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Vol. XI. MARCH, 1911. No. 9.

Proceedings
AT
Ordinary General Meeting, March 18, 1911.

The Ordinary General Meeting of the Society was held in the Lecture Theatre of the South African School of Mines, on Saturday evening, March 18th, Dr. James Moir (President) in the chair. There were also present :—

62 Members: Messrs. W. R. Dowling, R. Allen, K. L. Graham, Tom Johnson, E. J. Laschinger, J. E. Thomas, A. Whitby, H. A. White, J. A. Wilkinson, James Littlejohn, A. F. Crosse, E. H. Johnson, A. McA. Johnston (Members of Council), W. J. Abel. W. Beaver, B. V. Blundun, J. Chilton, F. W. Cindel, W. M. Coulter, Michael Dodd, E. A. Furner, A. Gullachsen, R. Gascoyne, James Gray, A. J. Herald, J. I'Ons, A. J. Johnson, J. H. Johnson, R. A. Lawrie, G. A. Lawson, Henry Lea, J. Lea, L. Marks, T. G. Martyn, G. Melvill, H. H. Morrell, P. T. Morrisby, W. C. Mossop, S. Newton, W. J. North, E. A. Osterloh, J. C. Phillips, H. W. Pidsley, J. B. Polglase, J. F. Pyles, W. A. Quince, D. Robertson, G. A. Robertson, W. H. Roe, F. W. Rush, A. Salkinson, A. Schwarz, S. Shlom, S. H. Steels, Ralph Stokes, J. A. Taylor, A. Thomas, J. T. Triggs, C. F. Webb, E. M. Weston, J. Whitehouse, and A. Wilkinson.

26 Associates and Students: Messrs. C. F. Bayly, C. J. Crocker, J. Cronin, C. A. Damant, D. C. Edington, J. Gibson, J. S. Grace, W. J. R. Hunter, A. King, G. F. Mathews, F. J. Pooler, H. B. Powter, A. G. Rusden, H. Rusden, R. Sawyer, P. Scatterty, H. Stadler, E. H. Tamplin, A. M. Thomas, I. Tom, P. A. Tucker, H. Ward, F. Wartenweiler, J. C. Webster, P. Wilson, and L. A. Womble.

18 Visitors, and Fred. Rowland, Secretary.

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

NEW MEMBERS.

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

CASTLE, LEWIS PERCY DE, c/o Don Proprietary Mines, Ltd., Gwelo, Rhodesia. Mining Engineer.

GULLACHSEN, BERENT CONRAD, Geldenhuis Deep, Ltd., P. O. Box 5, Cleveland. Mining Engineer.

HARRISON, MARK JOHN, Knight Central G. M. Co., P. O. Box 91, Germiston. Amalgamator.

MARTYN, THOMAS GRAHAM, New Primrose G. M. Co., P. O. Box 193, Germiston. Metallurgical Chemist.

NEWBERY, J. W., The Tarquah Mining and Exploration Co., Tarquah, Gold Coast, West Africa. Mine Manager.

POLGLASE, J. B., Durban Roodepoort Deep, Ltd., P. O. Box 110, Roodepoort. Mine Overseer.

ROUILLARD, ANTHONY PHILIPPE, Durban Roodepoort Deep, Ltd., P. O. Box 110, Roodepoort. Mine Manager.

THOMPSON, GEORGE ROBERT, B.Sc., A.R.S.M., S. A. School of Mines, P. O. Box 1176, Johannesburg. Professor of Mining.

WILSON, JAMES CARRIE, Weynek Tin Co., Ltd., P. O. Rooiberg, *via* Warmbaths. Mine Manager.

WINTERTON, ARTHUR HENRY, East Rand Proprietary Mines, Ltd., P. O. Box 94, East Rand. Mill Foreman.

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

As Associates.—

BELL, HERBERT CHARLES FITZWILLIAM, P. O. Box 21, Randfontein. Surveyor. (*Transfer from Student Roll*).

COWLIN, HUGH HOUGHTON, A.R.S.M., Simmer Deep, Ltd., P. O. Box 178, Germiston. Metallurgical Engineer.

EDINGTON, DAVID CAMERON, B.Sc.(Edin.), Robinson Deep G. M. Co., Ltd., P. O. Box 1488, Johannesburg. Mining Engineer.

PORTER, ROBERT ALLAN, Simmer Deep, Ltd., P. O. Box 178, Germiston. Reduction Works Sampler.

As Students.—

ABREY, THOMAS SHAW HILLIER, Geldenhuis Deep, Ltd., North Section, P. O. Box 5, Cleveland. Cyanide Learner.

DAVIS, WALTER HERBERT, Geldenhuis Deep, North Section, P. O. Box 5, Cleveland. Learner.

SPENCE, ERIC HUDSON, City Deep, Ltd., P. O. Box 1411, Johannesburg. Learner.

