reaction are also much improved in their accuracy, if the operator will run a standard with his daily runs, not so much for a correction in a varying standard solution as to enable him better to judge the end point.

Modification of Low's Method as Suggested by the Experiments, Decomposition of the Ore.—Cover 0·5 gm. of the ore in a No. 3 casserole, with 7 c.c. concentrated nitric acid, after which add an equal amount of hydrochloric; allow these acids to act for 15 minutes at a temperature not exceeding 60 deg., after which add 7 gm. of ammonium chloride and evaporate to dryness on a hot plate. Remove from this hot plate, make alkaline with ammonia water, 5 c.c. is enough, add 15 c.c. of bromine water and boil for three minutes and filter while hot through a 11 cm. filter paper into a 400 c.c. beaker. Wash carefully three times with a hot solution of ammonium chloride and dilute ammonia. Make the filtrate weakly acid with dilute hydrochloric, place a piece of aluminium foil in the beaker, cover with a watch glass and boil for about three minutes, when all copper, lead and cadmium present will be precipitated. Remove the foil, wash it off and heat to boiling for titration. It is unnecessary to remove the precipitated metals, as their presence does not affect the results.

Titration.—The operator should use a constant bulk, which may be 150 c.c. or 200 c.c., as he elects. Add 5 c.c. strong hydrochloric acid and titrate according to the method given by Low, starting the titration at a temperature of about 85 deg. and completing it before temperature falls to 40 deg.

Standardisation.—A solution of about 22 gm. of ferrocyanide to the litre is satisfactory and the acetate, or nitrate, of uranium is used as indicator. If accurate work is to be done, the conditions for standardisation should be close to those employed in treating an ore. Zinc oxide, freshly ignited, or metallic zinc may be employed as convenient. Weigh out into casseroles 0·05, 0·1, 0·15 and 0·2 gm. of the oxide and treat precisely as if it were an ore, omitting only the bromine treatment, the aluminium foil and the filtration. Run one blank assay to determine the amount of ferrocyanide required to produce the colouration of the indicator to the depth selected by the operator and deduct this from all burette readings. Use the factor nearest corresponding to the percentage of zinc in the ore.

Remarks.—The ore must be decomposed. The method outlined has decomposed all the varieties of ores that have come under my observation. The operator will observe that on the addition of ammonium chloride there is an evolution of chlorine which assists decomposition. The ammonium chloride also makes the iron precipitate, formed when the ammonium is added, somewhat granular, making the filtration more rapid and eliminating the tendency of the gelatinous iron hydroxide to hold up zinc salts. As much as 1 gm. of potassium chlorate may be employed to decompose sulphides without disturbing the results; but if more is used, Low's direction to decompose the excess must be followed.

Hydrogen peroxide may be advantageously substituted for the bromine water to remove manganese. If manganese is known to be absent, the treatment may be omitted, just as the aluminium treatment may be omitted in the absence of copper, lead and cadmium." — W. H. SEAMAN. — *Journal of the American Chemical Society*, vol. xxix., No. 2, Feb., 1907, p. 205. (J. A. W.)

DETERMINATION OF GOLD IN AURIFEROUS SANDS BY THE WET METHOD.—"The sand is intimately mixed with 5—10 per cent. of manganese dioxide, treated with hydrochloric acid in the cold for 24 hours, then heated on the water-bath till the liquid has completely cleared. The liquid is then decanted, and the residue washed with acidulated water; the liquid and washings are concentrated till their consistence is pasty while the mass is still hot, and the pasty material is dropped into a sufficiency of 95 per cent. alcohol, thoroughly shaken, allowed to stand, for 48 hours, and decanted. The alcoholic solution, is heated on the water-bath and saturated with hydrogen sulphide, which may take 6—12 hours, or even more if quantities of more than a kilogramme are being treated. The alcohol has the advantages of: (1) Separating as insoluble a number of substances soluble in water; (2) favouring the agglomeration and settling out of the precipitated sulphides; (3) reducing considerably the precipitation of sulphur by ferric iron; (4) preventing the co-precipitation of ferrous sulphide, or facilitating its removal by washing. The sulphides are washed with boiling water, containing a little hydrochloric, acid, finally with pure water. They are then extracted with hydrochloric acid free from nitric acid, and afterwards dissolved in nitrohydrochloric acid, and treated to separate the gold. The advantage of this over the dry method is that much larger quantities can be treated."—A. FOURNIER.—*Journal of the Society of Chemical Industry*, March 30, 1907, p 257. (A. W.)

DETECTION OF SULPHITES IN PRESENCE OF THIOSULPHATES AND THIONATES.—"Dilute solutions of magenta or malachite green are, in contradistinction to the well-known reaction with sulphurous acid or bisulphites, instantaneously decolorised by normal sulphites, whereby the sulphite of the colour base is formed. If an aqueous solution of acetaldehyde be now added, the colour is regenerated. The author finds that thiosulphatss, di-, tri-, and tetra-thionates do not produce this decolorisation, so that it is possible to use this reaction to identify sulphites in presence of these salts. Bicarbonates, hydrosulphides, phosphates, etc., do not interfere with the reaction, but mono- and poly-sulphides have the same action as sulphites, and must, therefore, be first removed before carrying out the test. The most suitable solution was found to be composed of 3 vols. of magenta solution (0·25 grm. in 1 litre of water) mixed with 1 vol. of malachite green solution (0·25 grm. in 1 litre of water). This is added, drop by drop, to 2—3 c.c. of the solution to be tested; if normal sulphite be present, the dye solution is instantly decolorised, and on adding acetaldehyde solution, the colour reappears If the solution to be tested contain free alkali, this must first be converted into bicarbonate by a stream of carbon dioxide, and the presence of acids necessitates the addition of a bicarbonate. The test is a very delicate one; an aqueous solution containing 0·00006 grm. of sulphurous acid (as normal sulphite) in 1 c.c. at once decolourises the dye solution. The following table shows the qualitative analysis of solutions from which sulphur is separated on treatment with acids :—

I.—The solution, (A) decolorises the reagent, and regenerates a violet colour with acetaldehyde, (B) is coloured violet with alkaline sodium nitroprusside, (C) by treatment with cadmium carbonate, filtering, and saturating with carbon dioxide, neither decolorises the reagent nor separates sulphur with acid.	Presence of poly- or normal sulphides, but absence of sulphite.
II.—(A) Does not decolorise the reagent, (B) gives no colour with sodium nitroprusside.	Can contain thiosulphates or di-, tri-, or tetrathionates.
III.—(A) Decolorises the reagent, but, (B) gives no colour with sodium nitroprusside.	Presence of sulphite and thiosulphate.
IV.—(A) Decolorises the reagent, (B) gives violet colour with sodium nitroprusside, (C) when treated with cadmium carbonate, etc., as in I. (C), decolorises the reagent, but does not separate sulphur with acid.	Presence of monosulphide and sulphite.
V.—(A) Gives violet colour with sodium nitroprusside, (B) decolorises reagent, (C) treated with cadmium carbonate as in I. (C), does not decolorise reagent.	Presence of monosulphide and thiosulphate.
VI.—(A) Decolorises reagent, (B) gives violet colour with sodium nitroprusside, (C) treated with cadmium carbonate, as in I. (C), decolorises reagent, and separates sulphur with acid.	Presence of monosulphide, sulphite, and thiosulphate.

Dilute solutions of hydrosulphide and sulphides are distinguished in that the former does not decolorise the reagent, whilst the latter does. The reagent may also be used for the detection of sulphur dioxide in presence of hydrogen sulphide in gas analysis."—E. VOTOCEK.—*Journal of the Society for Chemical Industry*, March 15, 1907, p. 199. (A. W.)

DETECTION OF HYDROCYANIC ACID BY PHTHALOPHENONE PAPER.*—"The following method of applying Weehuizen's test is more sensitive than the original process. Absorbent paper, moistened with a 1 : 2,000 solution of cupric sulphate is dried and cut into suitable strips. An alkaline phthalophenone† reagent is thus prepared. 0·50 gm. of phenolphthalein is dissolved in 30 c.c. of absolute alcohol; sufficient distilled water is then added to produce a faint turbidity, when 20 gm. of sodium hydroxide are added. Aluminium dust is then added to the red alkaline solution, a little at a time, until the colour is discharged. The liquid is next diluted to 150 c.c. with distilled water, which has been boiled and cooled, excluded from contact with the air. The reagent is then filtered. It keeps indefinitely. To apply the test, the cupric sulphate test-paper is moistened immediately before use with a few drops of the reagent. It will detect the presence of 1 : 2,000,000 of hydrocyanic acid. Hydrogen peroxide, ferric chloride, nitric acid, and ethyl nitrate do not give a similar reaction. Liquids containing ammonium persulphate, hypochlorites, sodium peroxide, or perchlorates, give a positive reaction, but the colour produced slowly fades, entirely disappearing in a few hours, whereas that given with hydrocyanic acid is permanent for 24 hours."—THIÉRY.—*Journal of the Society of Chemical Industry*, Feb. 28, 1907, p. 168. (A. W.)

DETERMINATION OF THE PROPORTIONS OF CARBONATE AND SILICATE OF ZINC IN ORES.—"The method here suggested for determining the relative amounts of zinc present as carbonate and silicate in an ore gives results which, although only approximate, are more accurate than those given by existing processes. 0·5 gm. of the ore is heated with 5 c.c. of water and 10 c.c. of hydrochloric acid, and filtered at once and washed. The insoluble matter subtracted from the total silica gives the amount of silica in combination as zinc silicate, $H_2Zn_2SiO_5$, and hence the weight of zinc as silicate (A gm.). A portion of the ore is heated in a porcelain boat for 45 minutes at 350 deg.—400 deg. C., and the carbon dioxide evolved is absorbed and weighed. This gives the amount of zinc present as carbonate, $ZnCO_3$ (B gm.). The total zinc is then determined (C gm.). The true weight of zinc present as silicate is then taken as $(A + C - B)/_2$ gm., and of that present in the form of carbonate as $(B + C - A)/_2$ gm."—P. H. WALKER and H. SCHREIBER.—*Journal of the Society of Chemical Industry*, March 30, 1907, p. 257. (A. W.)

STANDARDISING PERMANGANATE. — "Metallic Ag is quantitatively dissolved by $Fe_2(SO_4)_3$. Hence a weighed amount of pure Ag when heated out of contact with the air (valved flask, etc.), with sufficient iron alum solution of suitable concentration reduces a corresponding amount of Fe to ferrous salt. Thus: $2Ag + Fe_2(SO_4)_3 = Ag_2SO_4 + 2FeSO_4$.
The solution may then be titrated with the permanganate." — E. WALLER. — *The Quarterly*, Columbia University, Jan., 1907, p. 226. (H. A. W.)

INCANDESCENT ILLUMINANTS.—"Mr. J. Swinburne, F.R.S., lectured on this subject. He pointed out that the incandescent mantle was now not only applied to the petroleum lamp, but to air carrying a little hydrocarbon gas, an application said to provide an extraordinary cheap light. The facility to light by merely turning on had been achieved by the use of the bye-pass, and further advances were being

* See this *Journal*, p. 59, vol. vi., August, 1905, where Phenolphthalein should read Phenolphthal*ine*.
† ? Phenolphthal*ine*. [ED. COM.]

made in this connection. Welsbach had discovered that an alloy of cerium and iron gave off sparks on being scraped or filed, and a burner had been designed in which the act of turning on the gas scraped a little wheel of this alloy, causing a spark which lighted the gas. Another invention permitted the gas to be lighted from a main tap. Each burner had an attachment which passed the gas straight through the burner when the pressure was on; but, on turning the main supply off, and allowing a little gas to pass at the controlling tap, the attachment to each burner turned off the burner and lighted a little pilot jet which remained burning until light was again wanted. On turning on the main tap, the pilot jets lighted the various burners and were themselves extinguished. By this means burners could be fully lighted up by turning one tap at the door of any room. The electric incandescent light was undergoing great changes in the direction of the replacement of carbon by metal filaments The tantalum lamp was now in great demand. It was made for 100 to 130 volts, and was much more efficient than the carbon lamp, but although it did not last long on alternating current. The wires of a lamp that had been run for some time on direct current showed a curious notch or crinkled appearance under the microscope. A wire that had been run on an alternating circuit looked as if the metal had been melted into short cylinders with round ends, and these cylinders had stuck together end to end without their centres being in a line. Occasionally the little cylinders were nearly separated. This very extraordinary action had never been explained. One of the most interesting of the new lamps was the Zircon. It was said to be made of zirconium and tungsten, and lamps of this material had been made for 200 volts, which was of the greatest importance from the point of view of distribution. It was possible that the conductor was really a zirconide of tungsten, and, if so, this opened up a new series of compounds. A Zircon lamp for 100 volts had really six separate loops of wire mounted in series inside a bulb. A recent improvement was to provide an extremely light spring for each loop in order to keep it taut, and the lamp could then be used in any position. Tungsten seemed to be the favourite metal for new lamps, and it had the advantage of giving a very high efficiency. It was probable that the lamp of the future would have an efficiency of nearly one candle per watt, and this was promised by the use of tungsten. At the same time it had to be admitted that to make a wire with a resistance of 500 ohms small enough to give 20 candles with 20 watts would be a triumph of inventive skill."—*Times Engineering Supplement*, May 1, 1907. (J. A. W.)

