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Proceedings

AT

**Ordinary General Meeting,
February 16, 1907.**

The Ordinary General Meeting of the Society was held in the Chamber of Mines, on Saturday, February 16th, Mr. E. H. Johnson (President), in the chair. There were also present:—

41 Members: Mr. Wager Bradford, Dr. J. Moir, Prof. J. Yates, Messrs. R. G. Bevington, W. R. Dowling, K. L. Graham, A. McA. Johnston, R. B. Pollitt, M. Torrente, A. Whitby, H. A. White, James Littlejohn, J. R. Williams, S. H. Pearce, W. A. Caldecott, A. J. R. Atkin, W. Beaver, H. D. Bell, P. Carter, Major A. B. Denne, P. L. Edwards, G. Goodwin, J. Gray, J. A. Jones, G. A. Lawson, W. D'A. Lloyd, W. P. O. Macqueen, Q. C. McMillan, G. Melvill, S. Newton, J. F. Pyles, E. T. Rand, A. Richardson, W. H. Roe, O. D. Ross, Prof. G. H. Stanley, Jas. Thomas, A. D. Viney, J. P. Ward and H. Wiley.

9 Associates and Student: Messrs. S. J. Cameron, Wm. M. Coulter, J. Cronin, J. A. P. Gibb, J. H. Herbert, C. B. Hilliard, H. C. Mannheim, 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.

The scrutineers having been appointed, after their scrutiny of the ballot papers, the President announced that all the candidates for membership had been duly elected, as follows:—

ALLEN, WILLIAM HOWARD, M.A., Bonanza, Ltd., P. O. Box 3138, Johannesburg. Cyanider.

CUNNINGHAM, WILLIAM, c/o Bechuanaland Trading Co., Salisbury, Rhodesia. Agent for Nobel's Explosive Co.

HOPTON, THOMAS HENRY WEBB, Lancaster West G. M. Co., Ltd., P. O. Box 360, Krugersdorp. Amalgamator.

MCKELLOW, WILLIAM HENRY, Robinson Deep G. M. Co., Ltd., P. O. Box 5168, Johannesburg. Cyanider.

PATTERSON, PHILIP FRANCIS, Eldorado Mine, Lomagundi, Rhodesia. Mine Surveyor.

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

CAMERON, SIDNEY JAMES, Robinson G. M. Co., Ltd., P. O. Box 1488, Johannesburg. Assistant Assayer.

FORAN, JOHN HENRY, M.M. Redjang Lebung, Benkoelen, Sumatra, Dutch East Indies. Cyanider and Assistant Mining Engineer.

MAXWELL, ROBERT WILLIAM, P. O. Box 3245, Johannesburg. Mechanical Engineer.

POPE, JOHN, Jr., Leopard and Pearl Mines, Gwelo, Rhodesia. Surveyor and Assayer.

PRESENTATION.

The President: I stated at the last meeting that it would probably be the last at which we should have the pleasure of the company of Mr. J. R. Williams. I am very glad now that I did insert that word "probably" and not make an untrue statement. I also expressed the hope that we should have an opportunity of bidding him farewell at some more formal function. I had in mind at the time the probability of giving him a banquet, but the Council after considering the matter came to the conclusion that perhaps we should live as pleasantly and perhaps longer in his memory, if we gave him something more tangible than a bad attack of indigestion. When the biography of Mr. Williams comes to be written it will inevitably be the history of the metallurgical progress of the Rand for the last twenty years. There has been no movement in the metallurgy of the Rand in which Mr. Williams has not been right in the forefront. I have had a close personal acquaintance with Mr. Williams' work for something like seventeen years, and it has been generally a case of "follow pretty quickly" if we wanted to keep up with him. My first recollection of his work was his successful treatment on a large scale of raw vanner-concentrates at the Langlaagte Estate at a time when the general belief was that we should always have to resort to chlorination for this product. A little later there was a committee appointed by the Chamber of Mines to go into this question when Mr. Williams, in the face of considerable opposition, advocated the abandonment of close concentration in favour of classification. At the Crown Reef a little later he demonstrated the great efficiency and economy of this method, which rapidly became almost universal on the Rand. Of his work in connection with the development of the decantation process which probably more than anything else has enhanced his reputation in the world of metallurgists, and has been such a fruitful source of profit to us in our operations, the transactions of this Society are such testimony, and the results enter to so large an extent into the daily work of our members, that it is unnecessary for me to enlarge upon this great

development. What is not so generally known, however, is that Mr. Williams was engaged before the war in the development of the fine-grinding of the spitzlutte product, and the late Major Seymour was designing at that time a crushing mill for the purpose. The war intervened and subsequently the necessary work of reorganisation after the war practically hung up this matter for four years, otherwise I do not think there is much doubt that the fine-grinding of banket ore would have synchronised with, if it had not actually preceded, the similar work done in Western Australia. As far as the work of this Society is concerned Mr. Williams has always been one of its most active members, both in Council and as President. He has always borne more than his fair share of the work. To Mr. Williams we also owe the inauguration of the system of prizes which has obtained during the last three years. This was entirely due to his initiative and energy. He has contributed three papers and has taken part in practically every discussion of importance. One paper dealt with copper smelting, and the two other papers were on the decantation process. His cheery personality and good-fellowship have endeared him to us all; and it will be a long time before we do not feel the blank his absence will cause. In the matter of education Mr. Williams has always been a very active force. Long before the war he was an examiner of the Cape University, and since the war he has been an examiner of the local University and one of the most active members of its Council.

Mr. Williams, on behalf of the members of this Society, I wish to tender you this little token of our great admiration and esteem, with the sincere hope that you may soon be restored to complete health and vigour, and that the little gift may perhaps be of some use to Mrs. Williams.

The President here handed Mr. Williams an exceedingly handsome solid silver tea and coffee service, the cost of which had been subscribed by the members as a token of their regard and esteem.

Mr. J. R. Williams: This is a very embarrassing position to be in. I am certain the Chemical Society has been altogether too good to me. It seems to me that it is with a very powerful microscope that you are looking at the small amount of work that I have done. I really did think that the Society at the last meeting conferred upon me what I consider to be the highest honour a member could ever possibly get, that of Honorary Life-Membership. I think it was Bulwer Lytton who was responsible for the words:—"Of all the cants of Heaven and Hell that of hypocrisy may be the worst, but that of flattery's the most

unbearable." I am sure our President has flattered me so much to-night that I feel like a seven-footer. I would like to reiterate the statement which I have made before, namely, that the development of the various processes in use on the Rand, and in other parts of the world, has not been the work of any one man. If it were not for the invaluable support received from everyone, managers, mill and cyanide managers, chemists, assayers, etc., it would be impossible to accomplish anything. They are as much responsible, if not more, for any little thing that I have done myself. Some time ago, before this Society, I ventured to make a suggestion that every man should try to *equal his possibilities*. I am pleased to say, and Mr. Pearce will bear me out, that we have seen a great improvement even during the past year. It is becoming more and more difficult to get increased extraction. When the cyanide process was first introduced it was like putting a handful of sovereigns down and picking them up again, but the time has now come when we have to pick them up one by one—we have to go to work so much more carefully. Our friend Mr. White to-night mentioned some figures about the increase of $\frac{1}{2}$ per cent. in extraction. I see that taking the tonnage crushed in the Transvaal for December, that one extra grain of gold is equivalent to £139,000 per annum, so that there is still a very big field for us. We know that in some cases the tube mill has undoubtedly improved our extraction, yet I know that Messrs. Pearce and Caldecott, who have devoted a lot of time to the tube mill, have come to the conclusion that it is still in the walking stage. We were creeping for a time, now we are walking, but we must soon be able to run. I am sure that the adoption of the tube mill has been a step in the right direction, and there are already some companies that get an extraction of 95 per cent. owing to it. When we find that 1 per cent. more means such a huge sum of money, let us endeavour to get that extra 1 per cent.

I must ask your indulgence this evening for a request I wish to make. In your journeys round the country and the mines, you very often see specimens which are usually put on the shelf and eventually thrown away. If you would send these to another institution in which I take great interest—the Transvaal University College—it would be a great help. My reason for asking you to do this is that whilst Examiner of the Cape University I noticed that the students' answers to questions on practice which they have seen at Kimberley or in the Transvaal were so much better; what they have seen, they have grasped and know. I am sure it would help if our members would send along pieces of banket carrying copper, lead, or zinc, and so on. If they

are not of any value the University would still be able to exchange them with outside institutions. I think the time has come in the Transvaal, I think I may say South Africa, when we have arrived at that stage when there is no longer any need to import men to teach us our own work here. A boy born in South Africa has as much brains as if he were born in any other part of the world. My experience of examinations at Home and here is in favour of the local boy if he is given an opportunity of learning.

To come back to your very kind presentation, all that I can say is to thank you. I am very pleased indeed that you have made the presentation in a form that will be very acceptable to Mrs. Williams. I cannot say whether I shall return here, but I can assure you that one of the chief inducements to come back would be my many friends in this Society.

LAST DRAININGS.

BY H. A. WHITE (Member).

Since the successful introduction of the Tube Mill on these fields we seem to have once more arrived at the point where our reasonable hopes, for profitably increasing the percentage extraction of the gold contents of the ore treated, are to be confined within narrow limits. It does not seem probable that such a leap as from 90 or 91 per cent. to 94 per cent. will ever be again made by the introduction of some, at present, unknown, factor into the scheme of our metallurgical operations. It is, however, some encouragement to our exertions in the direction of improvements to reflect that so little as $\frac{1}{2}$ per cent. increase would represent more than one hundred thousand pounds sterling on our annual output. I need, therefore, make no apologies for drawing your attention more closely to a consideration of some figures bearing on possibilities of increased extraction with our present treatment plants. Our Proceedings contain frequent reference to the fact that on most of our mines the accessible gold is readily dissolved in cyanide solutions of the usual composition, and that it is the removal of the dissolved gold by washing which requires most of the time given for treatment. We have also had figures of experiments (vol. V., p. 308) showing how the gold in leaching solutions rapidly rises to a maximum, and thereafter shows a steady decrease.

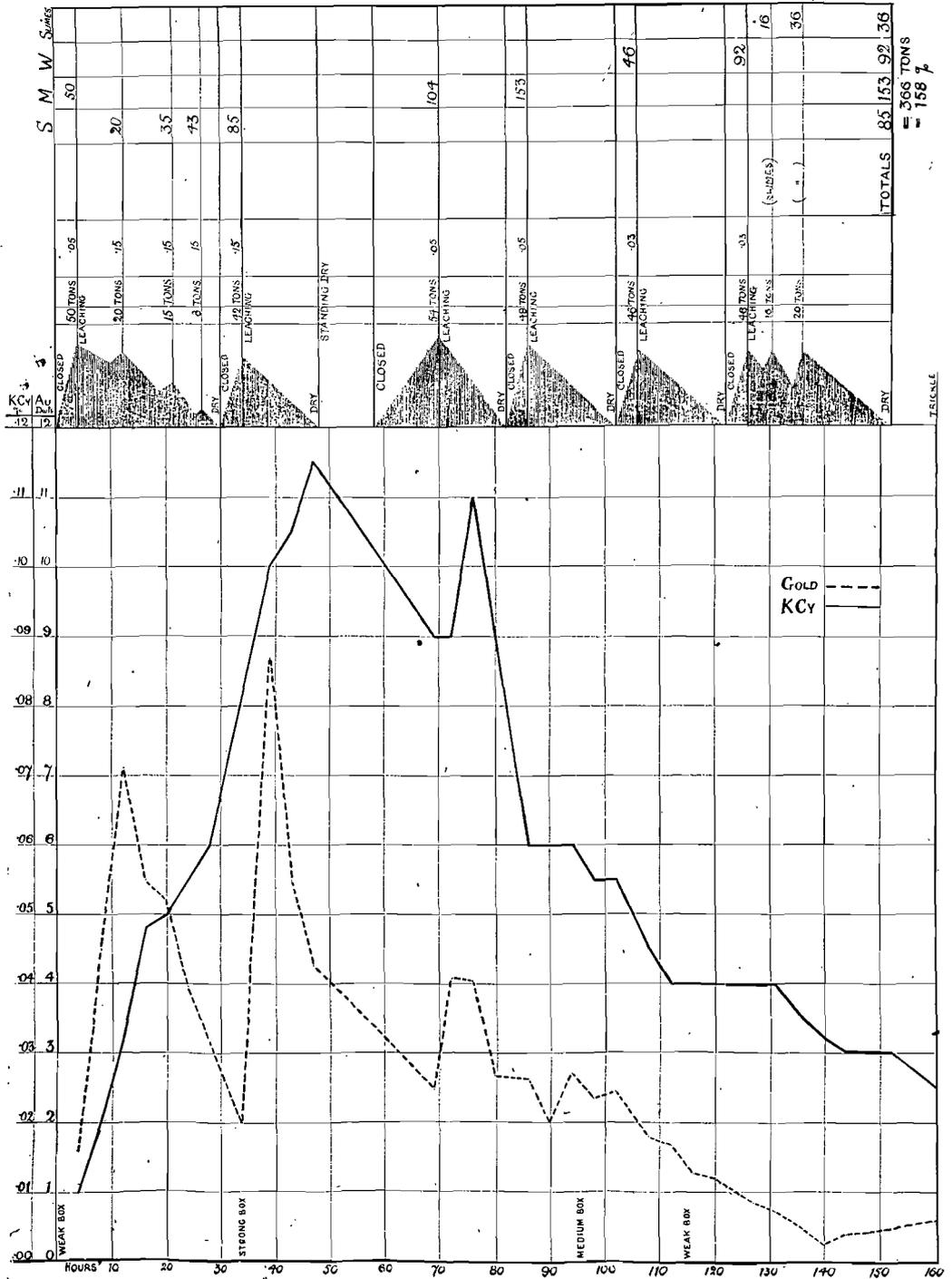
It is obvious that, if the above facts are admitted, we must take care that as little gold as possible is returned to the sands under treatment in the form of imperfectly precipitated solution.

On this point I will remark that sufficient capacity in the extractor boxes is absolutely essential and that capital and working expenditure show a heavy profit up to the point where gold in solution leaving the boxes is only present in traces. It is pertinent to call attention to the fact that the bulk of the cyanide and zinc consumption takes place in the first few compartments of the extractor box. These observations are in no way new, and I think it may be assumed that ordinary Rand practice recognises their accuracy. We then arrive at the question as to what point it is profitable to extend this washing treatment and what tonnage of solution is required to effect a given decrease in the dissolved gold contents. In the first place, it may be remarked that unprecipitated solution, if used as a wash, must inevitably increase the value of the solution remaining in contact with the sands by some definite fraction of its gold contents. It is difficult to imagine a case where such a method could be financially sound, unless the cost of precipitation were unusually high.

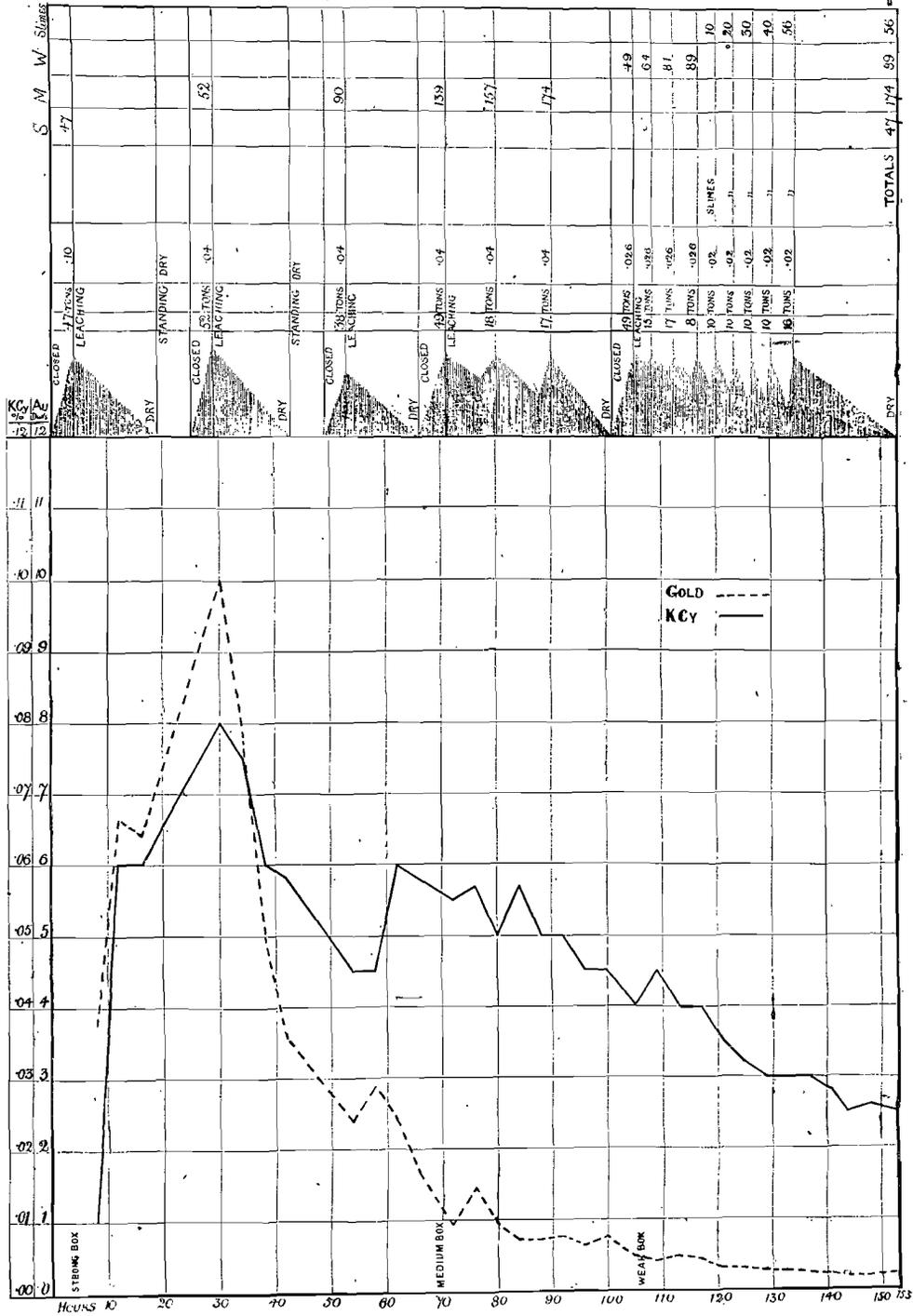
The method of applying the washes demands some consideration, and I have frequently heard the comparison made between this case and that of washing precipitates in chemical analysis. It is, of course, known that the greatest washing effect is got in the case of the precipitate by allowing each portion of wash water to drain completely and to use no more than sufficient to completely cover the precipitate. In that case it is usually important to keep down the amount of wash water, but time is not of great moment.

The case of sands under treatment is somewhat different inasmuch as time is of great practical importance, and the theoretical minimum of solution to effect a given reduction in dissolved gold values can not be reached in the period usually allotted for leaching if the above intermittent method is used. Of advantage, however, is the fact that a certain amount of displacement is effected by the method usually known as continuous leaching, and, of course, a greater rate of leaching can be maintained. To effect the most profitable compromise requires a knowledge of various factors of cost and a careful experimental determination of the values of solutions leaving the tanks under the different possible methods of treatment. I give herewith two diagrams of gold and cyanide values of solutions, leaving two fairly representative charges with same total weight of tonnage of solution (366 tons solution to 230 tons of sand). The method adopted on charge 439 was based upon observation of results from 429, the endeavour being made to reduce (1) cyanide consumption; (2) to keep the gold values in correspondence with the cyanide

**CHARGE 429. 230 TONS SAND , 5.2 DWTS ,
RESIDUES 10DWT , 366 TONS SOLUTION , TREATMENT 162 HOURS,
AT 152 HOURS = LAST DRAININGS 0.48DWT , AT 160 HOURS = TRICKLE 0.6 DWT .**



CHARGE 439. 230 TONS SAND, 4.5 DWTS,
RESIDUES 0.9 DWT, 366 TONS SOLUTION, TREATMENT 159 HOURS,
AT 153 HOURS=LAST DRAININGS 0.2 DWT, AT 158 HOURS = TRICKLE 0.28 DWT .



percentage so that all the rich solution might be leached into the "strong" boxes without the necessity of putting too weak (in cyanide) solutions through these, or too strong through the weaker boxes. It will be observed that no facts are brought to light which are new to students of the Proceedings of this society. I need only refer members to the mass of experimental material scattered through our pages, especially under the names of A. F. Crosse and W. A. Caldecott. These known facts are, however, nowhere brought together and made to bear upon our every-day methods of treatment, and I shall be excused for briefly stating the salient points. In the first place, the diagram referring to charge 429 shows that the practice of commencing with a weak or medium solution is not (except in special cases not now under consideration) to be recommended. The first leachings on this plan begin to carry gold very soon, and the cyanide strength lags behind too far. These leachings would have to go into weak boxes, and I think they largely account for the formation there of the objectionable "white precipitate." In passing, I may remark that in spite of all the literature on this subject, we have as yet had no practical advice upon eliminating it from our zinc boxes.

In my view its precipitation depends upon no definite factors such as alkalinity or cyanide strength as far as actual practice is concerned, but solely upon the rate of change of these factors. A certain amount is held in solution, as Dr. Moir has so ably shown, definitely corresponding to alkalinity, or in the form of double cyanide of zinc and potassium dependent upon cyanide strength. Temperature is also an important factor. Any reduction in strength in these constituents will cause zinc compounds to be precipitated and our aim must be to effect this in the tanks and not in the boxes. If these are true statements of the case it is clear that first drainings from the sand tanks, which carry practically no cyanide or alkalinity, must not go to a weak box where they mix with solutions nearly saturated with zinc, and so reduce their protective strength to the point where zinc hydrate falls out.

They must go to either strong or medium boxes where their lack of strength is counterbalanced by solution flowing from other tanks under treatment, and the danger point is not reached. To effect this with minimum loss the new tank must be started by a saturation with strong solution, and thereby a gain in time is also shown, as a comparison of the diagrams will confirm.

An inspection of these diagrams will also show that the solution of the gold is done almost entirely by the aid of interstitial oxygen, especially during the "standing dry," which is thus shown

to be the effective part of the treatment as far as this part of the process is concerned.

Tank 429 shows that the effect of pumping on the first strong was to stop solution, and commence washing out, of the gold.