GENERAL BUSINESS.

Mr. Andrew F. Crosse (*Past-President*): I have a suggestion which I should like to bring up before you this evening. It would be a very good scheme for our Society to arrange for either a temporary or perhaps even a permanent exhibition of the most modern and up-to-date appliances—chemical, physical and mechanical—which are required for assaying, sampling, ore crushing, testing various products, sieving, etc. The list is so long that time does not allow of my mentioning any more this evening—there are so many recent improvements and such varied new apparatus which could be exhibited, in some central position in this town, which would be very interesting to our members and valuable to the mining industry. For instance, the other day, during our visit to the Crown Mines, Mr. Stadler, who has done so much work in connection with grading analysis, showed me a very ingenious invention of his own, for passing a sample of crushed ore through a series of sieves. All such improvements as this should be collected under one roof and exhibited for the benefit and instruction of those interested.

I hope that our Council will take up this matter, and see if my suggestion cannot be made to assume a practical form.

The President: I think this is a very interesting suggestion, and I should like to ask Mr. Crosse whether he means this to be a permanent exhibition?

Mr. A. F. Crosse: My idea is that it should be a temporary one, though I believe a temporary exhibition would probably become a permanent one. I think that all the different people who sell these things would only be too keen to have an exhibition for housing their exhibits. Most of us have but little time to go about, though if one had time to go along the reef we would see many things in use that we are not familiar with, whereas if there was a central building where all these things could be assembled, a man could come in on a Saturday afternoon and see the whole outfit.

Mr. E. M. Weston (*Member*): I am speaking without authority, but I feel sure that the Council of the School of Mines would only be too happy to allow their museum to be utilised for such an exhibition.

The President: There is no doubt this idea would be very beneficial to the industry and especially to those who cannot get to Europe every second year to see what progress has been made. Of course there is also the other museum which belongs to the town in the old building of the college. It is a pity we cannot have one decent museum instead of a lot of separate little affairs. However, we will submit the idea to the Council and see if some practical scheme cannot be arranged.

THE AMALGAMATION OF GOLD IN BANKET ORE.

By W. R. DOWLING, M.I.M.M. (Vice-President).

Amalgamation being the most important source of gold recovery on the Witwatersrand goldfield, recent developments and present varied opinions upon this subject make the time opportune for a general discussion and review of the situation.

The author does not propose to discuss the manipulative details of the method of amalgamation; such knowledge is assumed, and the general features and present tendencies of the process with the influencing factors only are dealt with. The decreasing grade of the ore due to the fact that the reduction in working costs has brought into the paying zone ore formerly not payable, and the large scale of operations to-day, necessitates the simplification and reduction of plant and operations to the minimum. The enormous tonnages of ore milled make each item of equipment and its operation cost a great deal in the aggregate, and the fact that the interest on some particular non-essential item of expenditure does not amount to much per ton crushed is beside the mark, as in many cases the capital may not be available, or, if available, could possibly be employed more profitably otherwise.

As will be shown, the relative importance of the amalgamation recovery and the methods by which it was carried out have been considerably modified by various factors from time to time in the metallurgical history of the Witwatersrand. In the early days the main source of gold recovery was by amalgamation with a small amount of concentration and subsequent chlorination. The ore was crushed relatively fine by stamps, the screens mostly in use being 800 to 1,000 holes per square inch, and the discharge was set high to aid fine crushing and inside amalgamation. To obtain the maximum recovery it was considered necessary to catch the gold at the earliest possible moment after its release from the ore by crushing. To this end amalgamated copper plates were set inside the mortar boxes, which were made large and roomy and recessed at the back to protect the copper plates. Mercury was fed into the box periodically, and lip and splash plates received the outflow of pulp. Following these plates was the large apron plate, the latter being the only plate in use in most

mills to-day. It was usual to cut the apron plate into two or more lengths, and arrange these in steps, mercury wells being placed at each step for the purpose of mixing the pulp and recovering possible floating particles. A large proportion of gold remained in the mortar-box as amalgam on the inside plates and around the dies, and was periodically cleaned up. Some of the gold discharged through the battery screen had already been wetted by mercury, and was also in the form of amalgam, and was caught on the outside plates. At the present time amalgamation tends more and more to be performed by the passage of pulp over a single plain inclined stationary plate, and the above devices of the early millman are likely to be followed into oblivion by the mercury trap, in view of the most efficient mercury trap which the tube-mill circuit affords. Cyanide was the favourite chemical employed for dressing plates, and was liberally used. It might be mentioned in passing that in some mills, many years after the introduction of the cyanide process, cyanide was still in use (1). The large mortar boxes necessarily reduced the crushing capacity of the stamps, and it was due to the introduction of narrow straight-backed boxes for the purpose of increasing crushing efficiency that inside plates became impracticable. The mercury feed to the mortar boxes, however, continued much later (2), it being argued by the exponents of this practice that since a particle of amalgam was larger and heavier than its enclosed gold particle, it was more readily caught. Inside amalgamation and mercury feed to the mortar boxes increased the mercury consumption, and made it impossible to obtain a reliable sample of the ore crushed.