REPORT OF INTERNATIONAL COMMITTEE ON ATOMIC WEIGHTS, 1907.*—"From the evidence presented in this report, and in preceding years, we now feel justified in recommending the following changes in the table:

Nitrogen,	from	14·04	to	14·01
Bismuth,	from	208·5	to	208.0
Tantalum,	from	183·	to	181·
Terbium,	from	160·	to	159·2

Other changes, which seem to be needed because of alterations in the atomic weights of silver and chlorine, cannot yet be made with safety. The atomic weight of silver, as deduced from Stas' data, is probably too high, but by an unknown amount and that will affect the entire table. If we assume, with Guye, that Ag = 107·89, with the proportional changes in Cl and Br, the atomic weight of barium, as determined by Richards, will be reduced by 0·05. Such a change, which is probably extreme, does not effect the utility of the accepted atomic weights at all seriously, and no important interest will suffer if we delay the suggested alterations until our knowledge of the corrections to be applied is more exact. Guye's conclusions, although strongly supported, are not final; and they should be neither accepted nor rejected except upon the basis of much more complete evidence than we now possess. The atomic weight of chlorine, as shown in our last report, is certainly too low; but it depends in part upon the undetermined change to be applied to silver. For that reason, as well as for the reason that a change in chlorine affects many other values, we prefer to leave the figures as they are, and to wait for fuller information. That information will doubtless be supplied by researches now known to be in progress, and the corrections which they will furnish ought not to be delayed very long.

One addition to the table seems to be legitimate. Europium, with an approximate atomic weight of 152, appears to be a definite element, as shown by the investigations of Demarçay, Urbain and Lacombe, Eberhard, and Feit and Przibylla. Its existence is recognised in Abegg's Handbuch and its claims to a place in the table are certainly as great as those of erbium, thulium, or terbium. As for dysprosium, its admittance to the table may well be delayed until a better determination of its atomic weight shall have been made."—F. W. CLARKE, T. E. THORPE, H. MOISSAN and W. OSTWALD.—*Journal of the American Chemical Society*, vol. xxix., No. 2, p. 107, Feb., 1907. (J. A. W.)

* See this *Journal*, p. 346, vol. vi., May, 1906; also p. 215, vol. v., Feb. 1905; also p. 270, vol. vii., Feb., 1907.

FREE ACID IN PRESENCE OF FERRIC SALTS.—"By addition of sodium dihydrogen phosphate (NaH_2PO_4) Fe is removed from the solution and the ordinary acidimetric titration may be used. A correction must be applied for the amount of acid set free by the reaction which is proportional to the amount of Fe present, *e.g.*, $Fe_2Cl_6 + 2NaH_2PO_4 = 2FePO_4 + 2NaCl + 4HCl$."—E. WALLER.—*The Quarterly*, Columbia University, Jan., 1907, p. 227. (H. A. W.)

THE VAPOUR PRESSURE OF IODINE. — "1. The vapour pressure of solid iodine is found to have the following values:—

Temperature Degrees.	Vapour Pressure. mm.	Temperature Degrees.	Vapour Pressure. mm.
0	0·030	40	1·025
15	0·131	45	1·498
25	0·305	50	2·154
30	0·469	55	3·084
35	0·699		

2. The molecular heat of sublimation of iodine at these temperatures is calculated to be 15·1 kgm. calories or 63 kilojoules."—G. P. BAXTER, C. H. HICKEY and W. C. HOLMES. — *Journal of the American Chemical Society*, vol., xxix., No. 2, Feb., 1907, p. 127. (J. A. W.)

NON-EXPLOSIVE AIR GAS.—"A demonstration was given last week of the Cox air gas-producing plant. The method of manufacturing air gas consists in saturating ordinary air with the vapour of a light hydrocarbon, for which purpose petrol is admirably adapted. In this machine the work is accomplished through the operation of a small rotary pump, which passes air through a carburettor, in which is a mechanical appliance for distributing petrol in very small quantities over a very large area, and the regulation of supply and pressure is designed to give a continuous flow of a fixed mixture in varying quantities, according to the demand. It is claimed that the condensation difficulty has been overcome and that complete combustion is obtained. The burners are specially constructed with a view to preventing lighting back, but, while distribution can be effected by the ordinary pipe system used for coal gas, nothing smaller than a ⅝ in. diameter pipe is recommended. That the mixture is a fixed gas was tested by a visit to the Agricultural Hall, where an installation was on view in which the gas had to pass through 600 ft. of piping. Experiments are being made with a view to adapting the gas for train lighting by compression into cylinders, and representatives of the Great Eastern and other railways were present at the demonstration. The light is claimed to be superior in illuminating power to that produced by coal gas, and experience quoted is to the effect that by the use of 1 gallon of petrol as much light can be obtained as by means of 1,000 ft. of ordinary coal gas. The mixture, consisting as it does of about 98 per cent. of air and 2 per cent. of hydrocarbon, is non-explosive. The machine is portable and requires no skilled labour to manipulate it."—*Times Engineering Supplement*, May 1, 1907. (J. A. W.)

HYDRONITRIC ACID (AZOIMIDE).—"The authors have succeeded in preparing anhydrous hydronitric acid by allowing dilute sulphuric acid (2 of acid to 1 of water) to drop on to dry potassium trinitride, the acid being carried along by a stream of pure dry air, dried by means of calcium chloride, and then condensed by the use of liquid air. Its melting and boiling points were measured by a thermo-electric junction. The acid is a colourless, mobile liquid, heavier than water. Its boiling point is about 37 deg. C., as given by Curtius. It may be frozen to a white, crystalline mass, melting at − 80 deg. C. It is apparently quite stable at the ordinary temperature, but when heated, or subjected to slight shock, it explodes with great violence, although the damage is quite local. Its purity was ascertained by weighing in a sealed U-tube, and then blowing over the acid into caustic potash solution, neutralising the excess of alkali by dilute nitric acid, precipitating with silver nitrate, and then converting the silver trinitride into silver chloride, which was weighed. Its vapour density, taken in a bath of boiling chloroform, corresponds to the formula, HN_3."—L. M. DENNIS and H. ISHAM.—*Journal of the Society of Chemical Industry*, March 30, p. 282. F. SOHN. (J. A. W.)

METALLURGY.

RECENT IMPROVEMENTS IN THE CYANIDE PROCESS.*—"During recent years, no radical changes were made in the process in South Africa. This was due to the serious blow given the mining industry by the Boer war; and also, in part, to the conservatism of metallurgists on the Rand and their reluctance to adopt important innovations originating elsewhere. The brothers Denny were the first of the Rand metallurgists to recognise the importance of finer grinding, and their energetic advocacy of tube milling and filter pressing finally resulted in the acceptance of Australian methods.

It was during the lethargy of cyanidation on the Rand that the filter press and the tube mill were introduced in Western Australia. In this connection the interesting fact may be noted that all the important devices introduced into cyanide practice had been previously used in other industries. Even the pipe distributor used for distributing tailings in a leaching vat was an adaptation of the common lawn sprinkler. The filter press had been employed to strain solutions in the refining of sugar; the tube mill had been in use as a dry grinding machine in the cement industry.

The metallurgists of Australia never took kindly to decantation in slime treatment, and the introduction of the filter press was the result. In justice to African operators, however, it must be said that decantation was well suited to existing conditions. The product they were handling was of too low grade to stand the prohibitive cost of filter pressing. In Western Australia the filter press was applied to a much richer product, and one much better adapted to the method.

The obvious objection to the old type of filter presses is the high cost of installation and operation, but they nevertheless enjoyed a great success; and it is worthy of note, in observing the evolution of the process, that they were a means of emphasising the importance of fine grinding and helped to establish the tube mill. It has always been a truism in cyanidation that the finer the product the higher the extraction. This is the case with but few exceptions. To apply this principle required two things; an economical machine for fine grinding and a filtering system that would be at once efficient and economical. As a machine for economically reducing ore to an extreme fineness, the tube mill has no equal. The cost of operation is variable. In this country and Mexico it will range between 20 cents and 40 cents per ton. The work at El Oro, Mexico, and at Telluride, Colorado, is representative of the best practice on this continent; while that at the Combination mine, Goldfield, Nevada, probably represents the maximum of cost, owing to high price of power and labour. A small 4 by 12 ft. trunnion mill is installed at the Combination for sliming the 40 mesh product from a Bryan mill. The product of ten stamps, about 35 tons of ore per day, passes to the tube mill classifier; and of this product about 75 per cent. goes to the tube mill, or 24·6 tons per day. The following figures may be of interest:—

Cost of 2½ in. silex lining laid in mill ...	$323·50
Life of lining	4 months
Cost of lining per ton of ore stamped	7·7 cents
Cost of pebbles delivered at Goldfield	$71 per ton
Consumption of pebbles	2·03 lb. per ton of ore stamped
Cost of pebbles per ton of ore stamped	7·1 cents
Power consumed	25 h.p. at $11·25 per h p. per month
Cost of power ...	26·7 cents per ton of ore stamped

Summary, cost per ton of ore stamped:—

Pebbles	$0·071
Lining	0·077
Power...	0·267
	$0·415

Tending the mill is one of the several duties falling upon one man, and the consumption of lubricants is almost negligible; therefore I have not included these two items in the cost. This cost of 41 cents per ton may be assumed to be the maximum for tube milling, owing to the very high cost of labour and transport in southern Nevada camps.

I have already referred to filter pressing as an established practice in Western Australia. The press was never very popular in Amercia, and few successful installations are recorded. The most noteworthy, perhaps, is that of the Gold Road mine, near Kingman, in Arizona, where two 5-ton Dehne presses have been successfully operated for some time. About three years ago Mr. George Moore, after a series of failures in an attempt to filter press slime

* See this *Journal*, vol. vii., Jan., 1907, p. 224; Oct., 1906, p. 128; Nov., 1906, p. 150.

at the mill of the Consolidated Mercur Co., in Utah, devised a vacuum filter and installed a plant in the Mercur mill. This was the origin of the vacuum filter, recent modifications of which are to be found at a number of mills in this country. Experiments recently made in Australia so far demonstrate the superiority of this method over all others, that it seems safe to predict the early disappearance of the filter press.

The objections to the Moore filter are the high first cost of the mechanism required to shift the slime and the high cost of maintenance. The unmechanical and cumbersome features of this system led to the introduction by Cassel of a stationary filter, and the elimination of the awkward mechanism of the Moore scheme. It remained for Butters to adopt the Cassel principle, simplify it, and so modify it as to make it a pronounced success at his Virginia City plant. In the Butters filter, the leaves are set in a rectangular box or vat, the bottom of the box consisting of a series of pointed pockets, to facilitate the discharge of the spent cakes. The frames are approximately 5 by 10 ft. and consist of a piece of cocoa matting with a sheet of canvas quilted on each side, the whole being stitched on a frame of ½ in. pipe and securely sewed to this pipe frame, which in turn is supported on a timber header. The bottom arm of the frame is perforated with small holes, through which the solution enters the pipe when the vacuum is applied. On one side the pipe frame is projected through the wooden header, and is connected with a common vacuum pipe leading to the vacuum pump. The frames stand parallel in the filter box at about 4¼ in. centres. The pulp is drawn from the slime reservoir and pumped into the bottom of the filter box until all the frames are immersed. The vacuum is then applied until a cake of suitable thickness is deposited, and the excess of pulp is then withdrawn into the slime reservoir. This operation is repeated for the wash, and the cake finally discharged into the bottom of the box by introducing water under a low head into the interior of the leaves. The accumulated cakes from each charge are removed by sluicing.