These details confirm Mr. Crosse's observation upon the lack of oxygen in working solutions (zinc process) and also explain why in practice this proves no obstacle to solution of the gold. In an ordinary tank of sands "standing dry" about 49 per cent. of the volume of the portion of the tank in use would be occupied by sand, 13 per cent. by solution and 38 per cent. by air. If calculated to show average thickness of the film of solution it appears that this is greater than that due to surface tension (about $\frac{1}{1000}$ mm.) and must be partly due to capillarity between opposing surfaces of quartz. In any case, the thickness of solution through which diffusion of oxygen has to take place is so small as to require a very short time for the purpose. The diagrams indicate clearly that .10 per cent., or even .05 per cent. KCy is sufficient strength for rapid solution of the gold under properly favourable conditions, and .10 per cent. is chosen to provide sufficient strength for efficient precipitation of the following solutions, which should not get much below .02 per cent. KCy. The proportion of strong solution evidently need not exceed 20 per cent. It will probably be a relief to members if I mention that the problem presents so many uncertain factors, such as absorption effects, displacement, capillary and viscosity phenomena that a mathematical treatment of the question is, I find, much better omitted.

The conclusions arrived at are, therefore, based on some actual figures which I have had the opportunity of examining. I direct attention to the value of the solution leaving the tank towards the end of the treatment and to the effect of increasing the relative tonnage of solution used when other conditions remain almost the same. These "last drainings" vary slightly in accordance with the time at which they are taken, but, if taken in accordance with a definite plan, can be made the basis of reliable calculations, though the effect of small differences is not shown up in assay of the sand residues except on long period averages. On a certain mine, during the year 1905, the amount of solution used for treatment of the sands averaged 141 tons for 100 tons of sand. For the year 1906 it was possible to increase this amount to 161 tons, and during the last month of the year as much as 166 tons of solution were used per 100 tons of sand.

The following results were obtained:—

1905. Charges ...	4·279 dwt.
Residues ...	0·851 „

	Last drainings	0.710 dwt.	
	Zinc used ..	0.53 lb. per ton	
	Cyanide used	0.49 "	
	Solution ...	141 per cent.	
1906.	Charges ...	3.980 dwt.	
	Residues ...	0.745 "	
	Last drainings	0.390 "	
	Zinc used ...	0.51 lb. per ton	
	Cyanide used	0.55 "	
	Solution ...	156 per cent.	
December, 1906—			
	Solution, 166 per cent.	Last drainings, 0.21 dwt.	

The samples of solution taken to represent the last drainings in these cases showed a reasonable amount of regularity, and a comparison was made between the gold values of these and the value of solution remaining in contact with the residue, as ascertained by washing a weighed quantity of residue and well stirring as quickly as possible with a known quantity of recently boiled water; the solution was then filtered and assayed. On calculating the results it was found that approximately one-ninth of the value of the last drainings gave the value of dissolved gold per ton of dry residue. This is the average of twelve sets of experiments and the individual variations were not of importance.

If charges in 1905 had been the same as 1906, residues for that year (for the same extraction) would have been 0.792 dwt.

The difference between the residues for the two years would then amount to 0.47 dwt. of which 0.35 dwt. would be directly due to difference in value of the dissolved gold, as shown in the average of last drainings. I think this sufficiently demonstrates the fact that the values shown by last draining samples are not illusory and may be relied on in commercial calculations regarding a possible extension of treatment.

The figures also show that increased tonnage of solution may be to a certain extent depended upon to lower the amount of actually dissolved gold enriching our dumps.

It is by no means a simple matter to calculate the cost of handling the extra tonnage of solution as many factors are indeterminate. For example, it is easy to ascertain the consumption of zinc and cyanide per ton of solution, but an additional few tons would probably cost much less than the amounts so estimated. It will be, however, on the safe side if any such calculation is based on present average cost, and for a particular case I will assume that 6d. per ton of solution is quite sufficient to cover cost of pumping, zinc and cyanide consumption and other smaller items.

In this particular case the figures already quoted show as follows:—1.41 tons solution

per ton of sands give last drainings value 0.71 dwt., equal to 0.08 dwt., or 4.00d. per ton of sand. 1.56 tons similarly shows $\frac{0.39}{9} \times 50 =$

2.16d.; 1.66 tons also shows $\frac{0.21}{9} \times 50 = 1.16d.$;

1.76 tons would probably show $\frac{0.06}{9} \times 50 = 0.33d.$

The costs for increased tonnage would be:—

1.41 tons cost 8.46d.

1.56 " " 9.36d., increase 0.90d.

1.66 " " 9.96d., " 0.60d.

1.76 " " 10.56d., " 0.60d.

For the increased cost 0.90d. a profit of (4.00 - 2.16 - 0.90)d. = 0.94d. per ton of sand is shown; for the second increase a profit of 0.40d. per ton of sand is shown; for the last increase a profit of 0.23d. per ton of sand is shown.

On 100,000 tons of sand treated the profits for each extension of tonnage of solution would be £391, £166 and £95 respectively.

These figures indicate how far it may pay to carry the reduction in value of dissolved gold thrown away, but do not pretend to greater accuracy than is necessary to throw some light on the question. There are certainly still many plants working on the Rand where capacity is somewhat limited, and in any case it is necessary to utilise to the utmost extent all available facilities. In each case a diagram of gold and cyanide values in solutions leaving the tanks, taken at regular and frequent intervals, will throw a light on any deficiency in treatment and will show if it be possible to profitably secure any small increase of extraction in the direction I have indicated.

In writing this paper I am aware of the fact that novelty is not easily attainable on these questions, but if those who have been aware of these facts for years will contribute their experience, and those who have not made use of these methods will bring forward their objections, we should all benefit by a lively discussion. It would also be a convenient opportunity for members to express their opinion on our present methods of recording strengths of solution in terms of KCy for cyanide and NaHO for alkalinity. Considering the present position of the ionic theory I would suggest that all such statistics of analysis should be registered in terms of cyanidion and hydroxidion so as to avoid the difficulties caused by using sodium cyanide and calcium hydrate and keeping figures which refer to other compounds.

The President: I am sure you will all agree with me that Mr. White deserves our hearty thanks for this extremely interesting paper on a matter which we are apt to consider common

knowledge. He has practically photographed a mass of data which has been accumulating for years into one succinct paper, and I am sure it will greatly repay the study of those who are now on cyanide works. The point of the value of dissolved gold remaining is one that will crop up rather forcibly in connection with tube milling. As far as percolation at least is concerned the point will arrive when the increased gold dissolved will be balanced by the amount of gold remaining as dissolved gold after treatment. I am sure Mr. White deserves a hearty vote of thanks.

Mr. J. F. Pyles : I was very much interested in Mr. White's paper, and I trust it will provoke the very lively discussion it deserves. It is a subject which interests me much, as it is only when "last drainings" are low, that one can feel that nearly all the accessible gold has been obtained. The value of last drainings is often largely dependent upon the amount of hard lumps in the charges treated. When the lumps are broken before a charge receives its solution it will be found that with an average amount of solution, say, 1.7 of solution to 1 of sands, last drainings will be very low. I have collected some data on these lines which I shall submit later in the discussion.

Mr. G. Melvill : I would like to ask Mr. White at what stage he samples his last drainings. Is the sample taken from the last drippings or when the solution slows down?

Mr. H. A. White : As I pointed out, it makes some little difference when the last drainings are taken. We take ours in a perfectly definite manner, that is twenty hours after the solution is pumped on and we find that we get a certain proportion between the value of that and the amount of dissolved gold in the residue.

Mr. G. Melvill : I think that is a fair way to take it. If you sample what are called the last drippings you will find it will be high.

NOTES ON THE OCCURRENCE AND TREATMENT OF AN AURIFEROUS ORE CONTAINING INSOLUBLE ARSENIDES.

By J. K. WILSON, A.I.M.M., M.Aust.I.M.E.
(Member).

In bringing these notes before the Society the writer is aware of their incomplete and somewhat fragmentary nature. He can only plead for the indulgence of members on the grounds that there is exceedingly little literature available on the treatment of ores of this class, and that on small properties in outlying country districts a metallurgist has seldom the means or

opportunity of investigating as their interest and importance often merit the various phenomena brought to his notice.

So that while it is more with the object of obtaining than imparting information that this paper is written, it is, nevertheless, hoped that it may prove of interest and perhaps of use to some members who may be called upon to deal with an ore of a similar character.

The quartz veins at the Nooitgedacht Mine, Lydenburg District, are laid down horizontally in the well-known shale formation of the Pretoria Series. The veins following the dip of the country, which in this district is about 5 deg. north.

The vein matter is exceedingly variable in size, nature and gold contents, and bears evidence of having been subjected to intense heat and pressure subsequent to its formation—probably owing to intrusions of igneous rocks. In many places, for instance, the shale is baked hard, and in mining operations it is no uncommon occurrence to break into cavities in the quartz filled with crystals of pure sulphur, pointing rather to desulphurisation by thermal or volcanic action than by atmospheric influence. Subsequent weathering has naturally taken place, however, and has played an important part in oxidation.

In portions of the mine the veins are still heavily mineralised—arsenical pyrites predominating—but the gold values are usually associated with oxidised rubby quartz and ferric oxide, so that, as a rule, the ore milled is remarkably free from metallic sulphides. The bulk of the gold is free and fairly coarse, and is amenable to amalgamation.

Formerly frue vanners were used for concentration, but the sulphides so saved were apparently of too low value and in too small quantity to pay for their separation and treatment, so that vanning has long since been abandoned. Both accumulated and current tailings are treated by the cyanide process, and as it is here that the ore exhibits its characteristics to best advantage, its action on cyanide and alkaline solutions must be considered at some length.

Several analyses of the tailings have been made by different chemists and show the following composition :—

	...	%	to	%
Silica (SiO ₂)	...	36	to	40
Iron (Fe ₂ O ₃)	...	40	to	43
„ (Fe ₃ O ₄)	...	nil	to	2
Alumina (Al ₂ O ₃)	...	11.8	to	12.5
Arsenic (as arsenide)	...	2.05	to	4.7
Sulphur (S)2	to	2.0
Antimony (Sb)	...	nil	to	heavy traces
Copper (Cu)	...	nil	to	traces
Acid salts	...	nil	to	.087 (H ₂ SO ₄).

Although the amount of free acid or acid salts present is a negligible quantity, even in those tailings which have been exposed to weather for some years, the consumption of lime is heavy, and this led to a few experiments which proved interesting, and some of which will be described.

ACTION OF ORE ON SOLUTIONS OF ALKALI EXPERIMENT I., SAMPLE I.

Estimation of lime consumption on tailings which had been exposed to the atmosphere for some years, and which contained acid salts equal to 0.87 per cent. H_2SO_4 .

200 gm. tailings agitated with 1,000 c.c. lime water containing 1.08 per cent. alkali (in terms of caustic soda).

After 1 hour's agitation, alkali = 0.48 per cent. = 6.0 lb. consumed per ton of ore.

After 14 hours' agitation, alkali = 0.28 per cent. = 8.0 lb. consumed per ton of ore.

After 24 hours' agitation, alkali = 0.16 per cent. = 9.2 lb. consumed per ton of ore.

After 36 hours' agitation, alkali = 0.11 per cent. = 9.7 lb. consumed per ton of ore.

After 46 hours' agitation, alkali = 0.08 per cent. = 10.0 lb. consumed per ton of ore.

After 120 hours' agitation, alkali = 0.04 per cent. = 10.4 lb. consumed per ton of ore.

EXPERIMENT II., SAMPLE II.

Estimation of lime consumption on current tailings containing traces of soluble acid salts.

A similar experiment conducted on current tailings showed a consumption of 9.8 lb. alkali per ton in 120 hours.

It will be observed that it takes a considerable time for this ore to act upon the lime and decompose it; although after the first 24 hours the decomposition is comparatively slight.

Allowing for the small amount of acid present in the weathered tailings, there is practically no difference in the amount of lime consumed by current and accumulated tailings. This is not surprising, and although the difficulty of oxidising arsenides is too well known to be discussed here, the following experiments may be of interest as illustrating the point.

EXPERIMENT III., SAMPLE III.

200 gm. tailings (raw) agitated with 1,000 c.c. NaHO solution containing 2.24 per cent. alkali.

After 1 hour's agitation, alkali = 1.28 per cent. = 9.6 lb. consumed per ton of ore.

After 20 hours' agitation, alkali = 0.96 per cent. = 12.8 lb. consumed per ton of ore.

After 168 hours' agitation, alkali = 0.48 per cent. = 17.6 lb. consumed per ton of ore.

EXPERIMENT IV., SAMPLE III.

Another portion of sample was slowly raised to roasting heat in muffle furnace and maintained at this temperature for two hours with constant stirring. It was then agitated with NaHO solution as in Experiment III.

After 1 hour's agitation, alkali = 1.336 per cent. = 9.04 lb. consumed per ton of ore.

After 20 hours' agitation, alkali = 1.04 per cent. = 12.08 lb. consumed per ton of ore.

After 168 hours' agitation, alkali = 0.68 per cent. = 15.60 lb. consumed per ton of ore.

A portion of the roasted sample digested with water showed traces of neutral sulphates only, and no soluble arsenic. The supernatant alkali liquor contained a considerable amount of arsenic and traces of antimony and sulphates.

EXPERIMENT V., SAMPLE III.

A third portion of the sample was mixed with 5 per cent. of its weight of sulphur, and roasted for two hours in muffle furnace, attention being paid to the usual precautions observed in roasting arsenical ores.

200 gm. of the roasted tailings were treated with 1,000 c.c. NaHO solution, containing 2.32 per cent. alkali, as in Experiment III.

After 2 hours' agitation, alkali = 1.96 per cent. = 3.6 lb. consumed per ton of ore.

After 20 hours' agitation, alkali = 1.88 per cent. = 4.4 lb. consumed per ton of ore.

After 168 hours' agitation, alkali = 1.84 per cent. = 4.8 lb. consumed per ton of ore.

A portion of the roasted tailings on being digested with water showed traces of neutral sulphates only.

EXPERIMENT VI., SAMPLE I.

Estimation of alkali consumed by tailings, using lime water of various strengths.

Portions of Sample I. were agitated with lime water of the following strengths:

1. 200 gm. tailings with 1,000 c.c. lime water containing 0.088 per cent. alkali.

2. 200 gm. tailings with 1,000 c.c. lime water containing 0.176 per cent. alkali.

3. 200 gm. tailings with 1,000 c.c. lime water containing 0.416 per cent. alkali.

4. 200 gm. tailings with 1,000 c.c. lime water containing 0.600 per cent. alkali.

5. 200 gm. tailings with 1,000 c.c. lime water containing 1.400 per cent. alkali.

After 1 Hour's Agitation.

1. Tested nil per cent. alkali = 0.88 lb. consumed per ton of ore.

2. Tested 0.012 per cent. alkali = 1.64 lb. consumed per ton of ore.

3. Tested 0.040 per cent. alkali = 3.76 lb. consumed per ton of ore.

4. Tested .0124 per cent. alkali = 4.76 lb. consumed per ton of ore.

5. Tested .0568 per cent. alkali = 8.32 lb. consumed per ton of ore.

After 17 Hours' Agitation.

1. Tested nil per cent. alkali = .88 lb. consumed per ton of ore.

2. Tested nil per cent. alkali = 1.76 lb. consumed per ton of ore.

3. Tested .0004 per cent. alkali = 4.12 lb. consumed per ton of ore.

4. Tested .0028 per cent. alkali = 5.72 lb. consumed per ton of ore.

5. Tested .0312 per cent. alkali = 10.88 lb. consumed per ton of ore.

From the foregoing experiments it is evident that the heavy consumption of alkali by this ore is due chiefly to the presence of arsenic, which being in the condition of arsenide (probably of iron), and insoluble in water, acts slowly upon caustic alkalies decomposing them, and forming with lime compounds insoluble in water and with soda, of course, soluble compounds.

It is remarkable that with the weaker solutions of alkali, decomposition is less rapid and complete than with the stronger, and, indeed, when the percentage falls below a certain point, there is very little decomposition at all.

ACTION OF ORE ON CYANIDE SOLUTIONS.

These arsenides also decompose cyanide solution and are soluble in it, thus:—

EXPERIMENT VII., SAMPLE I.

Samples were agitated with cyanide solutions of strengths as under:—

	KCy %	Alkali %
1. 200 gm. tailings with 1,000 c.c. solution containing025	.028
2. 200 gm. tailings with 1,000 c.c. solution containing073	.028
3. 200 gm. tailings with 1,000 c.c. solution containing125	.028
4. 200 gm. tailings with 1,000 c.c. solution containing200	.028

After 20 Hours' Agitation.

1. Tested .024 per cent. KCy, alkali nil = 10 lb. KCy and 2.8 lb. alkali consumed per ton.

2. Tested .060 per cent. KCy, alkali nil = 1.30 lb. KCy and 2.8 lb. alkali consumed per ton.

3. Tested .091 per cent. KCy, alkali nil = 3.40 lb. KCy and 2.8 lb. alkali consumed per ton.

4. Tested .160 per cent. KCy, alkali nil = 4.00 lb. KCy and 2.8 lb. alkali consumed per ton.

After 40 Hours' Agitation.

1. Tested .022 per cent. KCy, alkali nil = .3 lb. KCy and 2.8 lb. alkali consumed per ton.

2. Tested .053 per cent. KCy, alkali nil = 2.00 lb. KCy and 2.8 lb. alkali consumed per ton.

3. Tested .088 per cent. KCy, alkali nil = 3.70 lb. KCy and 2.8 lb. alkali consumed per ton.

4. Tested .145 per cent. KCy, alkali nil = 5.50 lb. KCy and 2.8 lb. alkali consumed per ton.

An analysis of this solution showed that it contained arsenic.

It will be observed that with an ore of this class the task of maintaining the working solutions in a constant state of alkalinity is somewhat difficult and expensive, and although it is difficult to exaggerate the importance of so doing, still in inaugurating the process here the solutions were at times neutral, *i.e.*, contained no "protective alkali." A study of the condition of such solution is instructive.

It was noticed that when lime—or any caustic alkali—was added to the neutral solution a precipitate was formed varying in colour from white to dark brown, that the cyanide test rose and that the solvent power of the solution for gold was increased.

A sample of this precipitate was forwarded to Mr. A. Heymann for qualitative analysis and gave the following results:—

Arsenic	... traces.
Iron small quantity.
Zinc...	... large "
Calcium	... fair "

(the metals being present as oxides).

It is interesting to note that Dr. Bay* obtained a very similar precipitate to this by simply allowing working solution to remain standing; exposed to air, but in this instance it appears to be due more to the causes described by W. J. Sharwood† who found that "when the zinc cyanide corresponds to or is in excess of the proportion of one molecule to one of hydroxide, it is dissolved in that proportion, but one-half of the dissolved zinc separates out as an oxide."

That arsenic is precipitated from cyanide solution by lime there can be no doubt. One has only to dissolve arsenic or arsenious oxide in cyanide solution and add excess of clear lime water to find that a dense white flocculent precipitate is formed with rise in cyanide test.

The precipitation of arsenic is incomplete, however, and judging by the following experiment it would appear that only half the metal is precipitated in this case also.

* See December Journal, 1904, p. 149.

† See July Journal, 1904, p. 26.

EXPERIMENT VIII.

·5 gm. arsenious oxide was dissolved in 200 c.c. KCy solution containing ·55 per cent. KCy. The solution then tested ·46

To 50 c.c. of this solution 50 c.c. lime water, containing ·168 per cent. alkali, was added; after filtering the solution tested ·25 per cent. KCy and ·020 per cent. alkali.

Increase of cyanide on original solution equals ·04 per cent. which is as near as possible one-half the difference.

The precipitate slowly re-dissolves in excess of neutral or slightly alkaline cyanide solution. The precipitation of arsenic by the addition of ammonia or caustic soda to the neutral working solutions here would appear to be due to the fact that the solutions always contain a certain amount of CaCO₃ which may be hydrated by the addition of these reagents and so form the precipitate.

It is hardly necessary to detail the particulars of numerous gold leaf experiments which were carried out with a view to ascertaining the relative solvent power of neutral and alkaline working solution and freshly made-up solution, and also the effect of artificial oxidisers on these solutions. Suffice it to say that the addition of potassium permanganate or ferricyanide to a neutral solution had no appreciable effect in increasing the rate of solution of gold, whereas the addition of lime rendered the solution almost as energetic in its action as freshly made up cyanide solution of equal strength. The experiments also tended to show that this increased activity is due primarily to the elimination of the metals zinc, arsenic, etc., from solution, and secondly to the fact that a certain amount of cyanide is regenerated and rendered available as a solvent.

To recapitulate—

1st. We find that the consumption of lime by this ore is due principally to the presence of arsenic which exists in the form of arsenides insoluble in water.

2nd. That these arsenides act slowly upon alkalis decomposing them, but that the stronger the alkaline solution the more rapid and complete is the reaction which takes place.

3rd. That cyanide solution also dissolves these arsenides and is decomposed by them.

4th. That arsenic may be partially precipitated from cyanide solution by the addition of Ca(OH)₂ in excess, some of the cyanide being regenerated, and that such precipitate is more or less soluble in excess of neutral or slightly alkaline cyanide solution.

5th. That the presence of zinc, arsenic or other base metals in cyanide solution retards its solvent power for gold.

Bearing these facts in mind it seemed feasible, provided the compounds formed with lime and arsenic were less soluble in cyanide than the arsenic compounds already existing in the ore, that by applying the lime to the ore first and allowing a complete reaction to take place before adding the cyanide, the latter would be less liable to be attacked than would be the case if the two reagents (lime and cyanide) were added simultaneously to the ore, and that the tendency would be for the arsenic and other deleterious substances to be gradually eliminated from solution. With a view of determining this point the following experiment was carried out:—

EXPERIMENT IX., SAMPLE IV.

A. 200 gm. tailings were agitated for 16 hours. with 1,000 c.c. lime water containing ·156 per cent. alkali. 1,000 c.c. cyanide solution containing ·116 per cent. KCy and ·008 per cent. alkali were then added and the whole agitated for 20 hours. The solution was tested at intervals of 2, 6 and 20 hours giving the following results:—

After 2 hours.

KCy = ·056%, } KCy consumed per ton = ·40 lb.
alkali = ·038% } alkali " " = 8·80 "

After 6 hours.

KCy = ·056%, } KCy consumed per ton = ·40 lb.
alkali = ·032% } alkali " " = 10·00 "

After 20 hours.