This system possesses the great advantage of simplicity and low cost of maintenance. A plant of any size can be operated by one man, who stands on a platform on a level with the top of a filter box and manipulates the pumps with levers, and the valves with a simple drum-and-sheave mechanism. The 200-ton plant of this type in the Butters mill at Virginia City is operated at a cost of about 10 cents per ton of slime.

At the Combination mill 40 tons of this slime per day are being filtered at a cost of about 45 cents per ton, as follows:

Three men at $4; $12 per day ...	30 cents per ton
Twelve h.p. at $11·25 per h.p. per month	11 cents per ton
Lubricants and incidentals ...	4 ,, ,,
	45 cents per ton

This plant, however, has a capacity of 56 tons per day. If worked to its limit of capacity, the cost would be reduced to 31 cents per ton. The cost of filter-pressing at the same plant in the early days of operation was approximately $1 per ton.

The 15 h.p. consumed is used for the following purposes:

Driving a 4 in. Butters centrifugal pump.

Operating a 12 by 10 Gould's vacuum pump.

Operating a 2 in. centrifugal pump for raising the filtered solution to a clarifying filter-press.

Operating a 2 in. centrifugal pump for returning the slime-overflow from the leaching vats to the slime-settlers.

Operating stirring mechanism in two slime-reservoirs, 14 ft. diameter.

The power may be distributed as follows:

For actual operation of filter, capacity 56 tons per day	9 h.p.
For agitating slime-pump	3 ,,
For uses not connected with filter ...	3 ,,

The cycle of operation in the Butters filter consumes about 3 hours 20 minutes; it will vary, of course, with the nature of the slime to be filtered.

This type of filter has been installed or is in process of installation at six mills, in Nevada, Mexico, and Salvador.

There are certain conditions, however, where the product to be handled is too low grade to admit even of vacuum-filtering; these require special study and a special process. The need of a special process to suit a unique condition was never better exemplified than in the case of the Homestake ore.

I shall not venture to describe the different problems encountered and successfully solved by Mr. C. W. Merrill at the Homestake in the cyaniding of a tailing averaging less than $1·50 per ton. The next and most serious problem to engage his attention was the treatment of the slime, of which 1,600 tons per day of an average value of 80 cents per ton have been run to waste from the Homestake mills. Mr. Merrill has devised a filter-press, the unique feature of which is that it can be automatically discharged by sluicing without being opened, thus doing away with the chief objections of the old type of press, namely, the cost of operating. This press is of a common flush-plate and distance-frame pattern, but consists of much larger units. The dimensions are as follows:

Number of frames	92
Size of frame	4 by 6 ft.
Length of press	45 ft.
Capacity of press	26 tons
Weight of press	65 tons
Thickness of cake	4 in.

The slime, after partial de-watering, consists of about three parts of water to one of solid. In this form it is charged by gravity to the presses at about 30 lb. pressure. The leaching with cyanide solution is done in the press, the effluent solutions being conducted to four precipitating vats where the gold is recovered on zinc dust. There is no power-cost for agitating or elevating, except for elevating the solution to the press. There will be only 0·6 ton of solution handled per ton of slime, of which only 0·3 ton will be precipitated. All filtering will be done by gravity at a cost of two cents per ton.

This plant is being erected on the basis of tests made in a four-ton press of the type described. In all, 1,291 tons were treated with the following results:

Average assay-value of slime before treatment	91 cents per ton
Average assay-value of slime after treatment	10 cents per ton
Extraction by assay, per ton ...	81 cents or 90%
Recovered in precipitate per ton ...	83 cents or 91%

The following facts relating to this important installation are of interest:

Estimated cost of slime-treatment	25 cents per ton
Estimated profit per year ...	$300,000
Cost of plant	$400,000
Cost of pipe-lines	$50,000
Area covered by plant	270 by 65 ft.

Area covered per ton of capacity ..	Two square yards
Cost of installation per ton of capacity	$300
Filter-press cost per ton of capacity	$60
Power required ...	$\frac{1}{10}$ h.p. per ton of slime treated
Water required for sluicing ...	4 tons water to 1 ton of slime

In the elaboration of the above process and plant, Mr. Merrill shares with the projectors of the vacuum-filter, the credit of the most noteworthy advance made in recent years in the metallurgy of the cyanide process. His method is admirably adapted to Homestake conditions; and its striking economical features must appeal at once to every engineer and manager who concerns himself with such vital things as costs and profits.

Whether an automatically discharged press can compete under average conditions with the vacuum filter, remains to be seen. Much, of course, will depend upon local conditions, such as the site, the utilisation of gravity for various operations, water facilities, water supply, and the permeability of the material under treatment, as well as the rapidity with which it yields up its precious metal.

Space will not permit me to touch upon zinc dust precipitation further than to say that there is every indication that it will eventually take the place of zinc shaving in all plants of large capacity. Electric deposition offers a large and promising field for investigation, but has not as yet been brought to a perfectly satisfactory conclusion. Recent advances in cyanidation have mainly to do with the finer reduction of the product to be treated. The treatment of sand by leaching will probably continue to be the best method in a few instances; but no observer will deny that the trend of modern practice is toward fine grinding, and doing away with leaching—a result made possible by the introduction of the tube-mill and the efficient filtering methods now in vogue."—F. L. Bosqui.—*Mining and Scientific Press,* Dec. 15, 1906, p. 719. (K. L. G.)

Modern Gold Mining.—"In the modern plants a good deal of attention is being paid to securing good and substantial foundations for the apparatus, and especially the stamp batteries. Liberal use is made of concrete for this purpose, and the foundation work is sometimes very costly, but the increase in cost is more than made up for by the solidity and freedom from jarring and vibration of the building. In regard to the battery foundations, modern tendencies are overwhelmingly in favour of the concrete foundation. The prejudice which unquestionably existed in the mind of many managers, even men of wide experience in stamp-mill practice, against the concrete foundations, seems to be fast disappearing.

It is quite true that the opponents of the concrete foundation could point to instances where the foundations had not given satisfaction, but the greater part of the trouble was undoubtedly caused by the unfamiliarity of the men who built the foundation, with the kind of material required for this work, and perhaps their unfamiliarity with concrete work in general. It is not always easy in far-away mining regions to secure men particularly fitted for carrying out a delicate job of work which requires care and attention, as concrete work undoubtedly does, and the tendencies to carelessness in the working of the material and the proper mixing and proportioning of the ingredients are always present. Moreover, a good deal of care is required in getting the battery mortars to bear evenly with their lower surface on the top of the concrete foundation.

The interposition of a sheet of rubber to cushion the blow is, of course, an additional safeguard in this respect, but it is much more advisable to pay careful attention to the levelling of the top of the foundation and to use a comparatively thin sheet of rubber than to rely upon a thicker intervening layer. A well-built concrete foundation is sure to last indefinitely, and to cause much less trouble than a wooden foundation, which latter is undoubtedly the weakest point in the battery, and a source of considerable trouble when it needs repairs and renewals.

A point to which particular attention should be paid in the construction of concrete foundations is that the proper amount of cement should be allowed in the mixture. This is one of the instances where it does not pay to save on the cement and use too high a proportion of inert materials. It should always be remembered that the cement is the only material that holds the concrete together and gives strength to the mixture, and that a structure which is so much subjected to vibration and strains as a battery foundation is, needs to have considerable strength. The tendency, of course, to cheapen the cost of the concrete by decreasing the amount of cement is always present, especially when the work is done by independent contractors. One instance of such a procedure came to the notice of the writer, where the battery foundation had cracked on account of the poor material used by the contractor, and had to be largely replaced."—Geo. P. Scholl.—*Mining and Scientific Press* from *Electrochemical and Metallurgical Industry*, Feb. 16, 1907, p. 211. (K. L. G.)

Treatment of Arsenical Ores.—"The Cascade range is noted for the arsenical character of many of its ores. At Monte Cristo, Goat Lake, Silverton, Silver Creek, Elba and many other places a considerable arsenical content is a characteristic. Not all the ores of this range are arsenical by any means, but those in which this constituent is prominent are so widely spread and so considerable in amount that the handling of them and the getting of some return for the arsenic contained is a vital matter. The only plant in the state that can make marketable arsenic (the white arsenic of commerce, As_4O_6), is at Everett, and, indeed, until 1904, when the Anaconda works in Montana were supplied with a plant for extracting arsensic from flue dust, this was the only place in the United States where it was made.

At the present time, what ores of this class are treated at all are shipped to smelters and treated as gold or silver ores pure and simple, and a penalty of about 50 cents is exacted for each per cent. of arsenic over a certain amount, and the smelters refuse to take on any terms ores high in this metal; they claim that the volatilisation of arsenic in the roasting furnaces and the smelter stacks causes undue losses of the precious metal. Producers of ore have long contended that not only should they not be penalised for their arsenic, but they should be paid for it. They point out that the Anaconda works have produced, in the three years ending July 1st, 1906, 1,107,176 lb. of white arsenic, all of which was sold at a good price, and all of which came from a by-product, flue dust.

If this is the case how much more ought their ores rich in arsenic and of fair gold and silver values, to have the treatment suited to them, the pecuniary benefit of which would be felt in every mining district of the state.

The quantity of these ores and the development work done on the properties that can produce them is sufficient to warrant a constant and increasing supply; why, then, cannot this metal receive the same sort of attention accorded to copper, lead, zinc, etc.? The reason probably is that the metallurgy of arsenic has had no attention in this country, as heretofore there has been no visible supply of the raw material. The beginning of better things seems to be now at hand, and the American Smelting and Refining Company, to whose tender mercies we seem to be irrevocably committed, as far as reduction works are concerned, are said to have promised some return for arsenic in a certain district provided a sufficient tonnage to be regularly shipped was guaranteed them.

White arsenic is worth on an average from 3 to $3\frac{1}{2}$ cents per lb. This is the grade known as 'commercial.'

You will see quoted in the lists of chemicals 'refined' and 'C.P.' (chemically pure) at rates several times this, and at the present time, in sympathy with other metals, the price is considerably above this; but an average of $3\frac{1}{2}$ cents is all that it is safe to count on, year in and year out, for the unrefined product.

The general run of the Cascade ores of this class contain from 3 per cent. to 10 per cent. of metallic arsenic (As). About 6 per cent. may be taken as a fair average, as that is what a careful sampling of two of the best known mines gives; 6 per cent. metallic arsenic will produce 160 lb. of commercial white arsenic theoretically (6 per cent. is 120 lb. per ton; allow $\frac{1}{3}$ more for the oxygen absorbed from the air, as the atomic weight of As_2 is 150 and of O_3 is 48), and we have 160 lb. white arsenic; allow 5 per cent. loss on this in the process of obtaining it, and we have 150 lb. net; now deduct for the cost of reduction, packing in barrels, freight, etc., $1\frac{1}{2}$ cents per lb., and we have 150 lb. at 2 cents, or \$3·04, as a fair return on a 6 per cent. ore. Many of our Cascade ores carry from \$6 to \$10 in gold and silver, and this extra \$3 would make all the difference with them between success and failure, between profit and loss in the mining operations. It costs not a cent more to get this ore out than formerly, and if \$3 even can be added to the returns it is an addition to the net.

Aside from the possibilities of shipping and smelting the raw ore there are at least three other courses open: (1) water concentration with subsequent smelting of concentrates; (2) cyanidation of the raw ore for gold and silver, and subsequent roasting for arsenic; (3) roasting for arsenic with subsequent smelting of the roasted residue.

To take them up in order: Water concentration has been tried at Monte Cristo, with only partial success. The sulphides and arsenides which carry the values are softer and more brittle than the quartz in which they occur and excessive fines and slimes result from grinding, the losses in this particular case are said to have run from 30 per cent. to 40 per cent. This mill has been lately repaired and will begin a new campaign in the spring under new management and the results will be looked for with interest. There are arsenical ores in other places in the Cascades better adapted to concentration than those of Monte Cristo, ores that are less massive and more granular as to the sulphides and in which the quartz and other gangue matter is not so tough and hard.

Raw cyaniding of sulphide ores has made a good deal of progress in the last few years and has been very successful in a number of cases. Cascade arsenical ores have been the subject of experiment here in Seattle in the laboratory and on a small practical scale and enough has been done to show that the thing is feasible in this case. An extraction of gold of 80 per cent. to 90 per cent. has been obtained and the estimated cost in a 100-ton plant is \$1·50 a ton for operation. After cyaniding and washing, the ores can be roasted for arsenic. This roasting should not cost more than 50 cents per ton.

The third possibility is to roast the ore first for arsenic in a comparatively inexpensive plant at the mine and ship the roasted ores to the smelters for the extraction of the precious metals. This would obviate many difficulties at present encountered. The ores would be freed from nearly all the sulphur as well as their arsenic, which would result in a slight enrichment of the product, the arsenic could be marketed by the producer and the smelters would be eager to get the desulphurised and dearsenuretted product and would be willing to pay good rates for it.

Good returns await the company that has ore reserves large enough and the courage great enough to be the pioneer among these lines. Either of the two latter methods should be the subject of careful experiment on a small working scale before works of any size are built. Under competent direction tests can be made at a reasonable cost with results that will be reliable and of great benefit to the mine owner and the mining industry of this section."—C. F. LEE. —Reproduced in *Mining Reporter* from *Canadian Mining Review*, March 14, 1907, p. 245. (J. Y.)

MINING.

POLLUTION OF MINE AIR BY TIMBER.* — The pollution of air in mines is sometimes due partly to the emanation of gases from the surrounding rocks, but principally from the following artificial causes: (1) Respiration of persons and animals, (2) combustion of the candles and lamps, (3) absorption of the oxygen by pyrites and other minerals, (4) putrefaction of timber, (5) explosion of powder, (6) dust from boring, etc. Of these the decay of timber underground is one of the most important, and it should be urged that the practice of leaving the useless decaying timber to infect the new pieces put in, sometimes making a level a hotbed of putrescent matter, is both offensive to the senses, and injurious to the men. One of the recommendations of the Ventilation of Mines Board in Victoria (Australia) was that all the bark should be removed from the timber before it was sent underground.—*Mining and Scientific Press*, Feb. 19, 1907, p. 81. (W. A. C.)

DEFLECTION OF BOREHOLES IN DIAMOND DRILLING.—"The question by which means the deflection of boreholes may be prevented or at least reduced is of great importance. To prevent deflection entirely is almost impossible, since it demands unattainable ideal conditions of the boring tool. I shall, therefore, confine my remarks, based on foregoing observations, to the possible means to reduce the deflections. To attain this end I would suggest:—

(1) That the greatest care be taken in starting the borehole vertically, as any faulty setting of the drill is bound to develop into a deflection.

(2) To strengthen the lowest portion of the boring tool, viz., the core-barrel and the shell, especially the latter, even at the cost of the diameter of the core. This would, of course, entail the cutting of a large surface of rock, and make drilling more expensive.

* See this *Journal*, p. 206, Jan., 1907.

(3) To reduce the play between the core-barrel and borehole. Though for practical reasons (water circulation, etc.), the diameter of the smallest bit used at present ($2\frac{3}{32}$ in.) will not stand much reduction, it should be possible to reduce the difference in diameter between the bore-barrel, and the larger boring bits used higher up in the hole, thereby reducing the taper of the hole.

(4) To reduce the pressure brought to bear on the lowest portion of the boring tool. This could, of course, only be done at the cost of footage made and consequently of an increase in boring expenses.

(5) To watch the rods closely for any vibrations, and if such occur to stop them by stopping the drill."—A. Zboril.—Transactions South African Association of Engineers, March, 1907, p. 229. (J. Y.)

State Aid to Mining.—"Genuine assistance rendered by a State must ultimately lead to expansion, whilst probably benefiting a particular district; indirectly it is of great advantage to any Colony, since it frequently happens that by timely and judicious help promising mines are opened up. I propose in this article to touch briefly on State assistance given by the Government of West Australia to the mining industry, thereby showing how a large measure of prosperity is directly attributable to the foresight of former Administrations.

It is not suggested or proposed that similar methods should be adopted in this country; still, should the time eventually arrive when the mining conditions in the Transvaal are changed, State grants on modified lines may be instituted in districts eminently suitable for small mining ventures. One eliminates the Rand entirely, since the conditions prevailing render it necessary that large sums of money must be laid out before it is possible for a mine to arrive at the producing stage.

In West Australia State aid is granted in three ways:—

(1) The erecting of treatment plants or the subsidising of private mills for the benefit of small mine owners and others who have claims, but not sufficient capital to erect a reduction plant.

(2) The supplying of water to outlying fields away from the main Government supply scheme.

(3) Grants in cash to struggling companies or individual mine owners in order to test their mines in depth, the proving of which is likely to benefit the State.

Dealing first with No. 1, a State battery is erected provided the Minister of Mines is satisfied that:—

(*a*) Large deposits of metalliferous ore exist.

(*b*) A plant to treat them at a reasonable rate is not available.

(*c*) The establishment of such a plant is necessary for the development of mining.

It is sometimes difficult at first sight to decide whether there is or is not in the neighbourhood a battery available for the purposes of public crushing; the position then occurs that there are batteries in a district which are privately owned, and which will crush for the public at such times when it suits them, that is, when they have not sufficient ore to keep their mills going. In this case it has been found necessary either to erect a State mill or subsidise the owners of the private batteries to ensure their being willing to crush ore for all comers at reasonable rates.

Certainty of crushing is indispensable to the small mine owner, in order to enable him to develop his claims with any degree of confidence. Furthermore, the charges of private mills are sometimes so high that the general grade of the ores of a district will not allow of any margin of profit to the mine owners, hence a general reduction is necessary in the best interests of the community. As this lower rate is often not sufficient to pay a fair rate of intest on capital and redemption in addition to working costs, a privately owned concern cannot, as a matter of business, afford to reduce its crushing charges. Therefore, in practice, it is usual for the State to grant a subsidy to the private mill equal to the amount required to pay interest on capital and redemption on the condition that the crushing charges are reduced to the actual working costs.

The rules laid down by the Government are as follows:—

(1) All ores for treatment to be estimated at 22 cub. ft. per ton.

(2) The manager at the battery to have power to refuse any stone considered too poor to pay crushing charges, unless a deposit is paid in advance.

(3) The charge per ton for crushing at each mill to be determined from time to time by the Minister of Mines.

(4) A minimum charge of £5 to be made for parcels of 6 tons or under.

(5) Payment for crushing to be made to the manager on completion of treatment, all gold to be held pending payment of amounts due.

With regard to tailings, conditions are somewhat different, and, in order to avoid complications (it not always being a payable matter to test small quantities), certain rules have been made which on the whole have proved satisfactory.

(1) The manager of the battery can refuse to purchase tailings should he consider them unsuitable for treatment.

(2) The tailings from the mill are settled in pits, and three samples taken, one for assay by the manager, one for the owner, and a sealed packet for future reference.

Should any dispute arise and no agreement be come to, the sealed sample is sent to the Government Assayer, his result to be accepted as final.

(3) Upon the value of the tailings being determined, the owner is paid within one month 50 per cent. of the assay value, based upon an extraction of 75 per cent., less treatment charges, the balance to be paid within three months.

(4) A charge of 10s. per ton for treatment to the first charge on all sands treated, the gold contents being paid for at the rate of £4 per oz.

(5) All tailings to be calculated on the basis of 27 cub. ft. per ton.

(6) Slimes are not paid for (except where slime plants are available), and cyanide treatment is restricted to certain classes of ore.

A typical Government treatment plant is that at Mulline; it is equipped with two Cornish boilers 18 h.p. each, 20 head of stamps, 10 of 850 lb. and 10 of 1,000 lb. each. A compound condensing engine, four Berdan pans, four pumps for various uses, eight pits to catch the sands from the battery, slimes settling dam, 7 in. Cornish lift for pumping slimes, winch for hauling sands from pits to cyanide vats. The cyanide plant consists of 5 sands vats of 31 tons each, capable of treating 800 tons per month, four vats of a total capacity of 10,000 gallons for solutions, two extractor boxes, two 3,000 gallon sumps. For the slimes treatment there is a 14 h.p. portable engine and boiler, two 44 ton agitators, one vortex mixer, one Fernier spiral pump, one extractor box with 3,000 gallon sump, one Snow Pump, one 3-throw pump for press work, and one 4-ton filter press. There are also six 16 cub. ft. trucks and three sumps of 3,000 gallon capacity. In addition, connected with plant is a 5 ton weighbridge, office, store room, smith's shop and assay office. Up to June 30 this plant had treated 47,756 tons for a return of 61,311·25 oz., valued at £219,900. The charges are as follows, being based on the value of the ore :—

Milling.—

Up to 8 dwt.	...	10s.	per ton.
From 8 to 9 ,,	...	10s. 6d.	,,
,, 9 to 10 ,,	...	11s.	,,
,, 10 to 11 ,,	...	11s. 6d.	,,
,, 11 to 12 ,,	...	12s.	,,
,, 12 to 13 ,,	...	12s. 6d.	,,
,, 13 to 14 ,,	...	13s.	,,
,, 14 to 15 ,,	...	13s. 6d.	,,
Over 15 ,,	...	14s.	,,

Cyaniding.—10s. per ton. Filter pressing extra, actual cost only being charged.

There are working in different parts of the State 29 batteries, the majority averaging 10 stamps each ; up to June 30, 1906, 358,203·49 tons had been treated for a return of 404,974·23 oz., valued at £1,493,154.

In addition to these, the Government have erected a tin dressing plant at Greenbushes, which has treated 11,557·5 tons of ore for a yield of 287·145 tons of black tin (SnO_2). Also a small smelter has been erected on the Phillips River field for the treatment of the copper ore obtained there.

The water supply is undertaken by a special branch of the mines department under a supervision of a competent engineer. When a new district is discovered, and there is a rush to the spot, a party is dispatched to locate a supply of fresh water for domestic purposes, wells are sunk, and, if possible, a permanent supply is obtained, both for human consumption and for mining uses. Brief mention should be made of the gold fields water supply, by which Kalgoorlie and its mines are supplied with water pumped from the Darling Ranges, near the coast. The water is pumped through a 30 in. pipe line a distance of 352 miles, a supply of 5 000,000 gallons per day being available. The total cost of this scheme was £2,670,000, and the mines are supplied at a cost of 5s. per 1,000 gallons. This undertaking has been of the greatest advantage to Kalgoorlie, since it has ensured an ample and continuous supply, compared with what was previously a more or less intermittent one, which depended on a few wells and condensing plants.

The third class of assistance referred to at the beginning of this article is not given very frequently, it having generally been proved to be attended with unsatisfactory results. The Act states that advances should only be made when, in the opinion of a responsible Government officer, the prospects are so good that he is able to recommend a loan with confidence. In this case it is obvious that the investing public would doubtless be quite ready to advance the necessary cash without the owner having to apply to the Government.

The chief means of assistance has always been the State batteries, their usefulness having been proved beyond question. Many have been run at a loss, and many have had to be removed, yet, in spite of this, a number have proved of great benefit to the State. This class of enterprise has shown itself to be peculiarly adapted to the country ; there being many cases in which a central battery can be fed by a number of mines in its neighbourhood, and as means of communication are fairly easy, the country has benefited to a much greater extent than can be shown by a mere profit and loss statement. These batteries have been the means of keeping alive fields that would otherwise be abandoned, and of reviving others that were considered worked out. It has permitted the exploitation of low grade reefs, which could in no other way be profitably worked by the small mine owner (without sufficient money to erect a plant), and it has enabled such men to hold on to claims by giving them a ready and cheap means of paying for development from production.

The objection has been made that these public plants interfere with private enterprise. This does not hold good, since the State has never attempted to compete with private mills, but has only sought to encourage prospectors to give fields a trial which would probably never have been developed unless some means of crushing had been available. It was never anticipated that these plants as a whole would be worked at a profit ; but they have proved an important factor in adding to the wealth of the country, opening up new gold fields, and have assisted in further consolidating the gold mining industry."—W. BROADBRIDGE.—*South African Mining Review*, March, 1907, p. 92. (J. Y.)

AIR HAMMER DRILL.*—"In the large cross-cut or railway tunnel, the large piston drill still holds the advantage, as it is possible to drill deep rounds and break the ground far more economically and rapidly than would be possible with the small hammer type. Even in this class of work, however, there is a place for the small drill, as they can be used most successfully in block holing and in taking up bottom or trimming up the walls where projections are left, either on account of missed holes, or because the drill runner has miscalculated the load he had on a certain hole. They can also be used for drilling uppers in the roof for pipe hangers, or for putting in short relief holes in the heading where they will give the cut holes a better chance to break.