KCy = ·056%, } KCy consumed per ton = ·40 lb.
alkali = ·024% } alkali " " = 11·60 "

B. 200 gm. tailings agitated with 2,000 c.c. solution containing ·058 per cent. KCy and ·082 per cent. alkali (lime) agitated for 20 hours, the solution being tested at intervals, as under:—

After 2 hours.

KCy = ·054%, } KCy consumed per ton = ·80 lb.
alkali = ·052% } alkali " " = 6·00 "

After 6 hours.

KCy = ·053%, } KCy consumed per ton = 1·00 lb.
alkali = ·036% } alkali " " = 9·20 "

After 20 hours.

KCy = ·053%, } KCy consumed per ton = 1·00 lb.
alkali = ·026% } alkali " " = 11·20 "

The conditions under which the experiments A and B were conducted were exactly similar. It will be seen that in B, where the lime and cyanide were added simultaneously to the ore, the consumption of cyanide is greater, as both lime and cyanide have an equal opportunity of being attacked by it.

The ore has been treated on these lines for the last five months, and greatly improved results have been obtained, as will be shown immediately. In fact, in practice this has proved the key to the really successful treatment of this ore.

METHOD OF TREATMENT AND RESULTS OBTAINED.

The method of treatment is simple and effective. In the case of accumulated tailings the requisite amount of lime is fed into each truck as the vat is charged. The charge is then saturated with water and allowed to soak for about 24 hours. The water is then drawn off and cyanide solution from .04 per cent. to .06 per cent. KCy is run on. This strength of solution is maintained throughout the treatment, and after each wash the charge is leached dry.

The amount of solution necessary is about $3\frac{1}{2}$ to 1 of ore, and the time of treatment ranges from 8 to 12 days.

In the case of current tailings the necessary lime is added in the mill.

Formerly the cyanide solution was run on to the tailings direct without any preliminary soaking with lime. The moment the treatment was altered the cyanide consumption dropped, the extraction increased and the time necessary for treatment was gradually reduced. Moreover under the old system the solutions became "fouled" and had occasionally to be run to waste.

RESULTS OF TREATMENT UNDER OLD SYSTEM.

Amount of tailings treated	... 1,832 tons.
Assay value before treatment	... 3·977 dwt. per ton.
" " after	... 1·461 " "
Total gold contents of tailings	... 364·29 oz. "
" " residue	... 133·83 " "
Theoretical recovery	... 230·46 " "
" " per cent.	... 63·26 per cent.
Fine gold actually recovered	... 194·68 oz.
Estimated gold remain'g in zinc	25·00 "
Actual recovery per cent.	... 53·44 per cent.
Cyanidé consumed per ton	... 1·95 lb.
Lime	... 13·50 "
Average time of treatment	... 12 days.

RESULTS OF TREATMENT UNDER PRESENT SYSTEM.

Amount of tailings treated	... 1,853 tons.
Assay value before treatment	... 4·440 dwt. per ton.
" " after	... 1·143 " "
Total gold contents of tailings	... 411·36 oz. "
" " residues	... 105·99 " "
Theoretical recovery	... 305·37 " "
" " per cent.	... 74·23 per cent.
Fine gold actually recovered	... 300·61 oz.
Actual recovery per cent.	... 73·03 per cent.
Cyanide consumed per ton	... 1·45 lb.
Lime	... 14·00 "
Average time of treatment	... 9 days.

Considering that the accumulated tailings contain a large amount of slime, these results can only be regarded as satisfactory.

The President: It is very interesting, I think, to get a paper on ore other than local. There is a great difference between this ore and the ore we are accustomed to. We have some 96 per cent. of silica here, but in this there is only about 36 per cent., which means a considerable complication in the treatment. It is evident that Mr. Wilson has done a very large amount of work in trying to arrive at a satisfactory solution in dealing with this ore, and he deserves our thanks for placing the matter before us. I hope it will lead to a discussion, for whereas the difficulties of dealing with cupriferous ores have been previously discussed, the influence of arsenides is somewhat new to us.

WITWATERSRAND MINE AIR: RECENT INVESTIGATIONS.

(Read at September Meeting, 1906.)

By JAMES MOIR, D.Sc., M.A. (Member).

REPLY TO DISCUSSION.

Dr. J. Moir: In commencing to reply to this most important discussion, I feel that there has been a lack of proportion about it—that the matters which have been most severely criticised are the unessential ones. The only point that has really been made against me is that we have not accumulated enough evidence to prove my statements *up to the hilt*. This I admit, and at the end of this paper I am giving details of some 430 new tests, which, I hope, will make the position clearer.

Nevertheless, the unanimity and the strength of opinion expressed by my critics has been such, that I have to take up the challenge on this question of lack of evidence, and endeavour, even at some length, to rebut their arguments—the more so that so many people go on the assumption that "what everybody says must be true."

Now, this is not the first time in this country that I have had to defend what looks like a beaten cause,* and indeed, to have to defend it, like Cato, with the gods against me. Mr. Bradford will know who the gods are in this case, if fancy. I venture to think all the same, despite Dr. Irvine's pastoral metaphor, that I won the first combat. Certainly before 1903 nobody in the Transvaal knew—or cared, if they did know—that explosives could give off CO and NO₂ in ordinary blasting, or that compressors could be over-driven so as to poison the air which they deliver, or that mine air might continue to be vitiated by CO for hours after blasting, or that Rand miners' phthisis could involve anything more than silicosis.

* *Victrix causa deis placuit, sed victa Catoni.*

Nowadays we have advanced so far as to be thinking of preventing the misbehaviour of explosives, and of prohibiting the use of compressor oils of low flash-point. In the same way the proposed 15 per cent. CO₂ standard is the logical remedy for the other defects above mentioned.

I cannot resist saying that the attitude of some of my critics seems to be that they will be satisfied with the evidence regarding lack of ventilation somewhere about the time when all the mines are worked out. Remember, this evidence has been growing for four years already.

The attitude, for example, of Drs. Irvine and Macaulay towards experimental investigation is now just the same as their attitude in 1903, namely, obscurantism. Put bluntly, it is that not even hundreds of analyses can establish any scientific opinion. I have already admitted, of course, that their attitude was quite different in 1904-5, but that period now appears to have been only a lucid interval. I think they do not quite realise how long it takes to accumulate these *data*. Were 50 samples to be taken in each of 70 mines, the work would take eight or ten years with the present resources of the Mines Department, by which time quite a number of workers will have been either incapacitated or altogether finished off. Personally, I feel that the question is rather more urgent than that.

I fancy that if we get even 109 bad places rectified we shall, in passing, rectify hundreds of other bad ones at present unknown.

It would have been hopelessly invidious for us to select a number of mines and concentrate our efforts on them: it would be too unfair to some of them to be made to incur expenses which other mines, equally guilty but undetected, were escaping. This is our reason for attempting to test all the mines at once, even if it has been done quite inadequately. Later on it will be worth while, however, to give more attention to those mines which have shown only about 30 per cent. of bad samples in bad places.

I have to admit having done Mines "r" and "s" an injustice in calling them "disgraceful all over"; the new results show that they are not quite so bad as I thought. There are a number of smaller injustices in detail which I have committed, but I certainly do not see why I am not entitled to condemn as disgraceful all over a mine which has its main connections vitiated, because, really, when main levels are bad, it is a waste of time to test the stopes which are fed from these levels; they are bound to be bad.

It must be confessed that my adjectives were rather hastily chosen, yet I still think I erred on the safe side, and that "scandalous unhealthiness" was nearer the truth than, for example, the state-

ment that such results could have been obtained in any metalliferous mine in the world. That was a catching phrase, but all the evidence is against it. Where else in the world does one find the use of 1.5 lb. of explosive per ton broken, or stopes 24 in. wide, or 25 cub. ft. of air per head? The wonder is that there are so many as 20 mines on the Rand whose ventilation (as I have already said) could hardly be improved on. I do think that even my violent adjectives were justifiable so far as our present knowledge goes.

I now pass on to the details of the criticism, and I shall first deal with Mr. W. E. C. Mitchell. He wants to know if I am conversant with the working conditions underground? Well, I can only claim to have been underground some 20 or 30 times in my life, and I only went once in connection with the present investigation, which was to Mr. Mitchell's mine, by the way. Practical mining is, no doubt, an art in which I should be hopelessly incompetent, but theoretical mining as a science is certainly elementary compared with heterocyclic chemistry, for example.

By the way, Mr. Mitchell's little jibe at the scientist who has not been there looks really rather foolish in view of our knowledge of the chemistry of the sun and stars.

For the rest, I depend on the Inspectors of Mines for my information, though I may not always fully understand that information, and I may say in passing, that these gentlemen are perfectly capable of, and accustomed to, locate bad places, independently of the courtesy of the mine managers—a courtesy about which, methinks, Mr. Mitchell doth protest too much. One would have thought that that courtesy might have been taken for granted. Still, every one recognises that the assistance of the managers is of the greatest value, particularly in saving the time of the Inspectors.

I do not think there were more than three "repeats" in the whole series, and, therefore, no harm is done in including them. Of course, several samples were often taken in the same stope or drive, but they were at different distances from the workers or in different parts of the locality in question. Such samples are perfectly independent of each other.

With regard to his sentence ending, "the remaining half should also have the benefit of the doubt," I only wonder whether this misunderstanding of my plain words is intentional or merely due to mental hebetude on Mr. Mitchell's part. It must have been obvious to all our *other* members that what I meant was that the percentage of bad places in the mines is nothing like so high as 42.6 in reality. My present idea is that the number of bad places all over is about 15 per cent. of the whole. Surely, I made it

plain enough that the inspectors were more or less on the search for bad places during this investigation,—which is natural in starting a new method of testing. It would have been of only academic interest to sample a lot of places which one knows, *a priori*, to be good. Mr. Bradford will please take this also as an answer to par. 2 of his criticism.

I think I am right in saying that practically all the samples were taken between the hours of 11 a.m. and 3 p.m. Now this period, though undoubtedly it is far and away the most unfavourable to "natural ventilation," yet is by no means the worst possible time of sampling. In fact, 2 p.m. is probably one of the best times in respect of avoiding the gases from blasting—which nearly always takes place about 3 p.m. The real worst time of sampling would be the beginning of the night shift, when the mine would be full of explosive fumes, at the same time as the temperature is only, say, 5 deg. less than the maximum of the day (see later).

I do not propose to follow Mr. Mitchell into those realms of stercoraceous wit where he is, no doubt, such a master, but will content myself with remarking that the weapon of false analogy which he uses can cut both ways. One might find out quite a lot about the incidence of such a thing as ankylostomiasis by the very examination he mentions.

At the end of his remarks Mr. Mitchell accuses me of having recommended artificial ventilation at the ends of drives, in addition to the exhaust from the drill. I have never even imagined (much less suggested) anything so drastic as this. The men at the face are, of course, all right: it is farther back that the trouble begins. My real opinion is that the removal of even explosive fumes from close places can be quite satisfactorily effected by the simple and cheap pipe device mentioned by Mr. L. Carter.

Now I come to Mr. Wager Bradford, and to begin with, I must acknowledge that he is a foeman worthy of my steel—and also perhaps the most liberal-minded of his profession. I am sorry that it is not one of his less worthy colleagues that I am "up against." What could be finer than his phrase: "The mines are not afraid of the truth?" It is the attitude which we all should endeavour to preserve in life—the truth even if the heavens fall! I fancy, however, after reading Mr. Bradford's 15th paragraph, that I shall not be far wrong in suggesting that he (or at least some of his colleagues) would rather have the truth quietly talked over in a certain purlieu of the Rand Club, than published broadcast in a Journal. To me, on the other hand, the dissemination of the truth (or what portion of the

truth I am able to see) is a public duty higher by far than even the interests—fundamental as they are—of the chief industry of this country. I have no prejudice against the mines: in most cases I do not even know what mines correspond to the letters in my order of merit; but I do feel that nothing can be allowed to interfere with the sovereignty of the truth.

Mr. Bradford thinks that my words have more weight because I happen to be Chemist to the Mines Department—which puts me on a pedestal, if you please. I desire to say that if my words are not true intrinsically, the man is a fool who would believe me on my mere authority. My views are my own, and were given to this Society in my private capacity, for, of course, my official duty is to keep quiet. It would be ridiculous to suppose that the Mines Department accepts everything I may say: in fact, several of the officials actively dissociate themselves from my views. Besides, the official position cuts both ways. It renders my defence more difficult by preventing me from discussing the results in detail. I found I could not do so without giving details which would practically "give away" the names of the mines.

I regret that I should have prejudiced my case by using those adjectives, but the fact is, I used them quite impersonally, without any idea of attaching blame to individuals. I meant to point out in unmistakable language the existing defects, and I think I succeeded in doing so: but really, in the absence of the names of the mines concerned, no personal accusation has been made.

I may say also in palliation of my offence, that it is my opponents, not I, who have labelled (or libelled) the last ten mines in the September order-of-merit as "scandalously unhealthy." I did not myself draw any hard and fast line between the sheep and the goats, although I must now take the responsibility for this inference which has resulted from compressing my remarks too much. The ten which I really had in mind were not the last ten in the order of merit, but were those mines characterised by grossly deficient general air-supply: most of them occur in the tables of upcasts and main-connections (B & C) as "1.5 at least,"—although it is true I have committed the error of including, through oversight, one or two unconnected levels.

These ten or twelve mines are characterised by the fact that they are merely the upcasts of the corresponding deep levels, and possess no downcast shaft connected with the outside air. They use as an air-supply air containing 0.8 to 1.0 parts of CO₂ per 1,000, air which has already passed through one or more deep level mines. Under these circumstances, the wonder is not that so large a number of bad samples

should have turned up, but that there should be any good ones at all except near the inlets.

I freely admit that prophecy has affected my opinion on certain of the mines, but it is scientific prophecy, not inspiration as Mr. Bradford calls it. My belief was in fact that most of the bad ones would continue to show, even after fifty or a hundred samples have been tested in each, a similar high proportion of bad places. This belief arose through the ordinary scientific process of analogy, from the position in the order of merit of the Ferreira Deep. Taking the 1904 investigation along with the present one, we had 17 bad samples out of 53 tested from that mine, or 33 per cent., a fact which certainly suggests that other mines showing 60 per cent. or more of bad samples under the same conditions must be in a much worse state than the Ferreira Deep—a very representative mine.

The tremendous area of the workings of these ten mines is, no doubt, very impressive, but it does not affect this question. It is what is known in political circles as a "red herring." If Mr. Bradford knew that there were five cases of plague within the 82 square miles of the Johannesburg Municipality, he would, I think, consider it cause for excitement.

The difference between us is evidently one of how high an ideal one has in view. If I have shown that there are on the local mines 200 places *where men are working*, which contain air exceeding in impurity even that of a Wanderers' political meeting and sometimes approaching that of the Black Hole of Calcutta, then I say—without haste and certainly without partisanship—that those 200 places will have to be attended to sooner or later.

It is a pity from the aesthetic point of view that Mr. Bradford, who shows in many cases the possession of a "pretty wit," should have allowed himself to degenerate so often into mere vituperation and other forensic tricks. The rest of his lucubration consists merely of the rhetorical artifice known in the vernacular as "rubbing it in." Instead of giving his figures for these ten mines in the form of a table, he takes each mine in turn, spreads himself over two pages of the *Journal*, mutilates my statistics to his satisfaction, and adds a caustic commentary. These are the methods of political rather than of scientific controversy. Surely it was plain enough that samples marked "1.5 at least" were not cases where the CO₂ was *exactly* 1.5 per 1,000.

I regret extremely that Mr. Bradford should have thought fit to assert that I appear to have a case to prove against certain mines, regardless of evidence. Surely, my critic might have noticed my statement on the other side of the account, which amounted to this: that there

were found to be some 20 mines in which the Inspectors of Mines failed, despite stringent methods of sampling, to find anything wrong. In view of this fact it must be clear that there is really something seriously wrong with the mines which gave the very bad figures. This is not "feminine logic," or "assumption," or "prejudice," or "a half truth." The fact remains that some ten of the mines require something much more radical in the way of alterations, something additional to "natural ventilation," whereas the others either require nothing at all, or merely local tinkering.

Mr. Bradford's criticism is, on the whole, so successful, that, had he confined himself to pulverising my adjectives, I should have been content to cry "peccavi" and to say nothing more, but since he has gone further and entered on ground where his footing is much more precarious, it becomes my plain duty to walk in and, metaphorically speaking, slay him. The chief opening he gives me is about upcast shafts, and I may say at once that Mr. Bradford shows himself in respect of theoretical hygiene, to be "a babe in these matters," to use the classic phrase of Cecil Rhodes. It is not a mere "pretty theory" that every man doing hard work emits at this altitude about 1.8 cub. ft.* of carbon dioxide per hour—it is a hard and inevitable fact; nor did I think it necessary to do more than "point out" to a man of Mr. Bradford's intelligence that it requires about 30 cub. ft. of fresh air *per minute* to keep the air which the man has breathed down to 1.5 CO₂ per 1,000. This is merely rule-of-three and is to be found in any text-book. Still, in case Mr. Bradford prefers remaining in outer darkness to looking up a text-book, I shall work out this particular example at large.

400 men and candles @ 1.8 per hour = 720 cub. ft. CO₂ per hour = 12 cub. ft. per minute CO₂ have been added to whole quantity (*x*) of air. The effect has been to raise the CO₂ in the air by 0.11 (0.15—0.04) per cent. Therefore 12 = .11 per cent. of *x*

$$\therefore x = \frac{120,000}{11} = 11,000$$

which $\div 400 = 27$ per head.

The result is that every mine which showed a "bad" upcast must have had, at the time of sampling, an air supply of only about 30 cub. ft. per minute per head. The only modification of this calculation is in the case of those mines already mentioned which have no downcast air; in their case the actual air supply may be much more than 30 cub. ft. per min., but nevertheless that air supply is only *equivalent* to 30 cub. ft. of *fresh* air in purifying power.

It must be remembered that the upcast sampling has nearly always been done during the

* Really about 1.5 for the man plus 0.3 for his candle.

hottest time of the day, so that this 30 cub. ft. is practically a minimum instead of an average value for the ventilation. It is clear also, I hope, that the calculation only applies to the state of ventilation existing *just before* the upcast sample is taken. There is nothing really wonderful in the variations in the CO₂ in the upcast of the same mine, since the temperature can vary within such wide limits as it does in this country; besides, there is another factor, namely, the existence in some upcasts of intermittent pumps worked by compressed air; when these are working, it is evident that the upcast sample would be made to appear much better than it is in reality. I fancy (although this is trespassing on the engineer's province) that great improvements in the ventilation in such mines would result, if compressed air pumps were employed only in downcasts and steam pumps in upcasts.

Finally, I note with extreme disappointment that, not content with demolishing me off his own bat, Mr. Bradford has been reduced to quoting the *London Mining Journal*. It is deplorable that a man of such ability and with such a good case should show so little discrimination, for I did not think that he would have stooped to use as a weapon the ignorant rhodomontade of that somewhat self-satisfied publication. It is not worth my while to correct the *L.M.J.* in matters of detail, but I will content myself with remarking that, extraordinary as it may seem, that journal is evidently ignorant of the existence of the Report of the West Australian Commission; it has no idea of what a CO₂ standard is used for; it has no idea that CO₂ standards are enforced in English schools and factories; and, indeed, gives the impression that its editor has read nothing for two years, since the appearance of the Foster-Haldane text-book—which, by the way, the *L.M.J.* evidently considers to be the acme of possible scientific achievement. It is an excellent little book, and, no doubt, was quite the *dernier cri* in 1905, but we have progressed since then.

This then is the precious journal which lectures me in the following beautiful piece of periphrastic journalese, "a view which connotes a probable unfamiliarity with recent progress in mine sanitation research" and later on in the same editorial, talks smugly of the "general low level of education in South Africa." I hope that *Journal* now feels that it had better look before it leaps next time; if it wants any further details, it should read Dr. Irvine's excellent summary of the question on pp. 168-9 of the December *Journal* of this Society. If it does so, it will probably be interested to read also the reference to strabismus on p. 171.

As for the local publication which copied and amplified the slander, I fancy that by this time it has probably realised that it made a very bad mistake.

Mr. Bradford makes a second appearance in last month's *Journal* in which he is by comparison almost lamblike. In reply, I wish to say that it does not much matter *when* the samples are taken—so long as they are taken under working conditions and so long as we know roughly what the time of day and the temperature prevailing above ground is.

Mr. Bradford certainly has established the point that it is necessary to analyse the compressed air in order that the calculation of the air supply from the proportion of CO₂ in the upcast may be an exact one. Still, as I will show later on, the error introduced by ignoring the compressed air is less than 20 per cent. in all cases, and really it is not of much consequence whether a mine is getting 30 or 36 cub. ft. per min. all over—such a quantity is far too small.

I wish to thank Drs. Macaulay and Irvine for their useful comparative table (A) December *Journal*, showing the close resemblance in respect of the degree of local vitiation between my results and those of Mr. Mann of Western Australia. I take it that their point is that those particular 16 Witwatersrand mines which are on all fours with the Westralian ones in having a sufficient *general* air supply, are also just as good—or rather, a little better—in respect of the number of bad samples found in particular localities as the Australian mines; in other words, there is nothing to choose between the two countries in respect of the defects of *local* ventilation. This point, which has also been mentioned by Mr. Lane Carter, is quite clearly established, and it will in fact be seen from my new determinations that there are far more than 16 of our mines which are as good as the Australian ones in that particular respect.

The case is different, however, when one considers the whole of the Witwatersrand mines together (63 investigated to date). The figures for local vitiation remain almost unaltered, whereas there are now some 20 per cent. of bad places in main connections and upcasts, places which were never found to be bad in Australia (eighteen samples only taken there however).

My medical friends have therefore issued a pronouncement regarding "the upshot of the whole matter." I am confident, despite this *ukase*—for that is all it is—that I have proved that there is more in the Rand ventilation problem, as regards, say, one-sixth of the mines, than merely *local* vitiation.

They have also suggested that 15 per cent. is a "strict" CO₂ standard. In another place, they say it is a "reasonable" one. I will remind

them that at the sub-committee meeting (see September *Journal*) I proposed to make the standard .18 or .20 to compensate for the altitude (see later), but that they wished, as they put it, to err on the safe side by recommending the stricter .15 standard—and being outvoted, I agreed.