In shaft or winze work, however, they can be used for drilling the full round of holes and even in the larger sized shafts, records show a saving which is simply astounding. To those who are familiar with this class of work, the reason is obvious. With piston drills, even in the hands of the best of runners, more time is lost in setting up, tearing down, lining up, cranking in and out of holes and in changing steel, than is consumed in drilling. Where the ground is even and breaks well, it is not so bad, but where slips are encountered frequently, and time is consumed in preventing a 'hang up,' or where the formation is such that a great many holes must be drilled to break the ground, the air-hammer drill has all the best of the argument.

The machine, steel and hose, can all be put into one bucket, and the drill runner can be at work in five minutes from the time he goes on shift. The steel being loose in the chuck, he loses no time in changes, and he can start a new hole while a piston drill runner would be cranking out. Holes can be drilled in any direction and pointed to take advantage of any slip. No holes need be lost, as it is practically impossible to get a 'fitchured' hole even in the worst kind of ground. All of the holes can be drilled to any depth desired, up to 4 ft., except in very heavy sulphide ore, and the sump holes can be put in in any part of the shaft.

With a piston drill, the runner may get a hole down 3 ft., and then get hung up, and rather than lose time he will take a chance on breaking the ground and start another hole. On this hole he may have no trouble and he will perhaps drill it 6 ft. deep. What is the consequence? When the shaft is cleaned up he will find he has no sump, or one corner

* See this *Journal*, p. 59, vol. vii.; Aug., 1906.

of his shaft sticks out so that he is unable to get a set up which will enable him to drill his round to advantage. He may even have to drill a hole by hand and shoot it before he can get in to do any good with his next round. As a consequence, the shifts are thrown out and valuable time is lost. When timbering commences, the small hammer drill is indispensable for cutting hitches and taking out projections where they interfere with the sets, and for these purposes alone, will pay for itself in 30 days.

In a drift, a drill runner can either mount the air-feed drill on a column or, where it is possible to pull the cut out of the top, he can work with the air feed alone, and in any ordinary ground, one man can drill a round of holes in half a shift. Where the cut is pulled from the centre or bottom, he can either drill the back and top cut holes off a bar or simply use a stick of timber and butt the air feed against it. I have known of one man breaking 42 ft. in 19 shifts in a drift in granite, and not only did he do all the drilling alone, but the tramming as well.

There is a diversity of opinion as to whether it is advisable to use a $2\frac{1}{4}$ in. piston drill or an air-hammer drill in this class of work, and a decision, from an economical standpoint, can only be obtained by actual tests. As a usual thing, however, the hammer drill can be used advantageously in all drifts where a 4 ft. round, bottoming $1\frac{1}{8}$ in. will break without leaving any 'guns.' With a piston drill of the size mentioned or larger, it is possible to make more rapid progress, but the cost per foot will probably be greater, as much more powder and air will be required, and it is a hard matter for a drill runner to set up and tear down without some help from a timberman or trammer. In ground which breaks short, however, the piston drill usually has the advantage as a larger hole is drilled, and the powder can be gotten to the bottom of the holes where it belongs.

Cost of Stoping.—In stope work the $2\frac{1}{4}$ in. machine does not have a chance against the air-hammer tool, and the records on some of the Cripple Creek properties show that their stoping costs have been introduced one-half since the introduction of the small machines. Where the values lie in small streaks of very high grade ore, the saving is enormous, as the stopes can be carried even narrower than by hand, as the drill can be run in any place a runner can get his body.

I installed a small machine for a leaser in Cripple Creek some 18 months ago, and he informed me, after his lease had expired, that the little drill was worth $1,000 a month to him. He was stoping on a 14 to 18 in. streak of 4 to 5 oz. ore, and found that the ground was too hard to break by hand. He then put in a $2\frac{1}{4}$ in. machine, and was forced to carry his stope $3\frac{1}{2}$ to 4 ft. wide. In doing this he lowered his values to $30 to $35 per ton. After the installation of the little air-feed drill, he was able to break one-third more ore in an 18 in. stope than he had with the $2\frac{1}{4}$ in. machine in one $3\frac{1}{2}$ to 4 ft. wide. He ran his values back again $80 to $90, cut his hoisting and powder bills down to one-half, and his timber to one-third, and yet kept his production up to more than the tonnage he was able to get back of the $2\frac{1}{4}$ in. drill. With the piston machine his air cost him $3 per shift, while with the hammer drill the charge was but $1 for the same length of time. He figured the difference in net returns and found that the small machine saved him a little more than $1,000 per month. The usual method of operating the machine in an open stope is to put in a couple of light sprags well up to the back, and about 10 or 12 ft. apart. By putting a 2 x 12 in. plank on the timbers and drilling from it, a drill runner can fill a stope full of holes in half the time he would consume with a larger drill. Not only can he drill more holes, but he can drill them in any place he desires and take advantage of every slip.

The same method is used in upraising, and data furnished by some of the largest mines in the west shows that a saving of 50 per cent. can be readily given as the average.

Repairs.—Not only can more work be done with the hammer drills than with piston machines, but the saving in repairs is well worth considering. In nearly all of the small hammer drills, there is but the one moving part, and when properly made this should last for months. The rotation being accomplished by hand, there are no pawls or springs to break and no rifle nuts to wear out. As the machines are fed by air, no feed screws or nuts are required. Taken as a whole, with a properly made air-hammer drill, the repairs are less than 25 per cent. of those of a $2\frac{1}{4}$ in. machine.

Working at a drill pressure of 100 lb., a $2\frac{1}{4}$ in. machine requires 100 cub. ft. of free air per min., while an air-hammer drill of the valveless type uses but 25 cub. ft. at the same pressure.

Another advantage possessed by the hammer drill lies in the fact that it is not necessary to employ a first-class drill runner, as a good miner who knows how to point his holes will do just as much work after two or three days' experience, as the best man one could hire."—H. L. SINCLAIR.—*Engineering and Mining Journal*, April 13, 1907, pp. 715—716. (A. McA. J.)

CLASSES OF EXPLOSIVES.—"As the outcome of their practical experiments and after a careful consideration of all the facts, the departmental committee, presided over by Mr. H. H. Cunynghame, C.B., recommended that the Permitted Explosives should henceforth be divided into two classes for the purposes of the Explosives in Coal Mines Order:—

Class I.—Such explosives as shall have passed the existing Woolwich test and the further tests to be hereafter settled.

Class II.—Such explosives as shall have passed the Woolwich tests only.

They consider that the use of explosives, other than those in Class I., except under certain special provisions, should be prohibited in all mines and seams to which the Explosives in Coal Mines Order applies. The conditions under which those in the second class may be employed are that the mine in question is either naturally wet throughout or is effectually watered or damped artificially, so that no dry dust may be present; or that such seams are divided by zones of 100 yards, or some suitable distance, in which the floor, sides and roof are effectively watered or damped artificially; or, lastly, in mines from which all workmen have, with a few exceptions, retired during blasting operations with Class II. explosives. Indications are given by the committee of the general principles on which the further tests, in addition to the Woolwich test should be carried out. These involve practical experiments in a large gallery, similar to those in use in Germany and Belgium, and tests on coal dust in a disused mine, and they advocate the addition of a testing station at Woolwich for testing coal dust and safety lamps, and the possible use of methane for testing purposes. It will thus be evident that the report is of a very far-reaching and weighty character, and will require special and immediate attention."—*Times Engineering Supplement*, April 17, 1907. (J. A. W.)

NEW MINE FAN.—"A practical test will shortly be made at the Birthday Tunnel mine, Berringa, Victoria, of a machine, recently invented by Mr. J. J. Gilday, an experienced miner, for the ventilation of mines. The machine has a cylinder 3 ft. in diameter and length, with a 16 in. pipe, which descends into the shaft. This cylinder contains six blades or vanes, which work on the screen propeller principle, beginning with a fine pitch which gradually becomes coarser. To allow for the compression of the air in entering the machine in a rarified state, the centre outlet has been made in a cone shape and tapers back to the inlet. The air after entering the inlet, which is of greater size than the outlet, is slowly moved by the fine pitch of the vanes, until the gradually coarsening pitch of the vanes, in conjunction with taper cone, ejects the air at a high velocity. With a $6\frac{1}{2}$ h.p. machine, working $4\frac{1}{2}$ in. of water gauge pressure with a velocity of 8,060 ft. per minute, through a $15\frac{3}{4}$ in. pipe, 11 900 ft. per minute of air can be drawn off."—*Australian Mining Standard*, Feb. 13, 1907, p. 143. (W. A. C.)

MISCELLANEOUS.

THE ANALYST AND THE MEDICAL MAN.—". . . I venture to say a few words on the relations, past and future, between the analyst and the medical man. There is no need, I think, to repeat the indisputable statement that the doctor and the analyst share between them—not forgetting the work of the sanitary engineer—almost the entire burden of the maintenance of public health. It would be but repeating the obvious, moreover, to say that these two professions, having such common aims, should possess full sympathy and mutual understanding. But although we may believe that the sympathy is always present, yet the understanding is not quite always complete, and the lack of it has, now and again—we rejoice to know, not often—led to some friction at points of contact. When such small differences arise it must be admitted that the medical profession has this great advantage over yours—that its claims and its merits are so much more easily understood by the general public. This, should a need for judgment arise, somewhat unfairly strengthens the hands of the one disputant.

It must, unfortunately, be recognised that, even now, after all the years during which the Public Analyst has served the community, the layman has a most imperfect knowledge of what his work entails. The services of the engineer are understood, those of the doctor are appreciated, but the skill expended in the analytical laboratory is understood and appreciated but little. This, to you, is again a commonplace; but I believe I am justified in adding to it the really regrettable statement that even the average medical man is not greatly different from the man in the street in this respect. He has at one time known a little chemistry, but he possesses no standard to measure what is required of the skilled analyst. It should be understood that I speak of average cases only.

I am going to take the risk of making a bold generalisation here which I must justify afterwards. I believe that, of those professional men who in any sense apply science to practical affairs, the average doctor is among the least scientific, whilst the professional chemist is among the most scientific. I hope I am not making this statement gratuitously, and I am certainly not urging the contrast to ingratiate myself here, or to cast a slur upon the medical profession. There are many in the latter who are using high scientific knowledge, and applying it in the most difficult of all regions; the treatment of disease; but nowadays the medical man with a real scientific bent tends to cleave off from the profession, as a pathologist or physiologist. It is even the fashion, once more, for distinguished medical teachers to urge upon their students that the art rather than the science of their calling is the important thing; and we must all, indeed, recognise that in practical medicine, skilled and wise empiricism, based upon experience, is a much better endowment than ill-digested science. The progress of medicine depends upon science, but its practice does not demand scientific interests, properly so called, on the part of the practitioner. You must not, I think, expect that your work will be understood, or fully appreciated, by the doctor merely because he has passed through the medical curriculum." . . . .

"The Medical Officer of Health is a man whose education has been wide and various, and who has submitted himself to tests of efficiency which are without doubt rigorous. But although, among many other things, he has been taught more chemistry than the less specialised members of his profession, I venture to think that even *he* does not—at any rate, at the outset of his career—possess enough chemistry to appraise the high claims of the skilled analyst or to gain the full respect for the latter which comes with knowledge. He does not always recognise, as he should, that the consulting chemist and the analyst belong to a profession in which education is as prolonged, and in which personal ability and skill are as essential, as in his own. I feel that this lack of knowledge on the side of the medical officer is certainly one of the causes leading to misunderstanding. The Public Analyst on his side should not, of course, forget that the Medical Officer of Health is a man of multifarious and responsible duties, whose accomplishments are not to be measured by his knowledge or ignorance of special laboratory problems; he must demand only that these laboratory problems be viewed with a due sense of proportion." . . . .

The writer then discusses the question of the performance of water analysis by M.O.H., and then proceeds:

"But the existence of these exceptional cases is beside the mark. To the normal medical officer, even if he be possessed of some skill as a chemist, the chemical laboratory and its pursuits are but an accident. He can enter it only seldom, and if he come into it sample in hand it is usually to find standard solutions which time and accident have made unreliable, and apparatus untested for weeks or months. Nine times out of ten, therefore, the analytical effort is a trying one, and if the medical officer spend the time necessary for accuracy in a laboratory which is not strictly a *going concern*, the enterprise is to him unprofitable. He should realise how much simpler it all is to the professional analyst in constant practice." . . . .