Regarding this .15 per cent. CO₂ standard, there are two most important points to be mentioned, which, by the way, I believe are new.

The first is that this figure is an attempt to strike a mean for the different parts of a mine. .15 per cent. is almost certainly too strict for winzes, dead-ends and development work generally; but, on the other hand, it is far too lax for main connections and upcast shafts. I pointed out in the original paper that .16 per cent. CO₂ in the upcast corresponds to an air supply in the whole mine, of only 27 cub. ft. per minute per head. To get the 70 cub. ft. per minute prescribed by the Mining Regulations, it would be necessary to make the standard for upcasts only .085 per cent. CO₂ as a maximum—a figure which would be more than twice as difficult to maintain as the proposed .15 per cent. standard.

I think, as a matter of fact, that the Mines Department, in proposing a *universal* .15 per cent. standard, is in reality making a big concession to the mines; it is asking for only 30, instead of 70 cub. ft. per minute of fresh air, but expects in return for this concession, that these shall be real and not supposititious cub. ft. The test bottles, with the aid of a sum in proportion, will be able to decide whether in any particular case the correct amount of air is available. It is true that the calculation would be fallacious under certain circumstances, but there should be no difficulty about it in stopes, levels and upcasts, with ordinary precautions.

It is because Article 56 of the regulations has proved *inoperative for local ventilation* that a compulsory standard of maximum vitiation is suggested—it is the only means of *making* local ventilation good all over a mine. To put it in a nutshell, the effect of the .15 per cent. standard will be that there will never be more than .03 per cent. of CO (Dr. Haldane's limit) or more than traces of organic contamination in mine air.

I think my opponents have rather mixed up the two sides of this standard question. The question of the institution of a standard for checking mine ventilation is quite independent of the question whether the existing test samples have been taken in proper places. If our experiments are right, it is evident from the results that a standard is necessary: if on the other hand they are wrong and the mines are no worse than others, why do they object to a standard?

The other point is that a standard of .15 per cent. CO₂ in the Witwatersrand mines is not exactly the same thing as a .15 standard at sea level, or in Australia. This is rather an intricate point, and depends on the fact that although men working at different altitudes give out the same *weight* of carbon dioxide per minute, the weight in question possesses a different bulk at different altitudes. The result is that more air is required here to dilute that amount of CO₂ to .15 per cent. In figures, the true equivalent of the .15 standard of Australia is about .17 per cent. on the Witwatersrand for mines of the same average depth, or in other words, a .15 standard here corresponds to about .13 in Western Australia.

I first discovered this important point by accident whilst trying to check the method on air to which CO₂ had been added to make it up to .15 per cent. of CO₂. I found that this did not cause the complete disappearance of the pink colour, and that the latter only took place with air of .165 per cent. CO₂ in the laboratory. The difference is due to the fact that the test-solution was made up for air at the pressure existing at 1,200 ft. underground, and was therefore wrong for air at the laboratory pressure. The error corresponds exactly with the difference in pressure, a fact which shows that the method, originally a theoretical one, is also correct in practice, that is to say, for mines between 1,000 and 2,000 ft. deep. The corollary of this is that the solution corresponds to a .14 standard in a very deep mine, and to .13 per cent. at sea level.

In spite of all this, I myself think that it is better for the sake of simplicity and uniformity, to stick to the one test for every part of a mine; and *pace* Dr. Irvine, .15 is a fairly, but not an unduly strict, *general* standard.

I quite agree that it is practically almost impossible to ensure that *all* working places shall *always* have air containing less than .15 per cent. of CO₂, but that is no excuse for not making a good try at it.

It is a fact that the Witwatersrand mines are much better ventilated *in their central portions* than one would expect, considering that about 95 per cent. of them rely solely on "natural ventilation." This circumstance is, however, due almost entirely to the prevailing dryness of the air (not much in evidence this season) in the Transvaal, and not at all to the engineering methods of the Rand. This dryness causes a tremendous fall of temperature owing to evaporation of falling water in the air descending the downcast, sometimes amounting to nearly 20 deg. F. On the other hand, this favourable condition is partially—and sometimes wholly—counterbalanced by the low temperature of the

rock in the Transvaal. It is only at 4,000 ft. depth that we get temperatures comparable with those prevailing in shallow mines situated in other countries, and since "natural" ventilation depends on the difference of temperature between the downcast air and the rock in the body of the mine, it becomes severely handicapped in this country when it rains. It is perfectly certain that had these mines been situated near Pietersburg or Durban, half of them would have been unworkable in hot weather, because "natural ventilation" would stop altogether whenever the outside temperature got above, say, 90 deg. F. (see later on p. 256). It is well known that several of the Witwatersrand mines have upcast shafts in which the current reverses in hot weather, and it is obvious that while the reversal is taking place, there is no ventilation at all in that mine or portion of a mine. Of course, that is a phenomenon which is known elsewhere, but that is no reason why we should not try to cope with such a problem. I say again, on this fact alone, that those particular mines are, during the period of reversal, disgraceful all over (except, of course, at compressed air exhausts representing perhaps 5 per cent. of the workings).

Now I come to Mr. McArthur Johnston. His contribution is distinguished from the rest by containing some concrete addition to our knowledge of the ventilation question, and I propose, therefore, to devote some attention to his new data.

He hints in his opening sentences that I have failed to shed the "full truth" on our problems. This remark does not come particularly well from him, since he has been guilty of *suppressio veri* in keeping back for 2½ years these experimental results of such great importance—results which would certainly have been of far more value in 1904, when misrepresentations of the ventilation and Miners' phthisis questions were rather rife. Now that we have got them I hasten to thank Mr. Johnston for his figures, both for their intrinsic value, and also because it is so unusual for the people who discuss these things to have done any real work in the subject themselves. I am here speaking of the Transvaal, of course.

Mr. Johnston is the only one of my critics who has hit the nail on the head as regards the true nature of our problem, by saying that the real difficulty is the impurity of the total air of a mine. It has been my contention all along that in some fifty of the mines ventilation is merely a local question (in varying degrees); and it was just the remaining dozen or so which I condemned—in perhaps unnecessarily severe terms,

I admit—and I condemned them because their total air-supply is so deficient that they cannot help being in a bad state "all over." These are mines with no downcast, and I cannot see how they are to be put right except by either connecting them with the outside air, or by using mechanical means to direct the downcast air of their deeper neighbours into them. But I must insist that I have not condemned any mine in which the ventilation problem is a merely local one.

Again, when I said that a bad upcast condemns a whole mine, I thought that I had made it clear (in discussing "Mine d") that the condemnation was based on its air-supply being only 30 cub. ft. per minute, instead of the 70 prescribed by the Regulations. It does not really matter if parts of the mine concerned have a good supply; the rest must be correspondingly very bad if the average is 30. Mr. Johnston's idealised mine with four equal levels will serve to illustrate this point—although Mr. Johnston will probably think that I am taking its name in vain. Assume it is supplied with outside air of .04 per cent. CO₂. Since there is 1.6 per 1,000 of CO₂ in the upcast after passing through the mine and rising above the four levels, the total

air-supply is $\frac{1.8}{1.6 - .4} \cdot \frac{1000}{60}$ or 25 cub. ft. per

minute per head (I neglect the explosives and compressed air, because, as regards CO₂ at least, they nearly cancel one another in their effects). This 25 cub. ft. is not very much air "to play with," anyway; and calculation shows that Mr. Johnston's first level will carry 38; No. 2, 33; No. 3, only 30; and, finally, his charming fourth level will have an air-supply of 14 cub. ft. In such a level one could shut one's eyes and imagine oneself in a winze. I fancy, after all, that I did "weigh my words when I formulated my statement," if my critic will allow me to quote his somewhat gravid phraseology.

Since, owing to the carelessness in writing, I seem to have given some people the impression that I think that CO amounts always to one-tenth of the CO₂ in all parts of a mine, I hasten to correct this view—an almost ridiculous one. The statement I made applied only to air from blasting; of course the ratio of CO to CO₂ is very much less in parts of the mine where the CO₂ is mainly due to respiration. Dr. Haldane has, in fact, calculated that in the particular case of the Ferreira Deep upcast air, the CO cannot amount to more than one-eightieth of the CO₂. (This *Journal*, November, 1905, p. 177). It is evident also that in most of the Witwatersrand mines CO is not of much consequence in "working mine air," except during the first two

hours after blasting; but CO is *always* of consequence in the dozen mines which have a poor current of air: they are in the nature of the case practically never free of CO.

I am sorry to have caused Mr. Johnston that "mental somersault," and trust that he will feel able to bear the above explanation with at least *some* degree of equanimity.

To come to what Mr. Johnston calls the more serious part of his contribution, may I ask him if the rest of his remarks are in comparison jocular, or frivolous? I had not noticed it. However, the important thing is that his figures do really permit of our gaining an idea of the general condition of the whole of this characteristic mine. I desire to congratulate the Meyer & Charlton authorities on their courage in allowing themselves to be discussed by name, for the mine belongs to the "condemned" class in respect of having no true downcast. It uses the waste air of its neighbours, eked out with compressed air. The average of the air supply figures is 1.11 CO₂ per 1,000, a figure so shocking that one is driven to suppose that some of the samples included in the list are not representative. Surely such values as 1.6 per 1,000 (and there are 7 cases out of 23 which give over 1.2 per 1,000) in an *air supply* of a mine do not happen often. I am glad to hear that the mine has recently had the energy to put in an electric fan, which has caused an immense improvement.

Nevertheless Mr. McArthur Johnston uses this appalling mine for the purpose of attempting to prove that I had no right to condemn it as a *whole*, since, as he shows, its average is below the 1.5 per 1,000 standard. Now the trouble is that such a mine has no margin of safety. Out of the 52 samples as many as 30 were 1.2 or over, and it is easy to see that a little thing like a rise of 5 deg. in temperature or some indifference towards the quality of compressor lubricant, would have sent the whole mine over 1.5 per 1,000.

Again, the maximum amount of total upcast air observed by Mr. Johnston was 15,000 cub. ft. per min., divided over some 250 (I understand) persons. This is 60 per head—quite a respectable amount—but some results of my own seem to show that even if the air supply amounts to nearly 50 cub. ft., yet owing to its contaminated state it is only equivalent in purifying power to a supply of 22 cub. ft. per min. of outside air. (See Table, p. 261).

To come to his startling figures for vitiation of compressed air itself, I can only hope that they are exceptional. Dr. Haldane's results and those of the West Australian Commission gave nothing higher than 0.8 CO₂ per 1,000, and the Miners' Phthisis Commission (three samples only, how-

ever) gave as its highest 0.9 per 1,000 (Mr. Williams rather exaggerates this).

Carbon dioxide in excess in compressor air is so preventable a defect that its occurrence on the Meyer & Charlton must have been due to either a badly vitiated inlet or the use of some abominable substitute for lubricant. Possibly that compressor itself required relegating to the scrap-heap. Anyway, the *onus* of proving that they cannot keep their air supply down to 0.5 CO₂ per 1,000 is certainly on the mine; if they do succeed in establishing the fact, then certainly the standard should be raised in their case, but really some people seem to think that an intake for air to compressors vitiated by smoke stacks, engine-room fumes or ash heaps—that such a vitiated intake is an "act of God," or something equally inevitable. The same argument applies to Mr. J. R. Williams' suggestion that some of the CO₂ is due to decay of timber. The *onus* of proving that this is appreciable under existing conditions is on the mine concerned.

In any case, Mr. Johnston's figures are not so fatal to my argument as they appear at first sight to be, since compressed air does not as a rule form more than $\frac{1}{12}$ of the total air-supply of the mines. Evidently the Meyer & Charlton is exceptional in that respect, and, that being so, one would have expected their compressed air to be kept in a better condition. I do not agree that vitiation by CO₂ arising from overheated lubricant is entirely of no account, since it is pretty certain (though actually demonstrated in a few cases only) that such CO₂ is accompanied by a little CO, and is therefore probably quite as objectionable as air containing the same proportion of CO₂ resulting from respiration alone. I therefore cannot agree that a CO₂ standard is inapplicable in such cases. It would still be an indispensable *index* of other and more serious impurities (CO in this case), just as it is merely an index for organic poisons in its ordinary usage.

Mr. Ward has asked a question about the effect of the oil in the drills: since this oil is always cold it cannot harm the air beyond contributing somewhat to the unattractive smell of compressed air. There is not much for me to notice in Mr. Lashinger's remarks, as I am not concerned with engineering difficulties involved in the details of artificial ventilation. I am rather inclined to think that the "deep deeps" will be found to be more easily ventilated than the present mines—provided that they have two proper shafts. His suggestion for the chemical purification of compressor air seems to be an excellent one, but I should think that in this case also prevention is better than cure.

In the hope that the question of the dependence of natural ventilation on temperature is of interest to our members, I take the opportunity of mentioning here an interesting discovery which Mr. Langley, Inspector of Mines, and I worked out in 1904. The credit of the discovery belongs to Mr. Langley alone; I merely elaborated it somewhat. It is this: that the quantity of air circulating in the Ferreira Deep could be calculated for any day, without descending the mine, by using a formula which requires only one observation, viz., the reading of the wet-bulb thermometer at the surface. The formula was: Q (cub. ft. per min.) = $3,500 + 1,600(68 - w)$; in which 3,500 represents the total compressed-air exhaust of that date, 68 is the average temperature of the upcast shaft (almost constant throughout the year) and w is the wet-bulb reading. 68 deg. is the rock temperature at 1200 ft. corrected for the adiabatic decompression of the air in ascending the upcast. For the benefit of that fairly numerous section of the population which does not understand statistics, I append a diagram showing the close cor-

min. per head. As a matter of fact, the wet-bulb reading never rose above 62 deg. F. even on the hottest of days, but it is easy to imagine circumstances in which it might do so. It would do so every day in summer here were it not for the altitude. Also, it is evident that the other deep levels, with two equal shafts, will behave in the same way. Mr. McA. Johnston's second table, on p. 204, shows in an interesting way this variation of air supply with temperature in course of the day. On the other hand; it can be seen that the maximum of CO_2 occurs several hours later than the maximum temperature.

In view of this discovery I think it is most desirable that mines should keep a record of hygrometer readings in the form of an automatic record. The mines are at present required by law to keep a register of the air passing in them; but this is a regulation with which they do not comply, chiefly because it is impracticable on account of the smallness of their air-supply. Anemometers do not give reliable readings when the air passing is less than 40 cub. ft. per min

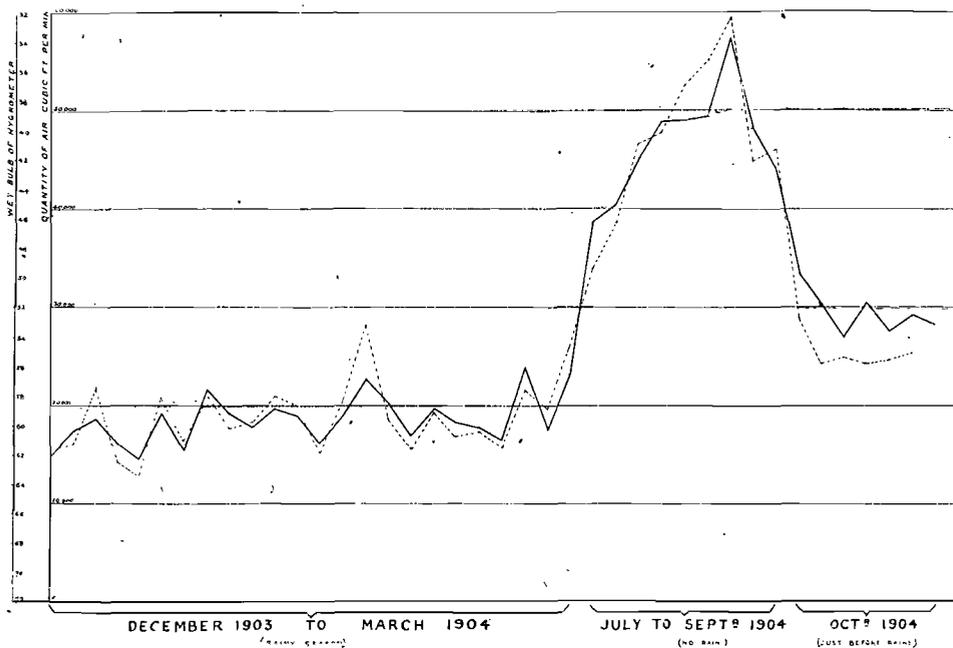


DIAGRAM SHOWING THE QUANTITY OF AIR (BLACK LINE) ACTUALLY MEASURED IN THE FERREIRA DEEP AND ITS CONNECTION WITH READINGS OF THE HYGROMETER (DOTTED LINE).

respondence between the two curves representing quantity of air and distance of wet-bulb reading from 68 deg. respectively. One result of this investigation is to show that if the wet-bulb reading were to reach 68 deg., natural ventilation in this mine would stop, and the mine would have to depend entirely on the compressed air supply amounting to less than 10 cub. ft. per

per head, *i.e.*, in upcasts. On the other hand, a register of wet-bulb readings is a very easily made one, and will be as satisfactory a check on the general ventilation of the mine as the indicator-diagram is of the efficiency of an engine. It will, of course, be necessary at first to find the "factor" for each mine by means of CO_2 tests in the upcast.

APPENDIX.

TABLE I.—Order of Merit of Witwatersrand mines in respect of ventilation at end of February, 1907.

No.	Letter in Sept. Order of Merit.	Position on Rand.	No. of Samples.	No. bad	Remarks.
1	F	Rd, outcrop	9	0	All good* except two
2	new	Ra, deep	8	0	All good except one
3	new	Ra, outcrop	8	0	Large air supply
4	new	G, outcrop	7	0	All good* except one
5	A	G, deep outcrop	6	0	No new samples
6	new	G, outcrop	6	0	All good
7	new	B, outcrop	6	0	All very good
8	G	Ra, outcrop	6	0	" "
9	E	Rd deep	14	1	Rest satisfactory
10	J	Ra, outcrop	14	1	" "
11	C	Rd, outcrop	11	1	New tests very good
12	X	Rd, outcrop	8	1	" "
13	new	Rd, outcrop	8	1	Rest good
14	I	B, outcrop	7	1	Poor supply
15	M	B, outcrop	6	1	" "
16	K	JE, deep outcrop	6	1	No new samples
17	T	JW, deep deep	11	2	Poor upcast
18	O	JW, deep outcrop	11	2	All middling, or poor
19	new	Ra, outcrop	8	2	Poor air supply
20	new	Ra, outcrop	18	3	Very erratic
21	g	Ra, deep	15	3	All new tests good
22	a	G, deep	14	3	Good; air supply rather poor
23	b	Ra, outcrop	12	3	" "
24	f	JW, deep outcrop	11	3	Three others nearly over 1·5
25	e	JW, deep outcrop	10	3	New tests excellent
26	h	Ra, outcrop	9	3	New tests excellent
27	new	G, deep	8	3	Poor air supply
28	new	Rd, deep	8	3	" "
29	Y	B, outcrop	7	3	Poor air supply
30	new	Rd, outcrop	7	3	Shaft sinking
31	new	G, outcrop	6	3	" "
32	c	G, deep	14	4	Poor air supply
33	i	Ferreira Deep	13	4	Upcast poor
34	new	JE, outcrop	11	4	Stopes rather neglected
35	H	JW, deep	10	4	Stopes far too bad
36	N	Meyer & Charlton	8	4	Very deficient air supply
37	new	JE, deep outcrop	6	4	Poor air supply
38	U	G, deep outcrop	19	5	Good supply, poor distribution
39	d	JW, deep	15	5	Poor supply, good distribution
40	V	G, deep	13	5	Poor air supply
41	s	Ra, outcrop	12	5	New tests good
42	m	Rd, deep outcrop	11	5	Poor air supply
43	r	Rd, outcrop	8	5	New ones all good
44	Z	B, outcrop	8	5	" "
45	S	G, very deep outcrop	22	6	Mostly poor
46	B	G, deep	18	6	Poor air supply
47	P	G, deep outcrop	13	6	Local neglect
48	q	JE, outcrop	12	6	All the others nearly over 1·5

No.	Letter in Sept. Order of Merit.	Position on Rand.	No. of Samples.	No. bad.	Remarks.
49	l	G, deep deep	16	7	Including mine G
50	n	JW, deep outcrop	15	7	All others nearly bad
51	t	" "	12	7	" "
52	k	B, outcrop	12	8	Only one good* sample
53	R	JW, deep outcrop	11	9	All new tests bad
54	p	JE, deep	20	10	" "
55	new	JE, outcrop	7	5	" "
56	W	JW, deep	23	11	Nearly all stope samples bad
57	new	G, deep deep	20	14	Singleshaft mine
58	o	JE, deep outcrop	31	16	Lower stopes extremely bad
Totals			665	227	or 34 per cent. bad

* "good" means under 1·3.

The following have also been done, but are not included in the order of merit through lack of data.

No.	Previous Order of Merit.	Position.	No. Samples.	No. Bad.
I.	D	G, deep	5	0
II.	new	JE, outcrop	5	1
III.	L	JE, deep.	4	1
IV.	new	JW, outcrop	5	5
V.	j	Rd, deep (closed down)	3	3
63 mines			22	10
Order of merit			665	227
Totals			687	237
Last order of merit			256	109
New tests			431	128 or 30% bad

TABLE II.—DETAILS OF NEW TESTS.