"But into the controversy about water analyses there came some years ago a certain *tu quoque* argument, and to the disinterested outsider the position was even a little amusing. The examination of water is only partially complete without a bacteriological study; and though it must be admitted that our power to dogmatise about the details of contamination has not yet been aided by bacteriology to the extent that was hoped for, yet it is certain that the water expert must be, on special and limited lines at any rate, a bacteriological expert. But in bacteriological study the medical man for the most part preceded the analyst, and "Hands-off!" at first seemed to him as legitimate an expression when uttered on his side of the boundary as when used on the other.

The technique of bacteriology on its cultural and merely diagnostic side is much simpler, more limited, and more empirical than that of analytical chemistry; but the former subject, like the latter, requires something more than a formal knowledge of *technique*. It is the development of certain special instincts, only given by long contact with the problems, which converts the amateur into the expert. The ordinary training of the medical officer or of the analyst gives in neither case this endowment as regards bacteriology. There is, however, I believe, this fundamental difference between the intrusion of the Public Analyst into bacteriology and that of the medical officer into chemistry: the analyst is first, last, and always a laboratory man, while the medical man is not. The six years of laboratory life which must elapse before the professional chemist is looked upon as fully qualified make the attempt upon new laboratory ventures an easier task for him. This confinement to the laboratory is, on the other hand, the cause of some distinct disadvantages to the analyst. The four walls of his workshop hide him from the gaze of the public; he triumphs over difficulties in secret, and without appreciation. But he should at least reap the advantage of a recognition, from those who know, of the fact that his constant practical experience at the laboratory bench endows him with fundamental instincts for laboratory work in general, which the occasional visitor to the laboratory can only very exceptionally possess." . . . . .

"If in the future the Public Analyst is to be reminded by other specialists that his business is chemistry, the medical officer will be treated on similar terms, and I have firm faith myself that an increase in the analyst's activities, on purely chemical lines, will leave him well content, as I hope in some sort to show later.

There are, it seems, some grounds of complaint against medical officers in other and, to my mind, more serious connections than those yet dealt with. It has occurred that the Medical Officer of Health of a district has acted as though he were the superior of the Public Analyst, in the sense of possessing a right to deal with the reports of the latter, publishing them as though they emanated from a mere departmental officer under his control. Such a distortion of fair conduct, or anything analogous to it, must be and remain rare, and need not be discussed as though it pertained to the normal. It could arise only from the existence of a complete misunderstanding of the situation. I am sure that every reasonable medical officer of health, recognising that the analyst's appointment is a direct one and coequal with his own, would wholly repudiate such a course. In such a case it would be well to see that the facts came to the knowledge of the medical press, in which, I am sure, it would meet with right and effective comment.

Leaving for the moment any further reference to the difficulties which have arisen between doctor and analyst, let me now proceed to the more satisfactory task of indicating future developments which may tend to bring them together.

The average medical man while not yet aware of all that chemistry can do for him, is also, for reasons already indicated, not yet clear as to whom he should turn to for help in those chemical problems which he has, even at the present time, in mind. I have read recent elaborate medical monographs, involving extraordinary futile chemical investigations, in which the author, himself innocent of chemistry, acknowledges his obligations to the pharmacy which is adjacent to him. Now, it is no reflection upon an honourable calling to suggest that in such a case the doctor ought to have gone elsewhere. One can see that in these conjoint researches it was the good nature of the pharmacist, and no worse motive, which made him consent to the unprofitable partnership. But I believe it is not going too far to say, that the rise of chemical pathology to its full importance will call almost for a new profession.

I wish very much to make a point here which, if it seems too remote at present from your practical interests, may be considered as in parentheses. The care of the body in sickness, with all the delicacies of human relationship which it involves, must remain always an entire and carefully guarded prerogative of the physician; but the innate respect of the public, and even of the non-medical scientific public, for the physician's calling has led to a somewhat illogical attitude, and has tended to make sacrosanct not only the calling of the physician, but the scientific material which he deals with. There has been, as it were, an averting of the gaze whenever a region of knowledge is stamped as "medical." Now, I am certain that the progress of scientific medicine demands a change here. While a large part of future scientific medical studies must always be carried out by men who, though medically qualified, have preferred the laboratory to practice, and whose special qualification, therefore, is that they have had personal touch with the problems offered by disease, yet in a middle region these must be joined in their work by men whose primary qualifications are non-medical—men who, saved from the long years of clinical study, are able to bring well-grounded laboratory knowledge and (I may add) a sufficient knowledge of the literature of pathology, which is open to all, to join their medically qualified *confrères* in attacking the huge problems which await solution.

While upon the business of prophecy, I am tempted to put another series of prognostications before you, the credibility of which is at the present time, perhaps, more obvious to the physiological chemist than to anybody else. I pass from pathology to an aspect of dietetics. This is a subject in which the medical man is the recognised authority, charged with instruction of the public, but for a scientific knowledge of which he depends largely on the chemical physiologist and the analyst.

Putting on one side the aspect of affairs which especially concerns this Society—the maintenance of purity and freedom from adulteration—and leaving out questions such as digestibility and the like, the chief practical points which have hitherto been considered in relation to the daily rations of mankind are the total energy value requisite for maintenance, the optimum ratio of fats and carbohydrates, and the optimum supply of protein. Now, these questions have recently received fresh attention, and experimental work has been done lately yielding, as you know, somewhat startling results, tending at first sight to modify our views concerning maximal, minimal, and optimum dietaries. But I am not going to discuss the work of Atwater or Chittenden, proposing rather to put before you very briefly facts of another sort, less known and seemingly academic. I believe, however, that my theme, which is that of the influence of minimal qualitative variations in dietaries, will one day become recognised as of great practical importance.

Physiological chemistry, chiefly owing to the work of Emil Fischer, has recently gained the knowledge that individual proteins, and among them those which contribute to human dietaries, may each bear a special chemical stamp; that a given protein may differ so widely from another protein as to have, quite possibly, a different nutritive value. I will illustrate this, first of all, by a somewhat extreme case. A protein, zein, forming no inconsiderable proportion of the total nitrogenous constituents of maize, is entirely deficient in at least one characteristic molecular grouping. It yields on digestion no tryptophane, the product which represents the indol group present in the molecule of most typical proteins.

Recently we have fed animals with this indol-free maize portein in such a way that it formed the only supply of protein, though associated with abundant fat and carbohydrate and suitable salts. The diet wholly failed to maintain tissue growth in young animals, which, however, grew at once when their zein was replaced by pure casein. When tryptophane was added to the zein diet, there was still inability to maintain tissue growth, doubtless because the zein has other deficiencies as a protein. But now an interesting fact came to light. The animals which received the missing indol derivative in addition to the zein did not grow, in fact, continued to lose weight daily, but were afterwards in much better health than, and long outlived, those which had the zein alone. These experiments seem to show two important facts: First, that in an extreme case a particular protein may wholly fail to support life, just as is the case with gelatine; and next, that a group in the protein molecule may serve some purpose in the body other than that of forming tissue or supplying energy. The usual discussions about food-stuffs attribute to them these two functions only—repair of the tissues and energy supply. But the body has other and more subtle needs equally urgent. Here, there, or elsewhere in the organs must appear special, indispensable, active substances which the tissues can only make from special precursors in the diet.

The indol grouping in the protein molecule serves some such special purpose, quite distinct from its necessary function in tissue repair. This matter of qualitative differences in proteins may be of no small significance in dietaries. It may account for what I believe is proved by experience—that rice may serve the races which rely upon it as an almost exclusive source of protein, while wheat is only suitable for races that take a much more varied dietary. It may explain many variations in nutritive values which at present we feel and recognise only vaguely. In the future the analyst will be asked to do more than determine the total protein of a food-stuff; he must essay the more difficult task of a discriminative analysis.

But, further, no animal can live upon a mixture of pure protein, fat, and carbohydrate, and even when the necessary inorganic material is carefully supplied the animal still cannot flourish. The animal body is adjusted to live either upon plant tissues or the tissues of other animals, and these contain countless substances other than the proteins, carbohydrates and fats.

In diseases such as rickets, and particularly in scurvy, we have had for long years knowledge of a dietetic factor; but though we know how to benefit these conditions empirically, the real errors in the diet are to this day quite obscure. They are, however, certainly of the kind which comprises these minimal qualitative factors that I am considering." . . .

"Scurvy and rickets are conditions so severe that they force themselves upon our attention; but many other nutritive errors affect the health of individuals to a degree most important to themselves, and some of them depend upon unsuspected dietetic factors.

But am I at present justified in troubling you, as practical men, with such matters—you who are interested in professional chemistry, and not in what is still more or less academic physiology?

I have been led to do so from two considerations. First, it is abundantly clear that the foundation of future progress in chemical pathology and dietetics on the lines I have been indicating calls for large efforts in purely analytical chemistry—efforts which have been too long delayed. And the delay has arisen from a circumstance of no small interest and importance.

I see abundant reasons for believing that in the near future events will march to the consummation of mutual appreciation and helpfulness, and to the disappearance of all misunderstanding, in the relations between analyst and medical man."—Read before the Society of Public Analysts by F. Gowland Hopkins, M.A., M.B., D.Sc., F.R.S.—*The Analyst*, Dec., 1906, p. 386. (J. M.)

The Hæmatology of Carbon-Monoxide Poisoning.—"That the poisonous nature of coal or illuminating gas is due to the carbon monoxide which it contains has long been known. Haldane has shown that the hæmoglobin of the blood has an affinity for carbon monoxide about 140 times as great as that which it has for oxygen. He has also placed beyond doubt the fact that carbon monoxide is not used up or oxidised in the body but is excreted by the lungs unchanged, and concludes that carbon monoxide acts as a poison purely in a negative way by uniting with the hæmoglobin of the blood, thus throwing it out of commission as an oxygen carrier, and thereby preventing normal oxidation in the tissues.

It was with the object of ascertaining what the ultimate action of carbon monoxide was upon the body, and whether it played any great part in producing anæmia, which has so often been associated with leaking gas pipes, and sometimes found in persons breathing gas-contaminated atmosphere, that this investigation was undertaken. Moreover, it was hoped that we might arrive at some conclusion as to why some individuals subjected to acute coal-gas poisoning recover rapidly and suffer little after-effects, while others linger for three or four days and finally succumb. Our experiments have been made throughout on guinea-pigs.

Chronic Coal-Gas Poisoning.

At the beginning of this work the animals were placed in a cage and a mixture of air and pure carbon monoxide passed through the cage until 50 per cent. of the hæmoglobin of the guinea-pigs was saturated with gas. As guinea-pigs just retain consciousness with 50 per cent. saturation they were kept at this point for one hour and then removed. Such doses having no apparent effect when repeated daily for weeks, either on the blood, weight or health, another method was resorted to. Twelve guinea-pigs were taken—six males and six females—and placed in two cages in the gas chamber after making careful estimations of the white and red blood corpuscles and hæmoglobin. Gas was then allowed to mix with the air drawn through the chamber until the mixture was of such a strength that 25 per cent. of the hæmoglobin of the guinea-pigs was saturated with carbon monoxide.

The weight of the animals was occasionally noted, and at intervals one of them was taken and killed, to examine for possible pathological changes which might have taken place.

Of the twelve guinea-pigs originally placed in the gas chamber fifteen months ago, two are still apparently as healthy as ever, one died of old age after two months' confinement, four died at one time through the carelessness of the attendant, one after seven months was transferred to fresh air again, and five have been killed for pathological study. Other guinea-pigs have been put in and taken out of the gas at various times for confirming certain points.

None of the animals decreased in weight—on the contrary, most of them gained. After the first few days, those living with 25 per cent. of their hæmoglobin saturated with CO appeared just as active and happy as those living in the air.

It was naturally expected that with only three-fourths of their hæmoglobin available for oxygen carrying purposes the animals would fail to resist and that symptoms of anæmia would follow. The reverse process actually took place, and to our surprise the animals succeeded in resisting. They, in short, became acclimatised.

With 25 per cent. of its blood rendered useless for oxygen carrying purposes by its union with CO, the guinea-pig is capable of compensating, and will manufacture new red corpuscles until it has reached a total of about 8,000,000, with a corresponding hæmoglobin content of 105 per cent. The normal number of erythrocytes in the peripheral circulation of guinea-pigs is about 6,000,000, with 88 per cent. hæmoglobin. The animal with true compensation has three-fourths of 8,000,000, or 6,000,000 erythrocytes, and three-fourths of 105 per cent., or 79 per cent. of its hæmoglobin still available. Thus there is only a total loss for oxygen carrying purposes of 9 per cent. of its hæmoglobin.

The guinea-pig responds very rapidly in its efforts to counteract the effect of the CO, and will manufacture 2,000,000 additional corpuscles per cubic millimetre in three or four weeks.

The effect of depriving an animal of the use of part of its hæmoglobin by allowing it to unite with CO is also in many respects similar to depriving it of part of its blood by bleeding. In the latter case the blood is rapidly renewed by the activity of the hematopoëitic organs, as evidenced by the appearance of normoblasts and the increasing quantity of erythrocytes and hæmoglobin.

As the erythrocytes of normal blood are being used up and replaced continually, and even under ordinary circumstances show these various stages in their life history, so this process is constantly taking place only more rapidly in the gas. The life history of the cells in the latter case is shorter because they are living under conditions where the normal metabolism of the cells is interfered with.

From what has been said above it is clear that the effects of chronic CO poisoning in the blood is similar to that which occurs at high altitudes. It is true that in the latter case nucleated red blood corpuscles have not been found, but we think the only reason for this is that the ascent has not been made rapidly or high enough. The similarity is obviously explained by the view that a lack of oxygen is the chief cause in both cases of the changes in the blood.

The Effects of a High Hæmoglobin Saturation on the Red Blood Corpuscles of Guinea-Pigs.—As it was of physiological interest to determine how high a percentage of saturation guinea-pigs can stand continuously, the gas was increased so that the blood attained a saturation of 35 per cent. After a couple of weeks, when the erythrocytes had increased to 9,000,000, the saturation was raised to 45 per cent. The animals looked ill for a few days, and a collapse was feared, more especially as they seemed to lose their appetites. In four or five days they again began to eat heartily, and gradually regained their normal appearance; examination of the blood showed a vast amount of degeneration of the erythrocytes, and large numbers of normoblasts appeared. The reds rapidly increased, however, and in three weeks the blood of the four guinea-pigs at this saturation showed 10,500,000 erythrocytes per c.mm. The corresponding hæmoglobin content was then 110 per cent. The blood was so thick that it became increasingly difficult to obtain good smears, owing to the fact that the blood would not spread on the glass slides.

To show what the effect would be on a normal animal when put in an atmosphere which would cause 45 per cent. hæmoglobin saturation, one was placed in a cage with the other four acclimatised guinea-pigs. It died in four days. As the animal constantly shivered and showed every indication of being cold, another guinea-pig was put in a cage with a current of warm water circulating below the pan of sawdust in the bottom. This kept the temperature quite warm, but the animal died in three days with exactly the same symptoms, except that it did not appear to be cold. A third guinea-pig was put in with the other four, and showed all the typical symptoms up to the fourth day. When removed to the air it could not move, and lay on its side for a couple of hours, apparently dying. It then began to improve, ate a little food, and after seven to eight hours was put back in the gas box. For a few days it appeared ill, and became very thin, but eventually recovered. In four to seven days from its entry into the gas the red blood corpuscles showed such an amount of degeneration that it was difficult to understand how the animal could possibly maintain life. If, therefore, a guinea-pig were taken into an atmosphere containing 7 per cent. oxygen, or elevated 28,000 ft. above the sea level, we should expect it to die in about three or four days, since this height corresponds to a hæmoglobin saturation of 50 per cent. As a matter of fact, it would probably die long before this time on account of the intense cold, dilation of blood vessels, and other effects resulting from rarefied atmospheres.

Finally, two guinea-pigs were placed in gas of such strength that they were rendered unconscious. After being kept in that state for some time they were removed, and the blood examined from time to time. In these cases of acute poisoning—that is, when the animals are rendered unconscious, and kept so for two or three hours—the blood picture differs from that of the chronic gas poisoning in two particulars. On the one hand, the oxyphil leucocytosis is much greater in the acute cases, and, on the other, this oxyphil leucocytosis is in itself of a different character. The oxyphil granular forms increase from 23 per cent. to 89 per cent. in the second example, while the lymphocytes decrease from 70 per cent. to 7 per cent. The coarsely granular form or eosinophil, about five hours after the removal of the guinea-pig from the gas, totally disappears for six or seven hours, and then gradually regains its former number.

From the results obtained by us it seems quite apparent that lack of oxygen does produce auto-intoxication, and that, therefore, gas-poisoning is really due to a toxæmia caused by lack of oxygen, the CO itself being merely the substance which prevents the oxygen from reaching the tissues.

It is known that many, if not all, poisons cause a destruction of tissue, which destruction means production of poisons from the tissues themselves. All of these poisons affect the central nervous system, and thus produce the abnormal symptoms of poisoning. We have in gas-poisoning, at any rate, an indication, as evidenced by the severe leucocytosis and degeneration of the blood produced, that poisoning due to retarded metabolic activity has taken place. When this abnormal metabolism is continued for a length of time, especially when the saturation of the blood with CO is high, the effect on the system is disastrous. The higher the saturation and the longer the time involved the greater will be the damage resulting to the body cells at large, and to those of the central nervous system in particular. It is probably for this reason that cases of poisoning produced by a rapid saturation of the hæmoglobin with CO and continued only for a short time recover rapidly when by fresh air and artificial respiration the gas is got rid of. When the inhalation of the gas has continued for a longer time the products of incomplete combustion in the body cells become so large, and the consequent damage to the tissues so extensive, that the disastrous effects upon the body prove irreparable, and perhaps several days after the gas has disappeared from the blood the patient will succumb. Taking the blood itself—a liquid tissue—as evidence, we see how extensive their degeneration may be.

Chief Conclusions.

1. Carbon monoxide acts as a poison solely by its ability to prevent the normal supply of oxygen from reaching the tissues, and thereby deranging the normal metabolism of the body cells.

2. Guinea-pigs living continuously in a dilute carbon monoxide atmosphere so that the oxygen carrying capacity of the blood is reduced, are able, by increasing the quantity of hæmoglobin and number of erythrocytes, to compensate for this loss, and maintain an oxygen carrying capacity approximately equivalent to that of the original blood.

3. Carbon monoxide poisoning is followed by a leucocytosis of the eosinophil and pseudo-eosinophil forms. A moderate saturation produces a moderate toxæmia, involving an eosinophilia, like nearly all moderate toxæmias. A high saturation causes a severe toxæmia, in the course of which the eosinophils disappear as in all severe toxæmia. A high prolonged saturation brings about the appearance of erythroblasts and myelocytes, indicating an hyperactivity on the part of the parent cells in the bone marrow.

4. The effect of carbon monoxide in increasing the number of erythrocytes in the blood is, in many respects, similar to that of high altitudes, in the peripheral circulation at least."—G. G. Nasmith and D. A. L. Graham, *The Journal of Physiology.—The Mining Journal* (London), March 16, 1907, p. 357. (D. M.)

Economy in Steam Production.—" An investigation of supply records will show that from half to four-fifths of the works' cost of generation is incurred in the boiler house, and this is chiefly for coal; also it is obvious that in places where electricity supply stations are already in operation, and where a fixed sum must be allocated annually for the repayment of capital charges, the only possible chance of cheapening the cost of production to any substantial extent lies in reducing the consumption of coal, and it is an index of healthy progress in economy if, while continuing to use the same plant, the annual expenditure on fuel is decreasing, while the number of units sold is increasing. The free use of steam traps is a source of heavy loss in some generating stations. Due to their liberal use, and to the scattered positions they occupy, the hot water they discharge is frequently led into the nearest drain, and runs to waste. A long experience has shown that an equally satisfactory and more economical arrangement is to connect all the water separators, valve casings, cylinder jackets, etc., to a small pipe from which steam is taken to operate a bank of boiler feed injectors; by this means all the steam required to work the injectors is derived from points where condensed water collects, and it is thus all saved, and expeditiously carried back with the steam through the injectors, and direct into the boilers. The injectors used should be of varied sizes, the smallest, when at work continuously, being equal to feeding the boilers on low loads. One steam trap of suitable size is provided at the end of the injector block, and this trap comes into action during the short and infrequent intervals when the injectors are out of action for repairs. Another advantage of this arrangement is that the large and often wasteful steam driven feed pumps can be shut down and maintained as reserve plant. An economical stand-by boiler-feed arrangement is obtained by employing three-throw pumps driven by electric motors, and fitted with variable stroke gear, which allows the use of a constant speed motor, and permits the stroke of the plunger being graduated from the maximum to zero, or vice versa whilst in action. An indicator should be fixed on each pump to show the amount of water delivered per hour.

A further easily realised and very substantial economy is to employ steam temperature water to feed the boilers, *i.e.*, to bring the boiler feed water, after it has left the economiser, or other preliminary heating device, into direct contact with live steam, so that the feed water temperature is raised before it enters the water space in the boilers to a point equal to that of steam itself. Its adoption will secure a greater output of steam from any steam boiler using ordinary hot feed water, and at an increased economy averaging at least 7 per cent., and in most cases considerably more. Effective plant for securing steam temperature feed is simple, easily applied, and neither costly nor cumbersome. Where steam temperature feed water is introduced a further step in point of output and efficiency may be expected if the boiler furnaces are enlarged, and fitted in every case with a brick arch, so that the flames do not directly impinge upon the heating surfaces; this will not only reduce the smoke, but will raise the furnace temperature, and increase the steam output of the boiler. Practical objections to such enlarged furnaces are (1) the difficulty of covering the bars with fuel; (2) the liability of the fire to burn in holes—these objections I think may be efficiently met in tubular boilers, by employing the well-known chain grate bars of extra length and width, and in Lancashire boilers the most effective remedy appears to be the removal of the furnaces from the narrow flue tubes to the front of the boiler, large furnaces being built out on the boiler front by wrought iron plates, well lined inside with firebrick, and provided with firebrick arches, and on the outside by ordinary brickwork, a cavity being formed between the outside brick lining and the metal shell of the furnace. Air for supporting combustion will be drawn through this cavity, entering by an aperture located over the furnace and round the sides thereof, thereby picking up any waste heat, and passing at a high temperature through the grate bars to the fuel. By these methods not only will a higher furnace temperature be reached, but the heat upon the boiler plates will be better distributed, and the ebullition area thereby extended. In many boiler houses forced hot air draught could be easily applied, the heat to the air being derived from the hot gases leaving the boiler for the chimney. The economy so produced is often very substantial, but the equipments at present used for this purpose are often cumbersome and very costly."—*Australian Mining Standard*, Jan. 23, 1907. (W. A. C.)

A DEVICE FOR CLEANING CONVEYOR BELTS.—"Compressed air for removing wet and sticky materials from belt conveyors has been found effective when the usual method depending upon the use of brushes failed. The device has for some time been in use at the Cananea Consolidated Copper Company's works at Cananea, Sonora, Mexico, and at the Old Dominion Copper Mining and Smelting Company's plant at Globe, Ariz. As far as known all credit for the method and its application is due to Chas. F. Shelby, superintendent of the reduction division of the Cananea Consolidated Copper Company.

The device depends upon the impact of compressed air. Air under a pressure of 90 lb. is brought to the belt at a point immediately below the return pulley. The pipe is provided with $\frac{1}{16}$ in. holes spaced by $\frac{1}{2}$ in. centres, on the side opposite the belt and extending the entire width. The force of the air escaping through the $\frac{1}{16}$ in. holes effectually clears the surface of the conveyor."—*Engineering and Mining Journal*, Feb. 9, 1907, p. 282. (W. A. C.)

RESEARCHES ON EXPLOSIVES.—In the following tables it is shown that alterations in the density of loading have a great influence on the composition of the gases resulting from an explosion, and it is quite possible that the same alterations take place when the gelatinous explosives used on these fields are detonated. In any case, it is well known that tamping has a very great influence on results obtained, and a very wide field of research seems to be opened up for investigation.

A reference to the tables will show that with the increase of density and pressure, the volume of total gases per gram decreases, but the heat developed increases. Also the calculated temperatures increase with the density, but those obtained from the pressure rise much more rapidly than the others, and although starting at a lower level they ultimately exceed the others in every explosive but one.

As a matter of fact the rise in the temperature, calculated from the pressure, amounts on the average to about 80 per cent., whereas the other calculated temperatures increase only about 19 per cent. on the average. To account for this wide difference in the calculated temperatures at low density, Noble puts forward the hypothesis that the products at these low densities are at the moment of explosion simpler than found on cooling. The carbonic acid gas, for instance, exists as carbon monoxide and oxygen, these combining as the temperature falls. He expresses the view that at high densities this dissociation would be less likely to happen. This dissociation would reduce the calories at the moment of explosion below that observed, and in this way give an erroneous value to the temperature calculated from the calories. Thus Noble proposes to account for a difference of about 70 per cent.