Reference (Table I.)	No. of Mine and Locality.	CO ₂ per 1,000.	No. of Persons.	No. of Machines.	Remarks.
No. 1, formerly F					
1	Upcast shaft (No. 2)	0·8			40 ft. from collar
2	Stope, 7—1 east	1·3	20	H*	
3	" 8—1 "	1·4	30	3	
4	Level, 8th "	1·2	?	?	
5	Drive, 9th, west	0·9	9	1	Near end
6	" 9th level	0·9	9	2	" "
No. 2 (new)					
1	Shaft, main upcast	1·1	40		40 ft. from 450 persons in collar mine
2	Upcast, 8th station	1·2			
3	Stope, 12—1, WMR	0·8	12	2	
4	Stope, 12—1, EMR	0·8	10	H	
5	Stope (8—1) MRW	1·0	?	1	

Reference (Table I.)	No. of Mine and Locality.	CO ₂ per 1,000.	No. of Persons.	No. of Machines.	Remarks.
6	Stope (9-1) MRW	1.0	?	H	
7	Stope (11-1) MRW	1.0	?	H	
8	Level (11th) east of shaft	1.4			
<u>No. 3 (new)</u>					
1	Shaft, upcast,	0.7	40 ft. from collar		530 persons in mine
2	Stope, 9-7, leader	1.4	27	H	
3	Stope, 9-2, WR	0.9	28	H	
4	Stope, 9-4 west reef	1.0		H	
5	Stope, 11-7 leader	1.4		H	
6	Level, 10th	0.5			400 ft. south of shaft
7	Station, 11th, upcast	0.8			
8	Station, 6th level upcast	1.3			
<u>No. 4 (new)</u>					
1	Station downcast	0.4			20 ft. from shaft
2	Stope, 7th level	0.5		H	Half way down
3	Stope	0.6	20	2	
4	Drive	0.7	6		Air blowing off
5	Drive, 11th level	1.2	2		300 ft. from winze, air on
6	Winze, below 11th level	1.4	7		Air on 50 ft. above level
7	Boxhole, 11th level, west drive	1.0			Machine stopped while test made
<u>No. 6 (new)</u>					
1	Stope	0.6		H	30 ft. from top
2	Stope (boundary)	0.6			
3	Level (5th)	1.0			400 ft. from shaft 100 ft. from last stope
4	Drive 5th level W.	0.7			Machine stopped
5	Winze	1.2			20 ft. from level
6	Winze, 4th level	1.3			70 ft. down
<u>No. 7 (new)</u>					
1	Upcast shaft	1.0			Near surface (this only takes a portion of upcast air)
2	Stope, No. 4, MR 6th level	0.7	30		
3	Stope, 5th level E, No. 1 leader	0.7	28		
4	Stope, 5th level E, No. 2 leader	0.7	30		
5	Drive, No. 5, MRL east	1.0	4	1	370 ft. in from cross-cut
6	Drive, No. 5, MRL west	1.0	?	1	Also from cross-cut
<u>No. 8=G</u>					
1	Shaft, main up-cast	0.6	40 ft. from collar		200 persons in mine
2	Stope, 8-1, E	0.7	20	2	
3	Stope, 7-1, W	0.6	10	1	

Locality.	CO ₂ per 1,000	No. of Persons.	No. of Machines.	Remarks.
<u>No. 9=E.</u>				
1. Upcast shaft (No. 2)	1.1			40 ft. from surface
2 Station, 4th, No. 0.6				3 shaft
3 Station, 5th, Up-cast	0.9			
4 Stope, 5-4, west	0.8	26		10 ft. from face
5 " 6-2 "	0.8	30	H	
6 Drive, 5th level MRW	1.0	?	2	At face
7 Level, 8th, MRE	1.1	3	1	
8 Raise, 7th level E (90 ft. up)	1.6*	5	1	30 ft. from face
9 Raise, 8th level W (80 ft. up)	1.1	4	1	
10 Winze, 8th level W (bottom)	1.4	4	0	60 ft. deep
11 Winze, 7th level E (bottom)	1.1	4	0	45 ft. deep, no air on
<u>No. 10=J</u>				
1 Upcast shaft (No. 5)	0.8			40 ft. from collar
2 Stope, 6-1 WMR	0.8	25	H	
3 Stope, 7-1 WMR	0.7	12	H	
4 Stope, 7-1 ESR	0.9	26	H	
5 Stope, 8-1 ESR	0.6	15	H	
6 Stope, 8-2 ESR	0.7	26	H	
<u>No. 11=C</u>				
1 Upcast shaft	0.7			40 ft. from collar
2 Stope, 8-3 east, leader	1.1	36	2	At bottom
3 Stope, 9-2 east	1.2	30		"
4 " 9-1 west	0.7	30		"
5 Drive, 8th level east	1.6*	6	1	Near end
6 Drive, 9th level west	1.2	7	1	600 ft. in
<u>No. 12=X</u>				
1 Shaft, main up-cast	1.2			incline 40 ft. from collar
2 Stope, 7-1 WSR	0.9	30	H	
3 " 9-3 "	0.7	35	H	
<u>No. 13 (new)</u>				
1 Shaft, main up-cast	0.5	40 ft. from collar		500 persons in mine
2 Drive, 8th level E	1.2			100 persons travelling in ladderway
3 " 13th "	0.5			500 ft. from shaft
4 Level, 8th W	0.9	25	3	Top of stope
5 Level, 14th east	1.6*			1,200 ft. from shaft
6 Stope, No. 3 W 12th level	1.1			
7 Cross-cut, 10th level	0.6			To upcast
8 Cross-cut, 15th level	1.1			To upcast
<u>No. 14=I</u>				
1 Upcast shaft	1.4			40 ft. from collar
2 Station (down-cast)	1.4	?	0	Below connection

* "H" means that no machines, but hammer boys were working.

Locality.	CO ₂ per 1,000	No. of Persons.	No. of Machines.	Remarks.
<u>No. 14=I. (Continued.)</u>				
3 Drive, 10th level, 0·8 W	4	2		400 ft. in
4 Drive, 11th level, 1·3 E of cross-cut	4	0		250 ft. from cross cut
5 Drive, 11th level, 1·4 W of cross-cut	10	1		"
6 Winze, 11th level, W of cross-cut	2·5*	3	H	60 ft. down
<u>No. 15=M</u>				
1 Upcast shaft	1·2			40 ft. from collar: strong current Taken at entrance
2 Stope, No. 2 MR, 0·9 5th level west				
3 Drive, 5th level, 1·3 MRW	5	?		1,000 ft. in: 25 ft. from face
<u>No. 17=T</u>				
1 Upcast (No. 2) 60 ft. from surface	1·3			900 underground
<u>No. 18=O</u>				
1 Upcast, 50 ft. from surface	1·3			650 persons underground
2 Stope, 5th level, 1-6, east	1·3	30	?	
3 Drive, 8th level, SRE	1·4	6	1	6 ft. from face
4 Drive, 8th level, SRW	1·4	6	1	" "
5 Drive, 8th level, MRE	1·6*	7	1	" "
6 Drive, 5th level, SRE (Star)	1·3	6	1	" "
7 Drive, 5th level, from cross-cut	1·4	7	1	" "
<u>No. 19=(new)</u>				
1 Shaft, main up-cast	1·6*			550 persons in mine
2 Station, 7th	0·7			Upcast
3 Stope, 10-2 WR	1·3	30	H	
4 " 10-4 leader	1·3	25	H	
5 Stope, 8-1, north leader	0·7	30		
6 Drive, 8th leader	1·0			1,000 ft. in.
7 " 9th WR	1·6*			800 ft. in.
8 " 10th north boundary	1·1			
<u>No. 20 (new)</u>				
1 Upcastshaft(No. 3)	0·6			40 ft. from collar
2 Upcast (main upcast)	1·4			
3 Upcast shaft, No. 4	0·7			
4 Stope, 8-3 east	0·8	2		
5 Stope, 10-1 east	0·8	22		
6 Stope, SR, 300 ft. west of shaft	0·8	22	H	
7 Stope, 8-1, SR, west of shaft	0·8	25	H	
8 Stope, 8-2, SR, west of shaft	1·4	20	H.	

Locality.	CO ₂ per 1,000	No. of Persons.	No. of Machines.	Remarks.
<u>No. 20 (new). (Continued.)</u>				
9 Stope, 6-1, SR, west of shaft	1·3	16	H	
10 Level, 11th, east	0·8			
11 Level, 11th, west of shaft	0·7	8	2	
12 Drive, 7th level, W	0·6			
13 Station, No. 3 shaft	0·8			
14 Raise(10-2 east)	1·4	4	1	40 ft. up.
15 " (11-2 west)	1·0	7	2	80 "
16 " (11-3 ")	1·6*	3	1	25 "
17 Winze, 9-1, west	1·6*	5	H	75 ft. deep
18 Winze, 11-1, west	1·6*	4	H	50 "
<u>No. 21=g</u>				
1 Upcastshaft(No. 1)	0·8			40 ft. from collar:
2 Upcastshaft(No. 2)	1·1			" "
3 Station, 10th level upcast	0·6			
4 Station, 13th level upcast	0·6			
5 Stope, 6a west	0·6	23	H	
6 " 12-1, MRE	0·7		H	Underhand
7 " 13-1, east	1·3	30	H	
8 " 13-1, MRL	0·8			
9 Drive, 14th level, west	1·2			Beyond shaft
10 Level, 12th	0·7			
11 Winze, 14-1, west	0·7	5	1	
<u>No. 22=a</u>				
1 Upcast shaft	1·1			40 ft. from surface
2 Stope, 1 east 7 MR	0·7	?	?	230 ft. from cross-cut
3 Stope, 1 east 6 MR	1·0	?	?	
4 Stope, 2 west 6 SR	1·0	?	?	
5 Drive, 7th level E	0·6	5	1	Full of dirt near face
6 " top of MR, stope	1·4	?	?	Furthest point north from shaft.
7 Winze, No. 1 S Reef	1·3	?	?	South side
<u>No. 23=b</u>				
1 Stope (back) 2-5	0·8			
2 " 5-5	0·8			Underhand
3 Stope, 4-3, WR	0·6			At box-hole
4 Stope (back) 6-4	1·0			
5 Cross-cut 6th level	1·6*			Near upcast station.
<u>No. 24=f</u>				
1 Stope, 14 MRL east	1·3	20	2	35 ft. down
2 Stope, 13th leader east	1·0	10	3	25 ft. down
3 Stope, 14th leader west	1·1	20	3	60 ft. down
4 Stope, 15th S Reef east	1·4	50	H	50 ft. down

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Locality.	CO ₂ per 1,000	No. of Persons.	No. of Machines.	Remarks	Locality.	CO ₂ per 1,000	No. of Persons.	No. of Machines.	Remarks.
<u>No. 24 = f. (Continued.)</u>					<u>No. 30 (new)</u>				
5 Drive, 15th level S. Reef west	1.4	8	2	8 ft. from face	1 Shaft, upcast No 1	1.6*	24	per-sons shaft sinking	60 ft. from bottom
6 Raise, 15th level S. Reef west	1.4	5	1	" "	2 Drive, 4th level ESR	1.6*	4	1	From No. 1 shaft
<u>No. 25 = e</u>					3 Drive, 3rd level EMR	1.6*	4	1	Machine stopped 30 minutes
1 Station (upcast) 9th level	0.9				4 Drive, 2nd level SRW	1.3	4	2	Machine stopped 5 minutes
2 Stope, 7-1, ES Reef	0.8		H		5 Drive, 2nd level MRE	1.4	3	1	Machine stopped 5 minutes
3 Stope, MRL (9-1 E)	0.8		1		6 Drive, 2nd level west	1.4	0	0	Beyond upcast
4 Stope, S Reef (9-1 W)	0.8		H		7 Drive, 2nd level east	0.6	6	H	" downcast
5 Level, 6th, near upcast	0.9				<u>No. 31 (new)</u>				
<u>No. 26 = h</u>					1 Stope, 7th level	1.6*	12		140 ft. from box-hole
1 Level, 5th, east	0.5				2 Stope 1, E, 5th levels	1.6*			
2 " 7th, "	0.5				3 Stope 5, E, 4th level	1.0	20		
3 Stope, 6-3 east	0.5	35	?		4 Stope (hand), 7th level	1.0			Near No. 4 shaft
4 " 7-3 "	0.7	30	?		5 Drive, 7th level	1.2			250 ft. from box-hole, machine working
5 " 7-2 west	0.7	33	H		6 Winze, 7th level, 50 ft. down	1.6*			130 ft. from box-hole
<u>No. 27 (new)</u>					<u>No. 32 = c</u>				
1 Upcast shaft	1.3			60 ft. from surface	1 Upcast shaft	1.4			50 ft. from surface
2 Upcast shaft (No. 2 incline)	1.3			" "	2 Stope, 11th level NRW	1.0			400 ft. from shaft
3 Stope, entrance to No. 3, MR	2.0*			1,070 ft. from shaft	3 Stope, 11th level SR	1.3			1,250 ft. " "
4 Stope, No. 1, MRW	1.6*			680 " "	4 Drive No. 11, middle Reef	1.3	6	2	2,260 ft. " "
5 Stope, No. 2, MRW	1.4			400 " "	5 Drive, 12th level NRW	1.0	?	2	500 ft. " "
6 Drive (cross-cut), No. 1 level	2.3*	?	1	1,250 " "	6 Raise, 11th level, 40 ft. up	1.4			2,200 ft. " "
7 Drive, No. 7, west of shaft	1.4	?	?	No connection	7 Raise, 11th level, SR	2.5*	2	H	20 ft. up
8 Drive, No. 7, 140 ft. from incline	1.4	?	1		8 Winze, 11th level, NR	1.3	6	2	170 ft. down, 450 ft. from shaft
<u>No. 28 (new)</u>					<u>No. 33 = i (Ferreira Deep)</u>				
1 Shaft, main up-cast collar	1.3	40		560 persons in mine	1 Upcast (No. 2), 50 ft. from surface	1.4			870 persons underground
2 Stope, 4-3, ESR	1.1	35	H		<u>No. 34 (new)</u>				
3 " 7-1 "	0.8	48	H		1 Upcast Shaft, West incline	1.4			Near surface
4 Stope (3-1) west	1.6*		H		2 Stope 11, MRLW	1.6*	2	2	30 ft. down
5 Stope (6-2) west	1.6*		H		3 Stope 10, MRLE	1.4	18	2	50 ft. "
6 Level, 3rd west	1.6*			150 ft. from upcast	4 Stope 11, MRLE	1.6*, 18	2	2	20 ft. "
7 " 4th east	0.7			1,200 ft. from shaft	5 Stope 10, MR	1.4	20	?	80 ft. "
8 Station, 5th level	1.4				6 Stope 13, MRW (East incline)	1.3	24	2	40 ft. "
<u>No. 29 = Y</u>					7 Stope 13, W	1.7*	8	1	
1 Shaft upcast (hot-tom), 80 ft. below 12 station	1.0	28		Drills going	8 Stope 15, No. 1, (East incline)	1.1	30	H	
2 Station, No. 12	0.8			No connections below 10th level	9 Drive, 15th level, East SR	1.6*	8	1	
3 Drive, at face, 60 ft. from shaft	2.5*			Place full of "dirt," blower not working					
4 Drive, 11th level E face	0.8			Machine stopped					
5 Drive, 11th level W face	0.8			" "					

Locality.	CO ₂ per 1,000	No. of Persons.	No. of Machines.	Remarks.
<u>No. 34 (new). (Continued.)</u>				
10 Drive, 13th level, 1'4	6	1		East of incline shaft
11 Winze, 15th level 1'1	4	H		No air on
<u>No. 35=H</u>				
1 Upcast	1'3			60 ft. from surface
2 Stope, 2nd level, 1'2	17	3		West of shaft
<u>No. 12 SR</u>				
3 Stope, 2nd level, 1'6*	23	2		East of shaft
<u>No. 11 SR</u>				
4 Stope, 2nd level, 2'2*	10	H		" "
<u>No. 14 SR</u>				
5 Stope (back)	2'0*	26	3	" "
<u>No. 8 SR</u>				
6 Stope (back)	2'2*	16	2	" "
<u>No. 15 MR</u>				
7 Level, 2nd SR	1'3	?		Just beyond shaft
8 Connection with mine 56	1'1			Noon, January
<u>No. 36=N (Meyer & Charlton)</u>				
1 Upcast, 50 ft. from surface	2'0*			750 persons underground
2 Drive, 13th level, 1'6*	4	1		6 ft. from face
<u>MRE</u>				
3 Drive, 13th level, 1'4	4	1		6 ft. " "
<u>MRW</u>				
4 Raise, 8th level, 1'6*	4	1		60 ft. up, 7 ft. from face
<u>MRLE</u>				
<u>No. 37 (new)</u>				
1 Upcast (No. 3)	60	1'4		900 underground
<u>ft. from collar</u>				
2 Drive, 9th level 1'4	5	1		6 ft. from face
<u>SRW</u>				
3 Winze, 9th level 2'0*	5	H		60 ft. deep, no air on
<u>SR (No. 7 W)</u>				
4 Winze, 9th level 1'7*	6	H		85 ft. deep
5 Raise, 9th level 1'6*	5	1		
<u>SRW</u>				
6 Raise, 10th level 1'8*	4	1		
<u>SRW</u>				
<u>No. 38=U</u>				
1 Upcast shaft	0'9			Below collar
2 " 16 E 3	1'0	50		
3 " (back) SR	1'2		2	
4 Stope, 11 ENR	1'2	20	2	
5 " (back), 6th level	1'6*	12	H	East of shaft
6 Stope, 9th level 0'8	16	2		"
<u>(No. 2)</u>				
7 Stope, 10th level 0'7	11	2		"
<u>(No. 2)</u>				
8 Stope, 11th level 0'6	19	2		"
<u>(No. 2)</u>				
9 Stope, 16 E 2	1'0			Between shafts
10 Drive, 30 yards from above	3'0*			Other side from upcast
11 Drive, same, 10 yards beyond connection	0'7			Upcast on this side
12 Raise, 16th level, east boundary	3'0*	8		Machine about to be rigged
<u>No. 39=d</u>				
1 Upcast shaft, 50 ft. from surface	1'5*			1,200 persons and candles in mine
2 Upcast shaft, low 4th level	1'4			

Locality.	CO ₂ per 1,000	No. of Persons.	No. of Machines.	Remarks.
<u>No. 39=d. (Continued.)</u>				
3 Upcast shaft, low 5th level	1'4			
4 Stope, 210 A, beyond upcast	1'3	22	H	20 ft. down
5 Stope, 212 A, beyond upcast	1'4	28	H	30 "
6 Stope, 310 B, beyond upcast	1'3	8	2	20 "
7 Stope, 514 B, beyond upcast	1'4	8	2	15 "
8 Stope, 612 B, beyond upcast	1'3	8	2	20 "
9 Drive, 4th level, beyond upcast	1'3	4	1	7 ft. from face
10 Drive, 5th level MR, beyond upcast	1'5*	4	1	9 "
There is evidently very little air available here, but it is excellently distributed; but as all are 1'3 or over, there is no margin of safety.				
<u>No. 40=V</u>				
1 Upcast shaft	1'3			40 ft. from collar
2 " "	1'3			Bottom of vertical
3 " "	1'4	70	6	Below last connected level
4 Raise, 9th level	1'4	7	1	
<u>SRE</u>				
5 Raise, 9th level	4'0*	5	?	60 ft. up: air off
<u>MRL</u>				
6 Winze, 11th level	1'7*	5	H	80 ft. down: air slightly on
<u>MRE</u>				
7 Winze, 11th level	2'5*	5	H	50 ft. down: air off
<u>SR & MRL</u>				
<u>No. 41=s</u>				
1 Shaft, main up-cast	1'1	40 ft. from collar		500 persons in mine
2 Stope, 6-1, WR	1'3	26	H	
3 " 7-3	1'1	30	H	
4 Stope, 4-7	0'7	25	H	
5 " 8-3	0'8	30	H	
6 Drive, 4th level, north	1'0	?	?	
7 Drive, 8th level, north	0'7	10	1	
<u>No. 42=m</u>				
1 Upcast shaft (No. 2)	0'8	30		40 ft. from collar
2 Stope, 8-5 west	1'1	41		At bottom
3 " 9-1 "	0'8	34		"
4 " 13-2 cast	0'5			Empty
5 " 15-2 "	0'5			"
6 Level, 9th, west	1'6*	9	1	Beyond upcast
<u>No. 43=r</u>				
1 Shaft, upcast, vertical	0'6	40 ft. from collar		50 persons in mine
2 Drive, 8th level E	1'2'	Main		400 ft. from station
3 " 9th " "	0'7	working		300 " "
<u>levels</u>				
<u>No. 44=Z</u>				
*1 Upcast shaft, below collar	0'8			No blasting for 45 hours before

* Most of upcast air escapes into neighbouring mines.

Locality.	CO ₂ per 1,000	No. of Persons.	No. of Machines.	Remarks.	Locality.	CO ₂ per 1,000	No. of Persons.	No. of Machines.	Remarks.
No. 44 = Z. (Continued.)					No. 47 = P. (Continued.)				
2 Stope, N reef, in 1·0				No blasting for 45	5 Drive, end	1·0			600ft. from cross-cut
in box-hole				hours before	6 Station, No. 8	1·4			
3 Stope, 3rd level 3·5*				Do. : Taken in box-	7 " No. 9	1·6			
east, new reef				hole, W face	8 Drive, No. 9	2·0*			400 ft. from cross-
4 Stope, 3rd level 4·5*				Do. : Taken at					cut. Machine
east, new reef				bottom, E face					just stopped, 200
5 Stope, S reef 1·3									ft. from boxhole
No. 45 = S									9 Level, end of No. 1·6*
1 Upcast shaft, 1·3				Towards end of	9 cross-cut			3	100 ft. from upcast
40 ft. from sur-				shift	No. 48 = q				
face					1 Drive, 11th level 1·4	6	1		8 ft. from face
2 Stope, 14th level, 1·1	?	?		Strong current	SRW				
12th east					2 Drive, 11th level 1·4	5	1		
3 Stope, 14th level, 1·6*	?	?		400 ft. farther in	SRE				
13th east					3 Stope, 11th level 1·2	20	2		45 ft. down east of
4 Stope, 16th level, 1·3	?	?		Strong current	MRLE				shaft
12th east					4 Stope, 11th level 1·3	24	H		50 ft. down east of
5 Stope, 18th level, 1·3	?	?		"	SR				shaft
5th east					5 Stope, 11th level 1·4	28	2		30 ft. down, east of
6 Stope, 23rd level, 1·1	?	2			MRL (GH) W				shaft
10th east					6 Stope, 11th level 1·6*	26	H		30 ft. down, east of
7 Stope, 25th level, 1·3	?	?		Last in from No. 3	MRL (J) W				shaft
10th east				shaft	No. 49 = l				
8 Drive, 23rd level, 4·0*	4	?		1,050 ft. in from	1 Drive, 3,050 ft. 0·9				1,000 ft. in from
east				shaft	level				shaft
9 Drive, 24th level, 1·3	?	1		1,300 ft. in from	2 Drive, 3,050 ft. 0·6				400 ft. back from
east				shaft	level				machine
10 Drive, 25th level, 3·5*	3	H		1,120 ft. in from	3 Drive, 3,400 ft. 1·4				500 ft. from face.
east				shaft	level				Just before blast-
11 Drive, 26th level, 5·0*	3	H		1,000 ft. in from	4 Drive, 1st off in-1·2	7			ing
east				shaft	cline 3,500 ft.				A little air on
No. 46 = B									depth
1 Upcast shaft 1·3				60 ft. from collar	5 Drive, 2nd off in-1·1	11	sever ¹		
2 Shaft bottom 2·0* 25				A little air on	cline 3,650 ft.				M
3 Stope (back), 0·6 25		H		Taken halfway up	depth				
1,100 ft. level					6 Drive, 3rd of in-1·6*	6	2		
4 Stope (back), 1·2 20				10 ft. from face, at	cline 3,900 ft.				
1,200 ft. level				top	depth				
5 Stope, No. 7 0·7		2			7 Shaft (not work-2·0*	7			60 ft. from bottom
6 Stope, 1,100 ft. 1·6*				600 ft. beyond	ing)				
level MR				shaft	No. 50 = n				
(isolated)					1 Upcast, 50ft. from 1·4				660 persons under-
7 Stope, 1,100 ft. 2·0*				700 ft. beyond	surface				ground
level SR				shaft	2 Stope, 5th level 1·4	10	H		50 ft. down and
8 Stope, 1,200 ft. 1·4				700 ft. beyond	MRL				near face
level SR				shaft	3 Stope, 15th level 1·4	28	H		40 ft. down and
9 Stope, 1,400 ft. 1·1					MRL West				near face
level SR (No. 8)					4 Drive, 17th level 1·4	5	1		6 ft. from face
10 Stope, 1,300 ft. 1·1					MRW				
level (No. 7)					5 Raise, 7th level 1·6*	5	1		60 ft. up
11 Drive, 1,400 ft. 2·0*				850 ft. from shaft	MRL West				
level MR					No. 51 = t				
12 Drive, 1,400 ft. 2·0*	6	1		10 ft. from face,	1 Stope, 14th level 1·4	19			
level, 80 ft. in				machine stopped	(1) MR				
				for an hour	2 Stope, 14th level 1·4	12	2		
				Machine just	(2) MR				
				rigged up, no air	3 Stope, 15th level 1·3	6	1		
					(1) MR				
No. 47 = P					4 Stope, 14th level 1·4	13	H		
1 Upcast 1·1					(1) SR				
2 Stope (back) 504	2·0*	26			5 Winze, 14th level 1·6*	6			20 ft. down, a little
3 Stope 1·2	6	2			MRW (1)				air on
4 Cross-cut No. 8	1·6*			90 ft. from upcast.	6 Winze, 14th level 1·4	5			15 ft. down, a little
				Only connected	MRW (2)				air on
				through stopes					

Locality.	CO ₂ per 1,000	No. of Persons.	No. of Machines.	Remarks.
No. 52 = k				
1 Upcast (also of Comet)	1.4			1,500 persons in two mines
2 Stope, 3rd level, East	1.4			750 ft. from shaft
3 Stope, 4th level, East	1.3			1,450 ft. from shaft
4 Winze, 3rd level, East	3.5*	2		100 ft. down, air off 4 hours
5 Winze, 4th level, East	2.0*	2	H	90 ft. down, air off
6 Drive, 4th level, East	1.1	6	2	100 ft. from face, 1,500 ft. in
7 Raise, 4th level, East	3.5*	3		1,100 ft. in and 40 ft. up
8 Raise, 5th level, East	4.0*	5	1	60 ft. up, machine just started
No. 53 = R				
1 Stope, 10th level, MR West	2.0*	20		50 ft. down
2 Stope, 10th level, MR West	1.8*	20		30 ft. ,,
3 Stope, 9th level, MR West	2.0*	26		45 ft. ,,
4 Stope, 9th level, MR West, 20 ft. from face	1.6*	26		70 ft. ,,
5 Stope, 10th level, SR East	2.0*	17		35 ft. ,,
6 Stope, 9th level, SR East	2.0*	12	2	50 ft. ,,
7 Stope (boundary) 9th level, SR East	1.7*	15	2	70 ft. ,,
8 Stope, 10th level, MR East	1.6*	20	6	25 ft. ,,
(This is a rather startling series)				
No. 54 = p				
1 Upcast (No. 1) 50 ft. from surface	1.4			900 underground
2 Drive, 3rd level, West	1.6*	7	2	6 ft. from face, beyond shaft
No. 55 (new)				
1 Upcast shaft	1.6*	260 ft.		250 underground from surface
2 Upcast shaft	1.8*	50 ft.		Upcast of mine from surface 48 also, with total of 900 underground
3 Stope, 6th level, MRE	1.0	17		Middle
4 Stope, 7th level, SRW	1.6*	10		20 ft. down
5 Stope, 8th level, SRW	1.6*	20		30 ,,
6 Stope, 7th level, SRW	0.8	22		50 ,,
7 Stope, 8th level, MRW	1.7*	22		40 ,,
No. 56 = W				
1 Stope, 6th level, MR East	1.0	30		Middle
2 Stope, 6th level, MR East	0.8	30		Face

Locality.	CO ₂ per 1,000	No. of Persons.	No. of Machines.	Remarks.
No. 56 = W. (Continued.)				
3 Stope, 5th level, MR East	1.8*	16		Middle
4 Stope, 5th level, MR East	3.0*	16		Back
5 Stope, 5th level, MR East	3.0*	16		Face
6 Drive, north of dyke, 5th level, MR East	2.5*	5		Face, no air on
7 Drive, same, 20 ft. back	2.5*	5		
8 Drive, same, 40 ft. back	1.3	5		Air on 3 minutes
9 Stope, 4th level, MR East	1.3	14	2	25 ft. down
10 Stope (No. 1), level, SR East	1.6*	30	6	50 ft. ,,
11 Stope, 7th level, SR West	1.8*	20	2	100 ft. ,,
12 Stope, 7th level, SR West, 20 ft. from face	1.8*	20	2	40 ft. ,,
13 Stope, 7th level, MR East	1.6*	27	H	30 ft. ,,
14 Stope, 7th level, MR West	1.4	12	2	20 ft. ,,
15 Stope, 6th level, MR West (1)	1.2	12	2	40 ft. ,,
16 Stope, 6th level, SR West (3)	1.3	20	2	40 ft. ,,
17 Stope, 6th level, MR East (6)	1.3	16	3	60 ft. ,,
18 Stope, 6th level, MRE (2-5)	1.3	20	2	60 ft. up
No. 57 (new) deep deep (single shaft bratticed)				
1 Shaft downcast half	0.5			Near bottom
2 Shaft upcast half	1.6*			20 ft. from surface
3 ,, level station	3,000 ft. 1.7*	3		In ladder way
4 Station, 3,900 ft. level	1.3	5		Temp. = 83 deg.
5 Station, 4,000 ft. level	1.2	12		Temperature = 84 deg.
6 Cross-cut, 3,900 ft. level (end)	1.7*	4		Temperature = 85 deg.
7 Crosscut, 3,900 ft. S section	1.2	9	2	10 ft. from face
8 ,, 4,000 ft. S section	2.0*	5		Junction of E. & W. drives
9 Ore-bin, 4,000 ft. station	1.7*	7		10 ft. below floor of station
10 Drive, 4,000 ft. over level W	2.5*	7	0	One hour after blasting: no air on
11 Drive, 4,000 ft. over level W	2.5*	4	0	150 ft. back
12 Drive, 4,000 ft. over level E	1.8*	9	1	No blasting since previous day
13 Drive, 4,000 ft. level, N section	1.0	5	0	Near face: one hour after blasting: air on
14 Drive, 4,000 ft. over level E drive	3.5*	several	0	20 ft. after blasting: boys tramping (!)

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Locality.	CO ₂ per 1,000	No. of Persons.	No. of Machines.	Remarks.
No. 57 (new) deep deep (single shaft bratticed). (Continued.)				
15 Drive, 3,900 ft. level E	1.2	2	0	Air blowing off strongly
16 Drive, 3,900 ft. level E 60 ft. back	1.6*	4	0	Air blowing, 6 persons near exhaust
17 Winze, 4,000 ft. over level N (bottom)	4.0*	4	0	120 ft. deep: air on 20 ft. from bottom. Temperature = 90 deg.
18 Winze, 3,900 ft. level (45 ft. over deep)	3.5*	5	1	0 hour. after blasting the cut, a little air on. Temperature = 84 deg.
19 Raise, 4,000 ft. over level North	2.5*	9	2	140 ft. up. Machines stopped 2 min. Temperature = 88 deg.
20 Raise, same, half way up	3.5*	8	2	Temperature = 87 deg.

NOTE.—Shaft ventilation greatly impeded at time of sampling by temporary "box" at bottom. Otherwise ordinary working conditions.

No. 58 = o

1 Upcast shaft, 50 ft. fm surface	1.4			900 persons and candles underground
2 Stope, SR, 14-16 levels	1.3	20	4	W of upcast, 30 ft. down
3 Stope, SR, 14-16 levels	1.4	30	6	E of upcast, 30 ft. down
4 Drive, 16th level, W of cross-cut	2.1*	7	7	8 ft. from face
5 Upcast shaft, 100 ft. from surface	0.8			800 persons underground. There are two other small upcasts
6 Stope, 16th level, SRW	3.0*	20	4	60 ft. up
7 Stope, 16th level, SRW	3.0*	20	4	50 ft. nearer East face
8 Stope, 16th level, SRE	2.2*	13	3	60 ft. down
9 Stope, 18th level, SRE	2.2*	10	H	50 ft. "
10 Stope, 14th level, SRW	2.3*	25	2	100 ft. "
11 Stope, 16-18 level (No. 1 West) (at top)	1.4	14	3	Winze at top being opened out
12 Drive, 18th level, MRLE	1.8*	5		Air off
13 Drive, 6th level, S R W (Main cross-cut)	0.6	?		8,000 cub. ft. passing
14 Drive, 8th level, MRLE	0.7	?		Strong current

Locality.	CO ₂ per 1,000	No. of Persons.	No. of Machines.	Remarks.
No. 58 = o. (Continued.)				
15 Drive, 16th level, SR (12 ft. from cross-cut)	1.3	?		Return air from lower stopes
16 Drive, 16th level, M R L (East face)	2.5*	4	1	Machine stopped
17 Drive, 16th level, M R L (West face)	3.0*	4	1	" "
18 Winze, 18th level, S R (No. 2 East)	2.5*	7		At top, no air
Following are the results in the unclassified mines:				
No. I.				
1 Upcast	1.2			near end 40 ft. from collar of shift
No. II.				
1 Stope, 11th level, MRE	1.0	20	21	25 ft. down
2 Stope, 12th level, MRLE	1.2	21	19	15 "
3 Stope, 12th level, SRE	1.2	24	24	15 "
4 Stope, 11th level, MRE	1.2	40	?	40 "
5 Stope, 11th level, MRE	1.6*	40	?	40 ft. down and 40 ft. along face.
No. IV.				
1 Stope, 4th level, SRW	2.5*	5		
2 Stope, 4th level, SRW	2.8*	6		50 ft. further west
3 Stope, 6th level, SRW	2.8*	5		100 ft. from boundary
4 Stope, 4th level, SRE	2.6*	8		60 ft. down
5 Stope, 4th level, MRE	1.6*	4		Top

I should mention that in all the new tests the method of sampling by means of small bellows, suggested by Mr. McArthur Johnston, has been adopted with satisfactory results. The chief advantage over the sucking method is, of course, that the sample can be taken farther away from the sampler's own breath.

These 430 results show only about 30 per cent. of bad samples on the average, which is chiefly due to a great improvement in the Krugersdorp district. I should mention also that quite half of them were taken not in the central portions of the mines, but beyond the two shafts—which amounts to very severe conditions of sampling.

The President: Those of us who know Dr. Moir have been anticipating for some time a very enjoyable evening on the occasion on which he would make his reply, and I am certain that we have not been disappointed. I think that considering the severe criticism he has been subjected to, his reply has been admirable, and I wish to congratulate him on his success.

THE COMPUTATION OF THE CRUSHING EFFICIENCY OF TUBE MILLS.

By S. H. PEARCE and W. A. CALDECOTT
(Members).

DISCUSSION.

Prof. R. H. Richards (*Communication to authors*): I am inclined at the start to divide the question into what we would do for the sake of a scientific investigation on the one hand, and what we would do for the sake of making a commercial rule which would serve in the daily work of the mills on the other hand.

I am inclined to think that your rule to which you refer using three sizes of screen, .024, .010 and .006 in. diam. is good, and perhaps as good as you can get for the mill man. I have drawn a curve in which the abscissae are fractions of an inch diameter of grain, and the ordinates are percentages. This is what I call a cumulative curve, by that I mean that the ordinate for the percentage of sand between .010 and .024 is measured vertically to a point *a*, and the quantity of sand between .006 and .010 is added to that and brings the curve up to the point *b*, and the material from .006 to .000 is added to the height *b* and brings it up to 100 per cent., thus completing the curve *before* the tube mill; and again the material between .010 and .024 laid off vertically up to the point *c*, and to this is added the amount between .006 and .010, and the sum is laid off vertically up to the point *d*, and the amount between .006 and .000 is added to this and laid up to 100 per cent. This completes the curve for *after* the tube mill.

Now to discuss the thing from a scientific point of view. The first thing one notices is that one knows very little about those two pulps. The larger percentage in the first case, and very much the larger percentage in the second case, are in a region that is entirely unknown, and we cannot tell how far my curve departs from the truth. Following the scientific question then, one must have the figures before one can argue, and I suggest to you that for exploring this large area that you should adopt the method pursued by me to obtain settling velocities in water, as well as the diameters of extremely fine grains of sand) by what we call the elutriation method accompanied by micrometer measurement with the microscope. I attach to this a table of such sizes and such velocities, and would say that the paper which leads up to this table, telling how it was obtained, is just now in the printer's hands and will appear in the Transactions of the American Institute of Mining Engineers, in a bulletin soon to appear.

The method in brief consists of placing a weighed quantity of the pulp in a beaker of water, starting it up and allowing it to settle 1,000 seconds, then decanting the muddy water. This stirring, settling and decanting is repeated five times, the sum of the five decantations put together and finally settled gives the product which will fail to settle 100 mm. in 1,000 seconds. That product having been completely removed from the heavier portion which is left in the beaker, it is next stirred up and is settled 840 seconds, the stirring, settling and decanting repeated five times, and a second product is obtained which contains the grains that will settle 100 mm. in 1,000 seconds but will not settle 100 mm. in 840 seconds. In like manner the seconds are diminished for product after product, going down by a scale of seconds repeated twice as often as they would be by Rittinger's scale, thus giving a very long series of settling velocities and therefore sizes, which appears to be the series, separated by an extremely small amount and covering the very field, that you are discussing in your paper, with your four points, namely, .024, .010, .006 and .000.

I think such a series as this carefully obtained from each of a few of your South African mills would give you data so accurate that the conclusions you drew from them would be of the greatest value.

I am not familiar with the derivation of Laschinger's formula, which locates the average grain at a point different from the arithmetical mean of the two apertures, and I should hesitate very much, if this formula is based on purely mathematical expressions, to adopt it unless it was proved to be true in practical operation.

This method of attack which I now suggest to you eliminates all doubt about the mean average size because those diameters are measured by micrometer, and the figures given in the table are each probably an average of as many as an hundred measures. This table of mine, unfortunately for your purpose, was not made on a stamp mill product, and the weights obtained on each size are not given. It can therefore only serve as indicating a method of attack. It seems to me that after a few such tests as this have been made on your different pulps, that you would be able to find your two points, which, I take it, are obtained from less exhaustive tests in the South African practice. Having so obtained them, you could perhaps approximate sieves to those points and proceed as you have done in the past, so far as your commercial tests are concerned.

The above is on the assumption that you prefer to use screens. If, on the other hand, you find that the elutriation process lends itself to the

daily commercial routine of mill practice, your two points can be perfectly well obtained by this method, and probably obtained more accurately than they could by sieves, and the whole question of fitting and exact size sieve hole to your process would be eliminated. The table that I here give is taken from my table of settling velocities, and it shows that the number of seconds per 100 mm. will be too small to suit your case. Your settling tube had better be 500 mm., and then the seconds for the finest decantation will be 29.95, or practically 30 seconds, the seconds needed for the next will be 16.5, and if you need the coarsest, which I do not suppose you do, it will be 8.85.

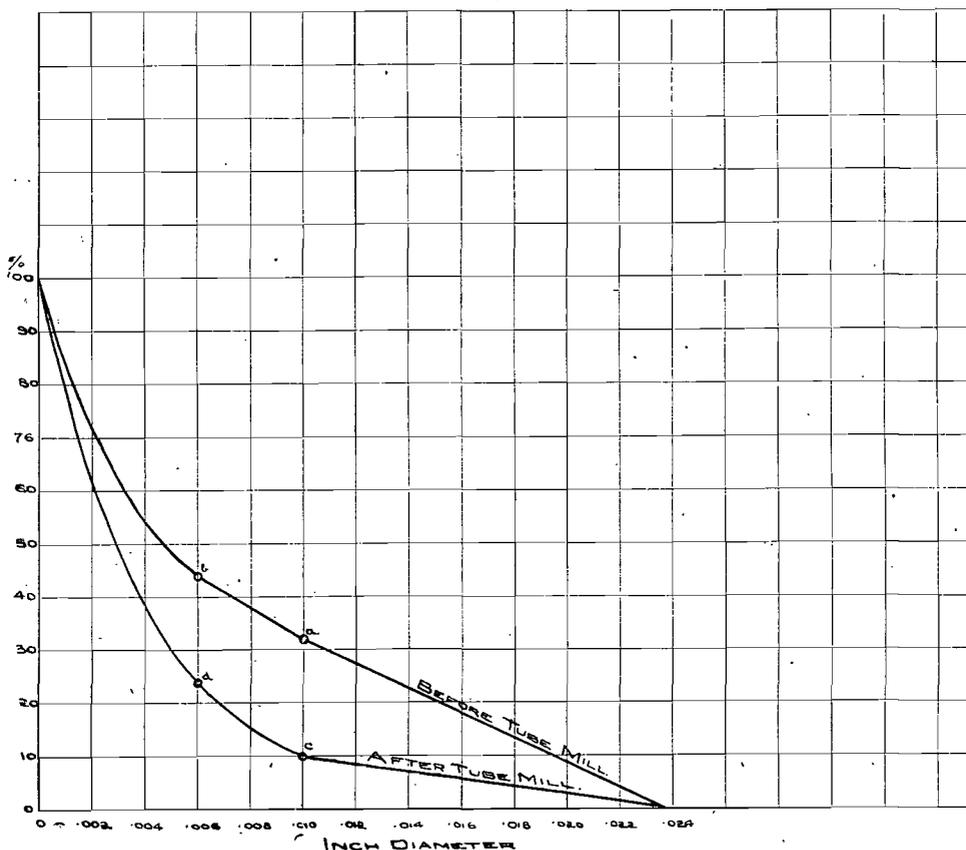
South African scale inch diameter.	Corresponding mm. diameter.	From "Velocity of Settling" mm. per-second	Seconds per 100 mm.	Seconds per 500 mm.
.024	0.61	56.45	1.77	8.85
.006	0.254	30.24	3.3	16.5
.000	0.152	16.72	5.99	29.95

The complete table of the series of times of settling for one of your mill pulps is here given, and it is based on the double Rittinger sieve

scale using a common multiplier in the geometric progression of 1.189; that is to say, any number multiplied by 1.189 gives the one above it.

Rittinger's multiplier is $\sqrt[4]{2} = 1.189$. This multiplier is $\sqrt[4]{2} = 1.189$.

In seconds for 100 mm.	In seconds for 500 mm.	In seconds for 100 mm.	In seconds for 500 mm.
1,000	5,000	37.20	186
841	4,205	31.25	156
707.5	3,537	26.35	131
595	2,975	22.15	111
500	2,500	18.58	92
421	2,105	15.625	78
354	1,770	13.13	66
297.5	1,498	11.04	55
250	1,250	9.30	46
210.5	1,052	7.8125	39
177	885	6.57	32
148.6	743	5.52	27.5
125	625	4.64	23.2
105.1	525	3.90	19.5
88.5	442	3.28	16.4
74.25	371	2.76	13.8
62.5	312	2.32	11.6
52.6	263	1.95	9.7
44.25	221		



VELOCITY OF SETTLING OF PARTICLES OF GALENA AND QUARTZ, UNDER FREE SETTLING CONDITIONS.

By Warren and Nagel—Average velocities are of 100 observations.

DIAMETERS OF GRAINS, MILLIMETERS.			CORRECTED GALENA VELOCITIES.			COLLECTED QUARTZ VELOCITIES.		
Through mm.	On mm.	Average mm.	Maximum mm. second.	Average mm. second.	Minimum mm. second.	Maximum mm. second.	Average mm. second.	Minimum mm. second.
12.85	11.01	11.93	1,201	934	647	489	393	271
11.01	9.51	10.26	1,016	865	647	489	361	252
9.51	7.79	8.65	956	810	678	489	340	231
7.79	6.85	7.32	895	729	546	489	303	227
6.85	5.97	6.41	841	680	506	416	289	185
5.97	4.95	5.46	793	631	489	362	260	172
4.95	4.21	4.58	713	558	372	345	240	153
4.21	4.12	4.17	678	538	393	345	225	154
4.12	2.83	3.53	647	513	382	307	209	154
2.83	2.82	2.83	525	450	344	256	191	127
2.82	2.05	2.43	546	420	321	252	168	119

By Barnaby and Hayden—Average velocities are of 100 observations.