It is also pointed out that although modern artillery is loaded with densities up to 0·50, the powder burns really as if fired in a much lower density, approximately to densities of 0·17 to 0·23; this is because the projectile moves while the combustion is going on. Artillerists are, therefore, concerned with the characteristics found on firing in closed chambers of 0·20 density. This point was demonstrated by collecting and analysing the permanent gases found in a 9·2 in. gun after firing 103 lb. of powder which gave a pressure of 16·1 tons and 2,600 f.—s. velocity. The following comparison is instructive:—

Gaseous Components.	Gases from 9·2 in. Gun.	Gases from Closed Chamber.	
		Triangle = 0·2.	Triangle = 0·5.
	Percentage.		
CO_2	25·9	23·95	38·45
CO	36·0	36·35	23·15
H	18·7	21·50	9·90
CH_4	0·4	3·25	12·15
N	18·7	14·95	16·35

MARK I. CORDITE.

Nitroglycerine 58·0. Guncotton 37·0. Vaseline 5·0.

Density of Loading.	PER GRAMME.		Calories Water Liquid.	Specific Heat of Product	CALCULATED TEMPERATURES.		Pre'sure Tons.
	Total Gas.	Permanent Gases.			By Pre'sure	By Calories	
	c. c.	c. c.			Deg. C	Deg. C	per sq in
·05	870	670	1,272·3	·2503	2,800	4,742	3·25
·10	878·5	692·5	1,253·5	·2521	3,100	4,665	7·00
·15	880	699	1,244·5	·2531	3,415	4,620	11·70
·20	875·5	697	1,246	·2540	3,760	4,608	17·00
·25	865	688	1,252·5	·2542	4,110	4,625	22·75
·30	848	671·5	1,265	·2544	4,435	4,665	28·90
·35	832·5	658	1,282	·2546	4,723	4,720	35·05
·40	820	644·9	1,304·5	·2546	4,960	4,800	41·35
·45	809·5	634·0	1,329	·2544	5,120	4,920	47·30
·50	798·8	623·6	1,355	·2541	5,270	5,060	53·30

M.D. CORDITE.

Nitroglycerine 30·0. Guncotton 65·0. Vaseline 5·0

·05	955·4	781·8	1,035·9	·2530	2,345	3,814	3·00
·10	948	790	1,024·5	·2544	2,565	3,770	6·50
·15	931	786·5	1,023·5	·2550	2,850	3,761	10·50
·20	913·5	773	1,030	·2550	3,240	3,790	15·45
·25	893·5	754	1,044·5	·2546	3,623	3,853	20·95
·30	873	733·5	1,070	·2544	3,961	3,962	26·70
·35	852	714	1,105	·2542	4,275	4,111	32·45
·40	832	693·5	1,145	·2541	4,551	4,290	38·25
·45	810·6	673·5	1,191	·2542	4,817	4,455	43·95
·50	789·5	653·5	1.242	·2544	5,051	4,630	49·50

—ANDREW NOBLE, Proceedings of the Royal Society, June 28, 1906.—*Arms and Explosives*, Dec., 1906. (W. C.)

Reviews and New Books.

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

MINING AND ENGINEERING AND MINERS' GUIDE. By Henry A. Gordon. 615 pp. Illustrated. 6 x 9½ in.; cloth. 10s. (Wellington, N.Z., 1906: John Mackay, Government Printer. London: Eyre & Spottiswood.)

"This work, which was originally published as the 'Miners' Guide,' has proved so useful that it has been expanded into a general treatise on mining engineering, especially in connection with gold mining, lode and placer. The author was formerly inspecting engineer of the Mines Department of New Zealand, and is a past-president of the Australasian Institute of Mining Engineers. Everyone who is familiar with the excellent 'Miners' Guide' will be glad to have the present work. The fact that it is the third edition manifests the general appreciation that it has met with. Many new chapters have been added and the old ones have been considerably amplified. American engineers will find much that is of service in this work, which has achieved such great popularity in Australia and New Zealand, where it is best known."—*Engineering and Mining Journal*, Jan. 26, 1907, p. 200. (W. A. C.)

GEOLOGICAL MAP, SHOWING THE RELATIONSHIP OF THE EAST, CENTRAL, AND WEST WITWATERSRAND TO THE POTCHEFSTROOM, KLERKSDORP, AND VENTERSKROON GOLDFIELDS, SOUTH TRANSVAAL. Compiled by Theo. Kassner, from personal observation. £30. (London: George Philip & Son, Ltd.)

"This map, issued in ten large separate sheets, represents the geology of over 12,000 square miles of country. The scale of 1 in. = 5,000 ft. has allowed the author to insert much useful information not hitherto recorded. The map purports to be compiled from personal observations, but whether this refers to the whole area or only to some districts is not stated; neither is any mention made of the sources from which the mining and topographical data have been obtained.

The amount of detail represented on the map, whether mining, geological, or topographical, is stupendous. Not only are the different formations and their subdivisions indicated by colour, but the outcrop of the numerous gold reefs, bands of quartzites, and shales in the Witwatersrand formation are individually traced across country, even in districts like the West Rand The mining information includes the delineation by colour of the areas possessed by the chief mining groups, the boundaries of claims and public diggings, the position of the gold reefs at different depths. The sites, and in many cases the depths of numerous boreholes are given. Among the topographical details the insertion of farm boundaries and their beacons and of numerous surface levels will be found useful. It will, therefore, be gathered that the value of the map depends largely on the sources from which the information was obtained, and such sources should have been acknowledged.

Mr. Kassner and his publishers are to be congratulated on the successful method used to represent by pattern and colour a combination of geological, mining, and commercial intelligence, which form a blend as unusual as it is undoubtedly serviceable.

The high price of the map shows it to be chiefly intended for the wealthy. It certainly lies beyond the purchasing power of most geologists; perhaps a fortunate circumstance, for with a great deal of Mr. Kassner's stratigraphy the student of South African geology will be sorely perplexed."—*London Mining Journal*, May 11, 1907, p. 629. (A. McA. J.)

THE CORROSION AND PROTECTION OF METALS. By A. Humboldt Sexton, F.I.C., F.C.S. Price, $2·00. Manchester: The Scientific Publishing Co.

"This small book is composed of notes that 'have been written to supply to some extent the need for information' on the important subject of the corrosion and protection of metals. 'These notes do not claim to be complete, but rather to represent a preliminary study of the subject.' The book certainly does not cover all the work that has been done on this subject, but we can well believe that 'its preparation has involved no small amount of labour.'

There can be no doubt that more attention must be devoted to this important subject, and it is not surprising that the author complains of the scanty literature on the subject. In this country we may hope that some valuable data may be obtained by the committee on preservative coatings for iron and steel of the American Society for Testing Materials. This committee has appointed various sub-committees, which presented interesting reports at the meeting held at Atlantic City in June, 1905. Prof. Sexton does not expect that the opinions expressed in his book will be unanimously accepted, and probably they will not. If the book, however, stimulates an interest in the subject, it will serve a most useful end.

Some recent work done on the subject of corrosion and protection of metals is not mentioned in this book, and this we might call attention to, were it not that there is no means of ascertaining when the book was written or published. The omission of the date of publication of books of this kind is irritating."—*Electrotechnical and Metallurgical Industry*, April, 1907, p. 155. (J. A. W.)

MODERN MINING PRACTICE. Vol. I. Edited by G. M. Bailes. (J. H. Bennett & Co., Sheffield.) 6¾ by 9¾. 252 pp.

"There is no preface to inform the reader to whom this volume is primarily addressed, nor does a perusal of the first and elementary chapters make it sufficiently clear. The elementary dissertation at the outset of the book is too superficial to render it of value to the serious student, and it is doubtful if an engineer would gather much from the series of loose definitions contained in the first chapter, and the somewhat broad generalisations in the departments of physics and chemistry. With the mining engineer also this part of the book will find little acceptance, though the second and subsequent chapters which deal with practical mining questions would probably be referred to if he wanted something more than is usually to be found in the handbooks. For instance, the gases encountered in coal mines are dealt with in a chapter which treats to some extent of the chemistry and composition of the gases and the methods known to the miner of dealing with the results of explosions. The subject of ventilation has a large space devoted to it, and it is well handled. The modern colliery with the vast labyrinth of entries and workings requires ventilating appliances on a scale formerly unknown, and the subject thus assumes an importance which the author fully recognises. The descriptions of the various kinds of fans employed are made clearer by illustrations, and their capacities in cubic feet of air, with the horse-power required to drive them, are shown by the calculations. The friction of air in the mine passages absorbs a chapter, but the method of treatment leaves much to be desired, and the calculations are encumbered with awkward fractions in some places, which are quite unnecessary. The volume closes with a discussion upon safety lamps, of which there are now a great variety. There are many diagrams and illustrations throughout the work, which alone constitute a fair survey of coal-mining practice, as represented by the apparatus and appliances now in use in English mines."—*Times Engineering Supplement*, April 3, 1907. (J. A. W.)

INTRODUCTION TO PHYSICAL CHEMISTRY. By James Walker. Fourth edition. 387 pp., illustrated. Price $3·25 net. New York: The Macmillan Co.

CHEMICAL EXPERIMENTS. Prepared to accompany Remsen's "Introduction to the Study of Chemistry." By Ira Remsen. Third edition, revised by John Elliott Gilpin, 160 pp., illustrated. Price, 50 cents. New York: Henry Holt & Co.

INTRODUCTION TO METALLURGICAL CHEMISTRY. For technical students. By J. H. Stansbie. 2nd edition. 252 pp., 45 illustrations. Bound in cloth. Price, $1·25 net. New York: Longmans, Green & Co.—*Electrotechnical and Metallurgical Industry*, April, 1907, p. 155. (J. A. W.)

Selected Transvaal Patent Applications

RELATING TO CHEMISTRY, METALLURGY AND MINING.

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

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

(P.) 115/07. T. Parker. Improvements in and relating to fuel. 22.3.07.

(C.) 117/07. W. Gillett (1), D. Lehmann (2). Carburetter for internal combustion engines. 22.3.07.

(C.) 118/07. A. H. Gibson. Improvements in electro-pneumatic channelers. 22.3.07.

(C.) 119/07. A. Samuel (1), The Karsam Soap Co., Ltd. (2). Improvements in the manufacture of soaps, detergents, lubricants, batching, candlestock and the like. 22.3.07.

(P.) 120/07. L. A. Desy (1), J. W. Harris (2). Improvements in excavating apparatus. 22.3.07.

(P.) 121/07. J. E. H. Grose. Improvements in rock drilling machines. 22.3.07.

(P.) 122/07. W. Wanliss. Improvements in apparatus for use in the treatment of alluvial material containing diamonds or other precious stones. 23.3.07.

(C.) 123/07. Dr. J. Bueb. Improvements in the manufacture and production of cyanides in a condition fit for transport and storing. 23.3.07.

(P.) 124/07. J. H. Holman (1), J. M. Holman (2), Holman Bros., Ltd. (3). Improvements in rock drilling machine chucks. 25.3.07.

(P.) 125/07. C. F. Duncker (1), J. P. Cloete (2). Machine for the extraction of gold and precious stones from virgin ground. 26.3.07.

(P.) 126/07. W. E. Morton. Improvements in the means for lubricating the pins of cranks or similar moving parts of locomotive or stationary engines and the like. 26.3.07.

(P.) 127/07. R. Harris. Improvements in ore feeders. 27.3.07.

(P.) 128/07. W. M. Cochrane. Improvements in *means for driving the ore feeders of stamp mills.* 27.3.07.

(P.) 129/07. H. A. Walker. Improvements in safety brake mechanism applicable to mine skips or cages, lifts, hoists, elevators and the like. 28.3.07.

(C.) 130/07. J. M. Martin. Lubricating composition. 2.4.07.

(C.) 131/07. C. Cromwell. Improvements in and relating to fence droppers and wire retaining means. 3.4.07.

(P.) 133/07. J. McGregor. Improvements in percussive rock drilling machines. 4.4.07.

(P.) 134/07. J. Fleming. Apparatus for slimes treatment. 4.4.07.

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Sharwood, W. S., *l/o* P. O. Box 540, Johannesburg; Macdonal Street, Jeppes Extension.

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