2.49	2.06	2.28	555	438	278	238	166.7	89
2.06	1.63	1.85	555	370	227	227	146.6	71
1.63	1.46	1.55	416	330.5	185	192	126.6	68
1.46	1.27	1.37	384	295.1	167	173	118.4	82
1.27	1.10	1.19	332	270.1	179	156	105.6	69
1.10	.97	1.04	332	255	161	147	93.5	64
.97	.84	.91	312	247.5	147	119	84.1	43
.84	.68	.76	278	207.8	109	114	76.7	40
.68	.57	.63	263	193.2	139	89	67.2	36
.57	.45	.51	238	160.5	82	78	52.7	32
.45	.36	.41	179	126.1	75	61	41.2	27
.36	.28	.32	152	103.1	55	47	31.9	15

By Bardwell, floating test—average diameters are of 10 microscopic fields, probably 100 observations

GALENA DIAMETER MM.			Velocity mm. per second.	QUARTZ DIAMETER MM.			Velocity mm. per second.
Largest.	Average.	Smallest.		Largest.	Average.	Smallest.	
.480	.345	.138	125	.509	.369	.243	41.67
.426	.279	.119	111.1	.413	.305	.163	34.48
.409	.215	.0764	88.5	.339	.234	.144	28.57
.321	.160	.0523	74.6	.321	.199	.130	24.39
.254	.127	.0445	62.9	.298	.182	.103	20.41
.215	.106	.0296	52.6	.254	.156	.0827	17.24
.201	.0967	.0273	43.5	.252	.135	.0655	14.49
.187	.0839	.0242	37.0	.249	.126	.0803	12.05
.165	.0798	.0179	31.3	.237	.121	.0679	10.20
.184	.0714	.0157	26.3	.219	.116	.0655	8.55
.115	.0667	.0195	22.2	.235	.112	.0569	7.14
.0879	.0599	.0125	18.5	.168	.0912	.0608	6.02
.0946	.0572	.00884	15.6	.158	.0846	.0429	5.05
.0853	.0535	.00780	13.0	.147	.0800	.0414	4.26
.0827	.0484	.00832	11.0	.120	.0747	.0364	3.57
.0684	.0437	.00624	9.26	.0972	.0689	.0387	3.00
.0764	.0419	.00676	7.75	.0970	.0629	.0315	2.52
.0741	.0401	.00546	6.49	.0918	.0555	.0317	2.12
.0653	.0390	.00520	5.46	.0842	.0503	.0200	1.78
.0552	.0354	.00419	4.59	.0718	.0478	.0164	1.50
.0499	.0311	3.86	.0715	.0425	.0138	1.26
.0440	.0284	3.25	.0660	.0377	.0109	1.06
.0431	.0259	2.72	.0629	.0344	.0109	0.887
.0419	.0235	2.29	.0611	.0319	.00806	0.746

By Bardwell, floating test—average diameters are of 10 microscopic fields, probably 100 observations.
(Continued.)

GALENA DIAMETER MM.			Velocity mm. per second.	QUARTZ DIAMETER MM.			Velocity mm. per second.
Largest.	Average.	Smallest.		Largest.	Average.	Smallest.	
·0408	·0219	1·92	·0530	·0282	·00962	0·627
·0394	·0205	1·61	·0543	·0267	·00780	0·526
·0301	·0193	1·36	·0462	·0253	·00868	0·442
·0267	·0176	1·14	·0462	·0232	·00666	0·372
·0305	·0160	0·959	·0394	·0209	·00419	0·313
·0233	·0149	0·806	·0408	·0188	·00326	0·262
·0193	·0141	0·676	·0323	·0182	0·220
·0230	·0134	0·568	·0301	·0161	0·185
·0226	·0128	0·478	·0412	·0144	0·156
·0173	·0112	0·401	·0403	·0126	0·131
·0167	·00956	0·337	·0318	·00884	0·110
·0166	·00790	0·283	·0264	·00589	0·0924
·0130	·00700	0·238				
·0124	·00593	0·200				
·0104	·00493	0·168				
·0117	·00431	0·141				
·0112	·00330	0·118				
·00745	·00152	0·0995				

CYANIDE WORKS' CLEAN-UP PRACTICE.

(Read at October Meeting, 1906.)

By JAS. E. THOMAS (Member).

REPLY TO DISCUSSION.

Mr. Jas E. Thomas : In reply to Mr. H. A. White's remarks I regret that I am unable to give the actual assay values of the solutions he asks for, as I am not now on the mine on which the experiments were carried out, and have not had time since reading his remarks to look up the records of the assays. From memory, however, I can say that there was very little difference in the assays immediately after clean up, and after the solution had been standing for more than fourteen days. The fact that the solutions were reduced in value by precipitants such as zinc, iron, charcoal and copper sulphate, and not by size, goes a long way to prove that the gold was in solution and not in suspension.

I am still, of course, in favour of bisulphate of sodium as against conl. sulphuric acid, but the mine I am at present employed on has not a plant suited to the use of bisulphate of sodium. The use of the steam jet for keeping the stock solution of the bisulphate at the proper temperature in winter is necessary, as Mr. White points out, and one was fitted to the original plant of the Simmer & Jack P. M., Ltd. It also serves to agitate the solution, so that the stock solution drawn off is of the same strength all the way through.

I confess I am still in favour of washing by decantation, as no power is required to drive shafting, filter press pumps, etc., the filter cloths last longer, and practically no time is lost since this operation can be carried on while the clean up appliances are being stowed away and everything made thoroughly clean and ship-shape. I find it much better policy not to try and get through the clean up too quickly and I have as much done in daylight as possible, as there is then less risk of losing gold by careless handling and, as Mr. Wilson says, considerable losses may occur through negligence or carelessness at clean up. Mr. Wilson's remarks as to thorough washing in order to avoid trouble when smelting I heartily endorse.

I have seen Mr. E. H. Johnson's device for agitating the solution in, and carrying off the fumes from, the acid tubs, and agree with Mr. Wilson as to its effectiveness.

The remark about there being less danger of boiling over in shallow than in deep tubs should be noted. I agree with Mr. Dowling's remarks as to having a sloping bottom of wood in the washing vats if the vat, like an extractor box, is built that way, but still retain my objection to a false bottom of wood, owing to the difficulty of fastening it down properly and its liability to warp. I can endorse the remarks about a V-shaped bottom for extractor boxes, also those referring to the scheme of dividing extractor boxes in halves.

Mr. Dowling's suggestion of having a filter press through which solutions could be pumped after clean up and dressing boxes is a most

excellent one, as there is no doubt that gold is carried over the tails of the boxes at these times, and gold that has once passed the extractor boxes is seldom, if ever, completely recovered again without the use of some such appliance.

I have tried Mr. Dowling's method of fluxing the calcined slimes, and the resultant bullion was quite as good as when the slimes and fluxes were thoroughly mixed before being put into the pots.

I am glad to know from Mr. Graham's remarks that overhead crawls for transporting zinc and gold slimes are already in use and not regarded as luxuries.

Mr. Graham's remarks and figures on the values of acid solutions before and after dilution are most instructive and valuable. In plants, however, where ordinary fluxing and smelting is in vogue, washing of the slimes is necessary when gold would pass into solution unless zinc were present in excess. I am glad that Mr. Graham agrees that a storage for washes is a wise precaution, as accidents will happen, even to the best regulated filter presses, so that it is distinctly advisable to guard against the possibility of losses occurring as much as possible.

In conclusion, I beg to tender my thanks to those gentlemen who have contributed to the discussion on this paper, and have thus helped to draw attention to this very important part of our metallurgical work.

SOME ACCESSORY STAMP MILL APPLIANCES.

(Read at November Meeting, 1906.)

By G. O. SMART (Member).

DISCUSSION.

Mr. Q. C. McMillan: I think the hydraulic press for removing broken stem ends from heads is an appliance that every millman should know about, for he has only to use it once to realise the great advantage it has over the old way of blowing the ends out with dynamite or getting the blacksmith to knock them out with the steam hammer.

Blowing out with dynamite is dangerous to all concerned, and generally proves fatal to the heads, for quite a number of heads are broken or cracked. If there is the least flaw or crack, it is pretty certain that the next time dynamite is used the head will break.

Removing stem ends with the steam hammer is very expensive, besides the blacksmith's shop is not always near and handy. Some millmen

accomplish the work with a wedge or drift. This can only apply to mills using light stamps, and then it generally means a lot of hand slogging with a sledge hammer. In many cases the man using the hammer is knocked out before the stem end. With heavy stamps, say, 1,550 lb., it would be wasted time to attempt to remove the broken piece with a hammer and drift. The stem is generally let into the head about 7½ in. and in some cases is very tight. Perhaps the anvil block foundations have something to do with the stem holding so hard. I have had several that would not move with a pressure of 140 tons.

It was suggested that the heat of a blow lamp would expand the metal sufficiently to enable our press, whose limit is 140 tons, to do the work. The heating scheme was tried and proved a complete success, the application of the blow lamp for a few minutes being sufficient to enable the press to squeeze out the broken ends with half the pressure previously applied unsuccessfully. The blow lamp has also proved extremely useful in loosening cams, both of the keyed and blanton pattern.

Mr. Smart is to be congratulated on his useful paper, and millmen in this Society are indebted to him.

NOTES ON THE USE OF THE FILTER PRESS FOR CLARIFYING SOLUTIONS.

(Read at July Meeting, 1906.)

By S. J. TRUSCOTT and A. YATES
(Members).

DISCUSSION.

Mr. W. S. Mann (Costa Rica): While treating dust from a dry crushing plant in Mexico, I had considerable trouble in obtaining a clear solution for precipitation. Settling tanks gave poor results, and a sand filter was unsatisfactory, owing to the fineness of the only sand available.

To remedy this I placed common oakum, such as is used for caulking tanks, in two individual zinc boxes, allowing the solution to pass through—from the bottom, as with the other compartments filled with zinc. This gave a perfectly clear solution.

To clean, the solution was first turned into the second compartment. The solution in the first was then drained into a bucket and passed through the second box, after which the oakum was easily and quickly cleaned by kneading with the hands, and rinsing with clean water.

The meeting then closed.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

INTERNATIONAL ATOMIC WEIGHTS, (1907).—

Element.	Symbol.	At. Wt.
Aluminium	Al	27.1
Antimony	Sb	120.2
Argon	A	39.9
Arsenic	As	75.0
Barium	Ba	137.4
*Bismuth	Bi	208.0
Boron	B	11.0
Bromine	Br	79.96
Cadmium	Cd	112.4
Cesium	Cs	132.9
Calcium	Ca	40.1
Carbon	C	12.00
Cerium	Ce	140.26
Chlorine	Cl	35.45
Chromium	Cr	52.1
Cobalt	Co	59.0
Columbium	Cb	94
Copper	Cu	63.6
Erbium	Er	166
*Europium	Eu	152
Fluorine	F	19.0
Gadolinium	Gd	156
Gallium	Ga	70
Germanium	Ge	72.5
Glucinum	Gl	9.1
Gold	Au	197.2
Helium	He	4.0
Hydrogen	H	1.008
Indium	In	115
Iodine	I	126.97
Iridium	Ir	193.0
Iron	Fe	55.9
Krypton	Kr	81.8
Lanthanum	La	138.9
Lead	Pb	206.9
Lithium	Li	7.03
Magnesium	Mg	24.36
Manganese	Mn	55.0
Mercury	Hg	200.0
Molybdenum	Mo	96.0
Neodymium	Nd	143.6
Neon	Ne	20
Nickel	Ni	58.7
*Nitrogen	N	14.01
Osmium	Os	191
Oxygen	O	16.00
Palladium	Pd	106.5
Phosphorus	P	31.0
Platinum	Pt	194.8
Potassium	K	39.15
Praseodymium	Pr	140.5
Radium	Rd	225
Rhodium	Rh	103.0
Rubidium	Rb	85.5
Ruthenium	Ru	101.7
Samarium	Sa	150.3
Scandium	Sc	44.1
Selenium	Se	79.2
Silicon	Si	28.4
Silver	Ag	107.93
Sodium	Na	23.05
Strontium	Sr	87.6
Sulphur	S	32.06
*Tantalum	Ta	181
Tellurium	Te	127.6

* These represent alterations since last year.

*Terbium	Tb	159.2
Thallium	Tl	204.1
Thorium	Th	232.5
Thulium	Tm	171
Tin	Sn	119.0
Titanium	Ti	48.1
Tungsten	W	184
Uranium	U	238.5
Vanadium	V	51.2
Xenon	Xe	128
Ytterbium	Yb	173.0
Yttrium	Yt	89.0
Zinc	Zn	65.4
Zirconium	Zr	90.6

—“Proceedings of the Chemical Society,” Jan. 30, 1907, p. 7. (W. A. C.)

TITRATION OF HYDROFLUOSILICIC ACID.—“There is a difference of opinion in the literature whether hydrofluosilicic acid can be titrated as a hexabasic acid requiring 6 mol. of potassium hydroxide (Offermann's method), or as a dibasic acid requiring 2 mol. (Weise's method). Treadwell advises titration in presence of alcohol in the latter case, whilst Penfield makes use of an indirect method by treating the silicon fluoride with a 50 per cent. alcoholic solution of potassium chloride. The authors find that, inasmuch as hydrofluosilicic acid is a strong acid, the nature of the indicator used is of minor importance, but that, in presence of alkaline solutions, *i.e.*, of hydroxidion, the hydrofluosilicic acid is hydrolysed easily into silicic acid and hydrogen fluoride. It can, thus, only be titrated as a dibasic acid when removed from the further action of the hydroxyl ions, *e.g.*, by precipitation with alcohol as insoluble potassium or barium salt. On heating, the hydrolysis is much accelerated, and the acid can then be directly titrated with sodium hydroxide as a hexabasic acid according to the equation:—



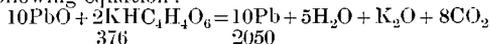
The thermal springs of the Kaiserbad in Aachen were found to contain 0.0008 gm. of fluorine per litre.”—N. SAHLBOM and F. W. HINRICHSSEN, *Ber.*, 1906, 39, 2609-2611.—*Journal of the Society of Chemical Industry*, Nov. 15, 1906, p. 1068. (A. W.)

SOME NOTES ON ASSAYING.—“The chief reducing agents in use are:—

1. Argol.
2. Charcoal or coke or coal dust.
3. Flour or sugar.

These are added to the charge in sufficient quantity to produce the proper size of lead button in the crucible assay. It often happens that an ore will contain reducing agents, chiefly sulphides, so that it becomes unnecessary to add an extraneous agent. In fact, it may contain an excess of natural reducing agent, requiring an oxidising agent to take care of this excess.

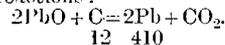
The reduction of lead by argol is expressed by the following equation:—



One gram of argol will reduce 5.45 gm. of lead from 5.85 or more grams of PbO.

The above formula for argol is that of pure bitartrate of potassium. The actual reducing power of 1 gm. of argol varies between 7 and 9.5 gm. of lead.

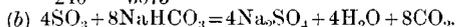
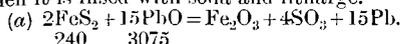
The reduction of lead by charcoal is expressed by the following reactions:—



One gram of charcoal will reduce about 34.1 gm. of lead. It will usually be found to range between 20 and 30 gm. of lead per gram of carbonaceous reducing agent used. Flour will reduce from 9 to 12 gm. of lead per gram of flour.

The amount of lead reduced per gram of the various sulphides varies according to the combination of conditions, which will be fully discussed.

Taking pyrite as an example, the following equation expresses the reaction which takes place, when it is fused with soda and litharge.



One gram of pyrite reduces 12.8 gm. of lead. The result can readily be obtained by the following charge, using a very pure pyrite:—

Pyrite	3 gm.
Sodium bicarbonate	10 "
Litharge	100 "

The result could not be obtained were the pyrite to be fused with litharge alone, as the presence of soda, a strongly alkaline base, induces the formation of sulphuric anhydride (SO₂), which combines with soda to form sodium sulphate (Na₂SO₄). This sodium sulphate will float on top of the rest of the slag and is not decomposed by the temperature usually attained.

When the oxidising action in the above charge is diminished by decreasing the litharge to below 70 gm., the iron is only partially oxidised to the ferric condition and the following equation expresses the reaction:—

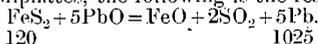


This equation will give 12 gm. of lead per gram of pyrite, whereas the other will give 12.8. The following table gives the reducing powers of the various substances as determined by the litharge—soda charge given for pyrite:—

TABLE I.

Name of Reducing Agent.	Quantity of Lead Reduced by 1 gm. of Reducing Agent
Argol	9.61
Flour	10.53
Sugar	11.78
Charcoal	26.45
Sulphur (see note to Table II.)	18.11
Pyrite	12.25
Pyrrhotite	8.71
Stibnite	7.17
Chalcocite	4.38
Sphalerite	8.16

When no soda is present to induce the formation of alkaline sulphates, the following is the reaction:—

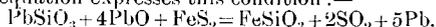


120 1025
or 1 gm. of pyrite reduces 8.5 gm. of lead.

In the assay, the foregoing conditions are modified by the presence of other substances, in the main, however, by silica. Lead oxide readily forms silicates with silica, and the mono, bi, and trisilicates are easily fusible, but those of a higher degree are difficultly fusible. When a reducing agent (argol, sulphides, etc.) is fused with a silicate of lead, or a charge containing litharge and silica, practically no lead is reduced when the silica is present in amounts to form a trisilicate or above, and little lead is reduced when the silica is present in amounts to form a mono or bisilicate. The reason for this is that the silicates of lead are not reduced by sulphides or carbonaceous reducing agents at temperatures approximating below 1,000 deg. C. Above that temperature reduction takes place more readily.

The higher the silicate degree, the more difficult is the reduction.

If, however, certain other bases such as ferric oxide (Fe₂O₃), soda (Na₂O), lime (CaO), etc., are present (as is the case with most ores) reduction of lead from the silicate occurs, with ferric oxide or soda, at a comparatively low temperature, and with lime alone, only at a high temperature. The following equation expresses this condition:—



Again, in this case 1 gm. of pyrite will reduce 8.5 gm. of lead.

While soda influences the amount of lead reduced from assay charges by the sulphides present, it has not that influence on carbonaceous reducing agents.

The following charge:—

Reducing agent	1 gm.
Litharge	45 "
Soda bicarbonate	10 "
Silica	7 "

gave results as tabulated.

TABLE II.

Name of Reducing Agent.	Quantity of Lead Reduced by 1 gm. of Reducing Agent.
Argol	9.6
Flour	10.92
Sugar	11.74
Charcoal	26.08
Pyrite	9.30
Sulphur	18.11

Pyrite, in this table, shows a reduction of 9.30 gm of lead per gram, a figure to be expected when its sulphur goes off partly as SO₂ and partly as SO₃. If the soda in the above charge is increased, the lead button will approach the maximum reducible by pyrite.

Oxidation of impurities in ores is frequently necessary in order to obtain good results in the assay. When ores contain an excess of sulphides, arsenides, etc. (by an excess is meant a quantity above that which will give the required size of lead button), an oxidising agent is required to oxidise this excess, enabling it to be volatilised or slagged.

Oxidation of impurities is accomplished in one of two ways:—

1. By the addition of potassium nitrate (KNO₃) to the charge (or other oxidising agents)
2. By roasting the ore, thus using the oxygen of the air for the oxidation of impurities.

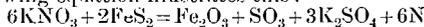
When nitre is added to an assay it reacts with the most easily oxidisable compound in the charge, which is usually the reducing agent: that is, the sulphide or the argol, charcoal, etc., when the temperature reaches the proper point for this reaction to take place.

The oxidising power of nitre is usually stated in terms of lead. Thus, 1 gm. of nitre oxidises 4.0 gm. of lead to litharge, according to the following equation:—



While this is a convenient way of stating the result, what really happens is that the nitre reacts with the reducing agent before the reducing agent reacts with the litharge to produce lead.

When looked at in this way it becomes evident that the oxidising power of nitre will vary with the kind of reducing agent in the charge. The following equation illustrates this:



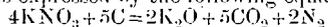
606 parts of nitre act on 240 parts of pyrite; 1 gm. of nitre oxidises 0.39 gm. of pyrite.

In a soda litharge charge, which is represented in the above conditions, 1 gm. of pyrite reduces 12.22

gm. of lead. Therefore 1 gm. of nitre in this case will oxidise 12.22 times 0.39, or 4.76 gm. of lead.

If now an assay charge is considered in which considerable silica is present, thus absorbing the greater part of the alkaline base as silicate, not leaving an appreciable amount to stimulate the formation of sulphate, we have this condition:—1 gm. of pyrite reduces 9 gm. of lead. Therefore, 1 gm. of nitre in this case will oxidise (seemingly) $4/3 \times 4.76$, or 6.34 gm. of lead. Thus, while seemingly the oxidising power of nitre varies, it is in reality the reducing agent which varies in power according to the nature of the charge: that is, the amount of alkaline base available for the formation of sulphates.

The oxidising power of nitre as compared to charcoal is expressed by the following equation:—



404 parts of nitre act on 60 parts of C. One gram of nitre oxidises 0.15 gm. of charcoal. Therefore, taking pure charcoal, as reducing 34 gm. of lead, 1 gm. of nitre will oxidise 0.15 times 34, or 5.1 gm. of lead. If now a charcoal is used which is impure, so that its reducing power is decreased to 26.0 gm. of lead, then it contains $\frac{34}{26.0}$, or 0.765 gm. of carbon. Then if 1 gm. of nitre has been added to the charge, the available carbon for reduction will be $0.765 - 0.15 = 0.615$ gm., which will reduce 20.90 gm. of lead. The oxidising power of nitre expressed in lead is $26.0 - 20.90 = 5.10$ gm.—CHAS. H. FULTON.—*Western Chemist and Metallurgist*, Nov., 1906, p. 169. (H. A. W.)

[EDITORIAL COMMENT.—It seems a pity that sodium bicarbonate should still be recommended in place of the carbonate, as its use has so frequently been condemned by members of this Society.]

METALLURGY.

TUBE-MILL LINING.—“Writing in the *Mining and Scientific Press*, the author says, I have read with pleasure the article on ‘Tube-Mill Lining,’ in which the Barry honeycomb is described and illustrated.

There is no doubt that this patent is a great improvement on the silix lining which the Waihi Co. imported with their tube-mills. Anyone who has had experience with the silix lining knows how expensive and troublesome it is when used for wet grinding, and consequently will welcome any improvement in the matter of initial cost and maintenance. The ordinary iron or steel liner, with smooth inner surface, is also unsatisfactory, for, owing to the stones sliding and thereby becoming flat, much of their efficiency for grinding is lost.

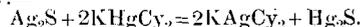
Having run tube-mills for some considerable time, grinding the sandy portion of the pulp from stamps, crushing hard ore through all grades of wire-screen from 30 mesh down to 10 mesh, and having had trouble, owing to the use of smooth iron liners, I set to work and brought out an improved iron liner, which consists of segments of a special hard mixture of iron, costing 14s. per cwt. (say, 3 cents per lb.), when manufactured into liners. These liners are 4 ft. long and $1\frac{1}{2}$ in. thick, and are of such a width that 12 form a circle of 4 ft. diameter, which is the diameter of the tube-mill. Instead of having holes cast for bolts, they have only two half holes on each long edge. Over the junction of each pair, a cast iron bar $2\frac{1}{2}$ in. sq. is placed and bolted by two square-headed bolts through the lining and outer shell of the tube-mill. Thus there are 12 cast iron bars which run longitudinally through the mill, and the effect is

to form a casing of flint stones which does not slip and which protects the iron lining.

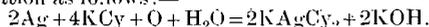
Such liners have been used for some time in the mill of the Komato Reefs Co., New Zealand, and have lasted for 18 months before being renewed. Without the square bars, the smooth iron liners only last six to eight months.

The cost of liners and bars for a 4 by 16 ft. tube-mill is £84, and two men can fix them all in position in three shifts of eight hours each; the total costs with bolts would be, say, £90. The amount of sand passing through the tube-mill is 70 tons per day of the material discharged through the 10 mesh screen of the stamp battery, and the finished product has the fineness that would be secured by the use of a 35 mesh screen on the battery. Thus the cost of lining is about $\frac{1}{5}$ d. or 1.4 cents per ton of sand ground.”—S. D. MCMIKEN.—*Mining and Scientific Press*, Nov. 3, 1906, p. 534. (K. L. G.)

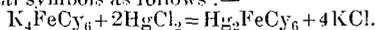
PROCESS OF SILVER EXTRACTION.—“The present invention consists of an improved process for extracting silver from its ores, and particularly from ores in which the silver is present in combination with sulphur, sulphur being present in almost all silver ores. The improved process is of very general application. Briefly outlined the process consists of treating the ores after such preliminary sizing as may be desired with a solution containing a mercurous potassium cyanide. This salt, the characteristics of which are described in serial No. 270,300, filed July 18, 1905, of which the present case is a division, has a selective affinity for silver in combination with sulphur. This selective affinity has been demonstrated experimentally a great many times. The formula of the salt is KHgCy_2 . It forms with silver when in combination with sulphur a double salt, the cyanide of silver and potassium, which is freely soluble in water, and also the sulphide of mercury, which is substantially insoluble in water, and passes off with the tailings. The reaction which takes place is:—



The solvent employed in the improved process does not attack gold in any of the forms in which it appears to exist in ore, and although it forms the same salt of silver and potassium in solution as cyanide of potassium when the latter dissolves silver, it does not do so in the same way, since potassium cyanide dissolves silver by the well-known Elsner's equation as follows:—



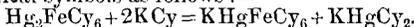
The same equation applies to the solution of gold, gold replacing silver in it. Free oxygen in solution is necessary to dissolve silver or gold and form the double salt of potassium and silver or gold with cyanogen. The mercurous potassic cyanide may be produced in any desired way, since the particular method of its production does not seem to affect the results. For example, add mercurous chloride (Hg_2Cl_2) to a solution of potassium cyanide (KCy). Another method which may be used is to add mercuric chloride (corrosive sublimate, HgCl_2) to the ordinary mill cyanide solutions containing ferro-cyanide of potassium, in which case the reaction may be stated verbally thus:—Ferrocyanide of potassium + mercuric chloride = ferrocyanide of mercury + chloride of potassium. This reaction may be expressed in chemical symbols as follows:—



This is succeeded by the following reaction:—Ferrocyanide of mercury + cyanide of potassium = ferri-cyanide of mercury and potassium + mercurous

* See this *Journal*, p. 127, vol. vii., Oct., 1906.

potassic cyanide. This reaction may be expressed in chemical symbols as follows:—



The most suitable strength of the solution is to some extent dependent upon the silver content of the ore and the form in which the metal is present. Good results have been obtained with solutions varying from .06 to .50 per cent. The metal may be separated from the solution in any of the approved methods commonly employed in cyanide processes, such as precipitation by zinc dust or shavings or electrolysis. The solution should be kept alkaline, and this may be effected by the addition of any suitable alkali, such as caustic potash or soda, although in most cases lime will be found to be not only the cheapest but the best alkali.—F. J. HOBSON, U.S. Patent, No. 828,287, Aug. 7, 1906.—*Mines and Minerals*, Dec., 1906, p. 237. (W. A. C.)

[ED. COMM.—All the equations have been corrected.]

ORE TREATMENT AT THE COMBINATION MINE, GOLDFIELD, NEVADA.—"The author in an illustrated paper deals at length with ore treatment at the above mine with particular reference to the slimes treatment of the oxidised ore. The slimes treatment is carried on by means of the Butters-Cassel filter. The slime is delivered to the centre of conical-bottom settlers two on each side of the mill, each being alternately allowed to fill and overflow with 12 hours for settling. The surplus water is carried off by a pipe decanter, the slime being left with about 50 per cent. moisture. A sufficient amount of strong cyanide is added to the charge of slime so as to make a solution of from .15 to .2 per cent. cyanide, the object being to obtain a consistency of the pulp of 3 parts solution to 1 part of slime. The pulp is then transferred to an agitator with a steep conical bottom, by means of a centrifugal pump, which pump also serves for agitation. A supplementary agitation being given by a mechanical stirrer which revolves slowly. The pulp is agitated from 12 to 18 hours, and is then discharged into a vat of large capacity provided with a mechanical stirrer. From this vat it is drawn as required for filtration in the Butters-Cassel filter plant. Before the introduction of the latter process, the slime was filtered in a filter press of American construction, which gave considerable trouble. The cost of filtering, exclusive of power, when the filter press was used amounted to 96 cents per ton of slime. The Butters-Cassel filter arrangement in the plant under consideration consists of 28 frames 5 ft. x 10 ft., set 4½ in. apart in a box 10 ft. square which has a pointed bottom inclined at an angle of 50 deg. The slime pulp is introduced at the point of the box, a vacuum of 22 in. of mercury being maintained for about 20 minutes. During that time a cake ¾ to 1 in. thick is deposited at each side of the frame. The surplus pulp is then drawn off into the slime reservoir, and a weak solution of cyanide introduced, which serves as a wash. When the cakes are thoroughly washed, the weak solution is drawn off into the weak solution vat, and wash water is then introduced into the box until the filter frames are completely submerged. This water serves the purpose of assisting in removing the cakes, more water being also introduced into the interior of the frames, under a low head. The action of this water causes the cakes of slime to drop off into the pointed bottom of the filter box, and they are then removed by sluicing. About 3½ hours are required for the operation, and about 9 tons of dry slimes are treated in one charge, one man at each shift is required for the operation. The principal power required is for

pumping the pulp and the various solutions in and out of the filter box and for the operation of the vacuum pump. The gold bearing solution is charged from the vacuum pump. Then raised about 30 ft. and forced through the discarded filter press in order to clarify it. The whole operation consumes about 10 h.p. The cost of filtering with the Butters-Cassel filter system is stated to have been reduced to 26 cents. It is stated that the recovery in the mill and the cyanide plant has attained a maximum of 93 per cent. for several consecutive months, or has averaged over 90 per cent.—F. L. BOSQUI, *Mining and Scientific Press*, Oct. 16, 1906.—*Mines and Minerals*, Dec., 1906, p. 236. (W. A. C.)

COMPARATIVE TEST BETWEEN COKE AND CRUDE OIL FOR MELTING ZINC GOLD SLIMES.—"Two experiments were made at the Butters Salvador mines, in Salvador, to determine the relative efficiency of coke and crude oil in melting precipitate from the cyanide plant; the first was on the ordinary San Sebastian clean-up from the acid refining box, and the other was on silver precipitate from Divisadero. The costs are calculated on a basis of coke at \$0.0208 (gold) per lb., and oil at \$0.252 (gold) per gallon laid down at San Sebastian; white labour at \$3 per shift of 10 hours; and compressed air at 1 cent per hour.

FIRST TEST.

OIL.		Sept. 11, 1906.
Precipitate	...	2,448 oz. Troy
Oil	...	15.2 gallons
Time	...	7½ hours
Cost of fuel per oz. of precipitate	...	\$0.00156
Cost of fuel per 1,000 oz. of precipitate	...	1.56
Cost of white labour per 1,000 oz. of precipitate	...	0.91
Cost of air per 1,000 oz. of precipitate	...	0.03
Total		\$2.50
Balance in favour of oil per 1,000 oz. of precipitate	...	1.06
Total		\$3.56

COKE.

COKE.		Sept. 13, 1906.
Precipitate	...	1,991 oz. Troy
Coke	...	221 lb.
Time	...	8½ hours
Cost of fuel per oz. of precipitate	...	\$0.00231
Cost of fuel per 1,000 oz. of precipitate	...	2.31
Cost of white labour per 1,000 oz. of precipitate	...	1.25
Total		\$3.56

SECOND TEST.

OIL.		Sept. 20, 1906.
Precipitate	...	5,664 oz. Troy
Oil	...	21 gallons
Time	...	10½ hours
Cost of fuel per oz. of precipitate	...	\$0.00093
Cost of fuel per 1,000 oz. of precipitate	...	0.93
Cost of white labour per 1,000 oz. of precipitate	...	0.55
Cost of air per 1,000 oz. of precipitate	...	0.02
Total		\$1.50
Balance in favour of oil per 1,000 oz. of precipitate	...	1.03
Total		\$2.53

COKE.

		Sept. 21, 1906.
Precipitate	3,524 oz. Troy
Coke	283 lb.
Time	10½ hours
Cost of fuel per oz. of precipitate	\$0.00167
Cost of fuel per 1,000 oz. of precipitate	1.67
Cost of white labour per 1,000 oz. of precipitate	0.86
Total		\$2.53

Thus, in both these tests, on entirely different classes of material, we get a uniform result, which shows strongly in favour of the crude-oil method of melting, at least as far as the Republic of Salvador is concerned. There are other advantages connected with the use of oil which do not appear above, but which should not be overlooked. Firstly, the elimination of a bye-product in the shape of coke ashes from the wind furnace; and secondly, a great saving in labour and personal discomfort to the man who does the melting, which is a point of some importance in a tropical climate."—E. M. HAMILTON.—*Mining and Scientific Press*, Nov. 24, 1906, p. 624. (W. A. C.)

TUBE MILLING IN KOREA.—"A description of the experimental plant at the Oriental Consolidated Mining Co., Chittabaltic, Korea, where regrinding of concentrates is practised in a tube mill with cyanide solution. The plant consists of one 2½ ft. by 12½ ft. tube mill, two mechanical agitators with plough shoes, 8 ft. by 6 ft. diameter, three filter and settling boxes 4 ft. by 5 ft. by 5 ft., 48 separate-compartment zinc boxes, two sumps and one vat for stock solution. The concentrate is put into the hopper with cyanide solution and is nearly all slimed in the tube mill, a good percentage of the gold being extracted. A spitzkasten, 3 in. wide by 1½ ft. long by 1½ ft. deep, supplied with clear water from the bottom, discharges coarse concentrate and clear water from the bottom, cyanide solution and slime passing over the top. The coarse concentrate is reground. The slime concentrate and cyanide solution go to an agitator, to be agitated for about 15 hours. The miller in the agitator is then raised, the machine stopped and the concentrate allowed to settle. The clear cyanide solution is decanted into the filter boxes and from there run to the zinc boxes for precipitation. The strong precipitated solution from the sump is then pumped up to the agitator as a wash, the miller being allowed to agitate the concentrates again for a few minutes. This treatment is followed by two successive weak cyanide washes and finally a water wash. The charge of about 5 tons of clean concentrate is then discharged through a discharge hole, the miller having been set in motion and lowered. By feeding the concentrate and the cyanide solution with a consistency of one to one through the tube mill, the finest grinding and the best results are obtained. While one agitator is being filled, the other is decanting and getting ready to be discharged. The strength of the cyanide solution used in the tube mill and agitators is .43 per cent., while the wash solution stands at .1 per cent. No cyanide is added to the weak solution. About 2 lb. of lime per ton is added with the concentrate as it is fed into the mill. The extraction at present with the tube mill plant described above is 93 per cent. on clean concentrate."—DRUCKER, *Mining and Scientific Press*, Sept. 22, 1906.—*Mines and Minerals*, Dec., 1906, p. 236. (W. A. C.)

MINING.

HEALTH IN THE MINES ON THE GOLD COAST.—"At a meeting of the Committee of the African Trade Section of the Liverpool Chamber of Commerce, held on December 10, Sir Alfred Jones, K.C.M.G., President of the Chamber and Chairman of the African Trade Section, in the chair.

Dr. R. E. McConnell stated that Sir Alfred Jones had asked him to say a few words with regard to his visit to the West Coast, and that he had rather hastily put together a few words. He stated that he went out on one of the expeditions sent by the Liverpool School of Tropical Medicine, Col. Giles being the senior member of the expedition. Unfortunately, after about a month Col. Giles was invalided home, and Dr. McConnell thought his wisest course would be to visit some of the mines where so many Europeans are employed. This he did, and he has latterly been some fifteen months with an important mine. In the considerable number of cases that came under his notice he was glad to be able to record no deaths from sickness. This may be largely attributed, he thought, to an improved class of men who now obtain employment on the West Coast, and to the greater care Europeans have learned to take of themselves, thanks to the knowledge that is being disseminated by the Schools of Tropical Medicine. Also, to the fact that mining companies, as a rule, have learnt that it is wise to spare no expense in supplying the requisite accessories for prevention and treatment of disease. He said that, naturally, blackwater fever interested him greatly, and he had observed six cases, in one of which a temperature of 108.6 was registered under the arm. All of these, under careful nursing, recovered, and he thought that in places where it could be taken early and treated carefully it is losing much of its terrors in the popular mind. This is a disease which presents many curious problems—it is held by many to be precipitated by quinine, but there are many facts which would seem to substantiate this; but in no case which he saw could he definitely attribute the onset of blackwater fever to this drug. The case for those who oppose this theory is upheld by the fact that of some sixteen Americans who went out to exploit timber, two who owing to their religion refused to take any drugs whatsoever, contracted the blackwater fever in some six weeks after their arrival, although the others did not so suffer. Again, some said that malarial fever is expelled from the system for some time after an attack of blackwater fever—in the cases which he saw the contrary was the case in almost every instance. Some four or five days after the disappearance of haemoglobinuria a definite attack of malaria supervened, as shown by the presence of the malarial parasite and the reaction of the fever to quinine. There is a tremendous field for research in this disease of blackwater fever; and he was glad to hear from the hon. secretary of the Liverpool School of Tropical Medicine that an expedition of the School is being organised to inquire into the cause, course, and treatment of this disease, which at present is the bête-noir of West, Central, and East Africa. He was glad during his residence at the mines to note how few cases of sickness one encounters which can be attributed to the water supplies—whether rain, river, or spring. Where he was latterly stationed the rivers were the sole supply, but by careful boiling and filtering he did not have a single case of sickness in which the water could be ascribed in any way as the cause. This again, shows the great care the men take to follow out any advice given by the

medical officers. The old type of Coaster, who knew everything about African illnesses or held that he did, is disappearing, and the men are usually very tractable, especially in the matter of taking quinine and in the careful use of the mosquito net. He wished to say that everywhere he found the work of the Liverpool School of Tropical Medicine thoroughly appreciated and a marked desire to help, in every way possible, any members of their expeditions; indeed, to help any medical officer who might wish to spend his spare time in elucidating any problem. They were treated with the greatest courtesy and given every facility for carrying out their work by the Government officials of the Gold Coast, and the Liverpool School of Tropical Medicine owed them many thanks. In conclusion, he hoped, the expeditions about to be sent out by the Liverpool School of Tropical Medicine will be able to find some means to exclude the spread of sleeping sickness from the West Coast generally and will make some progress in elucidating the intricacies of blackwater fever.

Sir Alfred Jones thanked Dr. McConnell on behalf of the Section for his interesting remarks, and said that he would never rest until the unhealthy parts of the tropics should be made as healthy as science and money expenditure could make them.—*London Mining Journal*, Dec. 29, 1906, p. 809. (J. Y.)

Reviews.

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

GEOLOGY. By T. C. CHAMBERLAIN and R. D. SALISBURY. Three Volumes. Copiously illustrated. Pp. 2,027. Price \$4 per Volume. (New York City: Henry Holt & Co.)

"This extensive and monumental work on geology will probably occupy the place held for so many years by Dana's celebrated volume. As large as it is, however, it cannot be expected that it contains all of the known geology of the present time. The enormous amount of material available in the reports of the United States and the several State Geological Surveys, to say nothing of the private publications and the work of foreign geologists, renders the compilation of an exhaustive book a herculean task, particularly if the authors attempt to digest present geological literature and to refer even by title only to many comprehensive treatises upon all phases of geology which are now extant. In this connection, the present work improves as it progresses. Vol. I. contains comparatively few references to original sources, and in this way leaves much to be desired by the student who wishes to go more deeply into the subject. Vol. II. is much better in this respect, and Vol. III. still better. The chapters deserving special mention are those on rock formation, the theories of the formation of the earth, glacial geology, and on the human or present period.

The authors have adopted the later designation of the Mississippi and Pennsylvania periods, instead of the older Upper and Lower Carboniferous. Very little that is new has been given on the subject of the geology of coal. The last chapter in the book is a very interesting one, tending to connect the point where geology is ordinarily supposed to stop and the

political history to begin. The illustrations are profuse and excellent, particularly the half tones. Some of the line drawings are not up to the standard of the half tones. The whole book is written in a most readable way, and the authors have avoided the snag upon which so many of our recent geologists have fallen down and have not coined new terms and phrases to express supposed infinitesimal differences. As a whole, the book is clearly written with an absence of pedantic language."—*Mines and Minerals*, Jan., 1907, p. 260. (W. A. C.)

A TREATISE ON PETROLEUM. By Sir BOVERTON REDWOOD, D.Sc., F.R.S.E., Assoc. Inst. C.E., F.I.C. Second edition. Two volumes. 45s. (London: Charles Griffin & Co., Limited.)

"Great interest has been attached to the production of the second edition of "Petroleum" and its products, by Sir Boverton Redwood, D.Sc., who in his first edition had the collaboration of Mr. G. T. Holloway, A.R.C.S. The author's name in connection with the ever-increasing commodity, petroleum, carries a guarantee of technical efficiency that is of such a high order that it might be described as unique. No undertaking that is connected with petroleum can be said to have received the seal of finality unless Sir Boverton Redwood has given his pronouncement; and it may be mentioned that his opinion is not only highly valued in the British Empire, but is equally appreciated in the United States, Russia, and indeed, all oil-producing parts of the world. The book as it is presented is necessarily better adapted for those who have a more or less intimate knowledge of the subject; but it is distinctly not confined solely to such as are technically equipped, but contains a sufficiency of interesting, and at the same time easily absorbed, information that would prove of great interest to such readers whose walks in life might never bring them in contact with oil in its crude state, but who are desirous of at least gaining an insight into the history and application of this universally demanded substance, together with its numerous products which enter so largely into the commercial economy of our times. The ever-increasing demand for petroleum has produced great changes in the methods of treating it, and its technical and commercial evolution during the past ten years has been clearly set forth in this present much enlarged edition, which brings our present knowledge of the subject practically up to date. From the point of view of the actual extraction of the oil from its source, the author gives us an idea of how much change has taken place in the methods of work, whilst he deals at length with the effective and comparatively new system of rotary drilling, so well adapted for soft formations by reason of its greater rapidity in work, and having the advantage of being less complicated to erect as compared to the well-known Staudard Rig, although possessing the disadvantage of not sufficiently indicating changes of formation, which is so important a factor in drilling economy of to-day, except in those fields having a known depth for their oil stratum. A description is given of the well-known and extremely efficient Diesel engine, whilst a complete amount of attention has been given to such important factors as storage, testing, and distillation, the chapters on which have been revised, with additions of extra plates. The transport of petroleum has been dealt with fully, and Appendix B—revised and extended by the author's brother, Mr. R. Redwood—gives a full list of the tank

steamers in commission, which from the point of view of the growing commercial importance of this commodity is sufficiently significant as to need no comment. There is a very interesting chapter on the geology and the geographical distribution of oil, whilst a very complete bibliography has been included which should be invaluable as a reference. In short, the book, which has been well got up in two volumes by the publishers, Messrs. Charles Griffin and Co., Limited, is up to the highest expectations, and the author has presented a work which should be capable of universal appreciation, and which will serve as a further proof of that thoroughness which has so characterised the career of Sir Boverton Redwood."—*The Mining Journal* (London), January 7, 1907, p. 16. (W. A. C.)

Selected Transvaal Patent Applications.

RELATING TO CHEMISTRY, MÉTALLURGY 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.) 40/07. C. F. Schmitt. Improvements in mortar boxes for stamp mills. 28.1.07.

(P.) 41/07. W. E. Kimber (1), J. H. Stuthridge (2). Improvements in means for connecting flexible pipes to couplings and the like. 28.1.07.

(C.) 42/07. R. Beresford. Improvements in wheel rims for inflated tyres. 29.1.07.

(P.) 44/07. R. Connolly (1), T. V. W. Swanton (2). Salt extractor. 29.1.07.

(C.) 45/07. P. Velter. Improvements in cupels. 30.1.07.

(C.) 46/07. M. D. Rochford. Excavator or elevator. 1.2.07.

(C.) 47/07. H. J. Hugnet. Amalgamating process applicable to all gold and silver ores. 1.2.07.

(P.) 48/07. J. Moir (1), G. S. Persse (2). A new or improved antacid dressing for wire ropes. 4.2.07.

(C.) 49/07. A. J. Bant (1), H. Gullett (2). Improvements relating to distributing valves for percussive rock drilling machines and other reciprocating engines. 4.2.07.

(P.) 50/07. E. Lloyd. A renewable bit for jumper drills applicable to machine drills or hand drills. 4.2.07.

(C.) 51/07. W. A. Merralls. Process of cyaniding and apparatus therefor. 8.2.07.

(C.) 52/07. La Societe Anonyme (1) Westinghouse (2), M. Le Blanc (3). Improvements in or relating to jet or surface condensers. 8.2.07.

(P.) 53/07. J. Ebbs. Apparatus for transporting material particularly in stopes. 8.2.07.

(P.) 55/07. C. H. Hamilton. Improvements in grinding mills. 8.2.07.

(P.) 57/07. C. H. Erskine. Improvements relating to rock drilling machines and means for manipulating the same. 9.2.07.

(P.) 58/07. L. Hampel. Improved process for the treatment of auriferous ores and apparatus therefor. 9.2.07.

(P.) 59/07. G. T. Cannon. Improvements in safety mechanism applicable to mine cages and skips and hoists, elevators and the like. 11.2.07.

(P.) 60/07. F. V. W. Swanton (1), B. Price (2). Improved detachable patent crown bit for percussive rock drills. 11.2.07.

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