Concentration by means of blanket strakes, buddles, Frue vanners, and other means was practised, and some amalgamable gold was recovered, as well as no doubt some rusty gold from the oxidised ore. Evidence of the amalgamable gold reaching the concentration plants is the amalgam recovery made on the small plates attached to the Frue vanners then running, and also on some shaking secondary plates installed at the Ferreira Mine (3).

Information is very meagre and unreliable as to the percentage of recovery by the means outlined above. The assayer in the early days was not considered a necessary official on the mine, the pan in the hands of the mine-captain or millman being thought the more reliable means of determining the value of the ore. In December, 1894, however, J. S. Curtis sets the recovery at over 60 per cent. where a 900 mesh screen was used (4). It appears, then, that the percentage recovery of the early days on an oxidised and rich ore, by the various amalgamation methods then in use, was not much less than is obtained by the present fine crushing from low-grade pyritic ore by the ordinary straight mill plate and shaking tube-mill plates, as used in most reduction works to-day.

A most important change in the methods of gold recovery was brought about by the introduction of the cyanide process early in 1890. The cyanide process in the first instance was only applied to leachable sand. This had its influence on the crushing of the ore and amalgamation recovery, since, with a view to the production of as little slime as possible, the tendency was to crush coarser, thus decreasing the amalgam recovery, although the total recovery of the gold contents of the ore was considerably improved.

When the treatment of slime was successfully introduced by our Past-Presidents, J. R. Williams and Charles Butters, in 1894, an entirely new view of ore treatment was rendered possible. The production of slime being no longer detrimental to the subsequent treatment by cyanide but the reverse, the ore was now crushed finer, thus again increasing the recovery by amalgamation. The recovery by the latter process was further improved by the use of lime in the mill service water, used to accelerate the settlement of the slime, as it was found that the agglomeration of the slime particles into larger aggregates enabled these to come into contact with the amalgamated surfaces and yield an appreciable proportion of their gold contents (5).

The increased recovery of gold obtainable by finer crushing and the removal of the main objections to this procedure by improvements in sand and slime treatment were generally recognised before the war (1899 to 1902). During this period, when local advance was debarred, progress in fine crushing was made in Australia by the introduction of tube-mills. The introduction of tube-mills for finer crushing in the Transvaal by J. R. Williams in 1904 (6) raised the recovery of gold by amalgamation to 65-70 per cent. of the contents of the ore, of which some 10 to 15 per cent. was by means of the tube-mill plates. The amalgamation recovery varied somewhat on different mines according to the re-crushing plant erected, screening used, tonnage crushed, and value and nature of the ore.

Coming nearer to the present time, it is found that the increased use of tube-mills for re-crushing enables so coarse a screen to be used in the stamp-mill that amalgamation of battery pulp becomes an impossibility owing to the scouring action of the coarse pulp upon the mill plates (7). This necessarily leads to the removal of the plates from the stamp-mill in such cases, and a consideration of the following possible alterna-

tives represented graphically in the corresponding diagrams :—

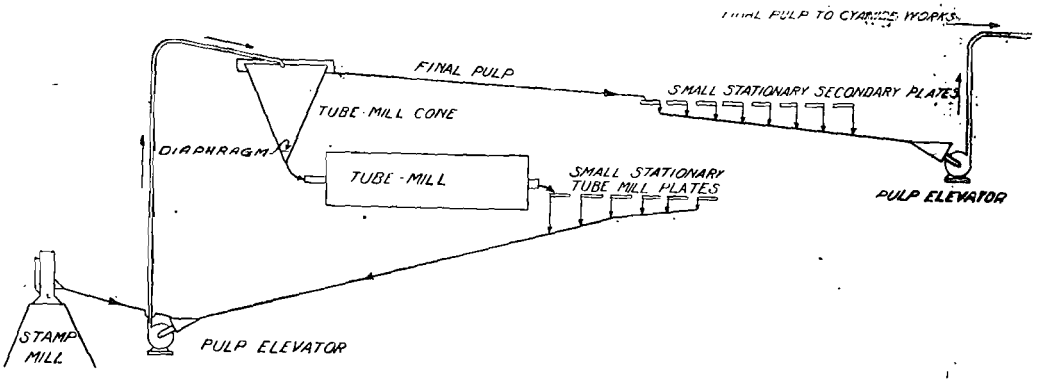
(A). Amalgamation of the tube mill pulp by one set of plates, and the overflow final pulp by another set, as at the Randfontein Central and Knight Central.

(B). Amalgamation of the tube-mill pulp only, as at the Simmer and Jack, Simmer Deep-Jupiter, and the various South Randfontein mills.

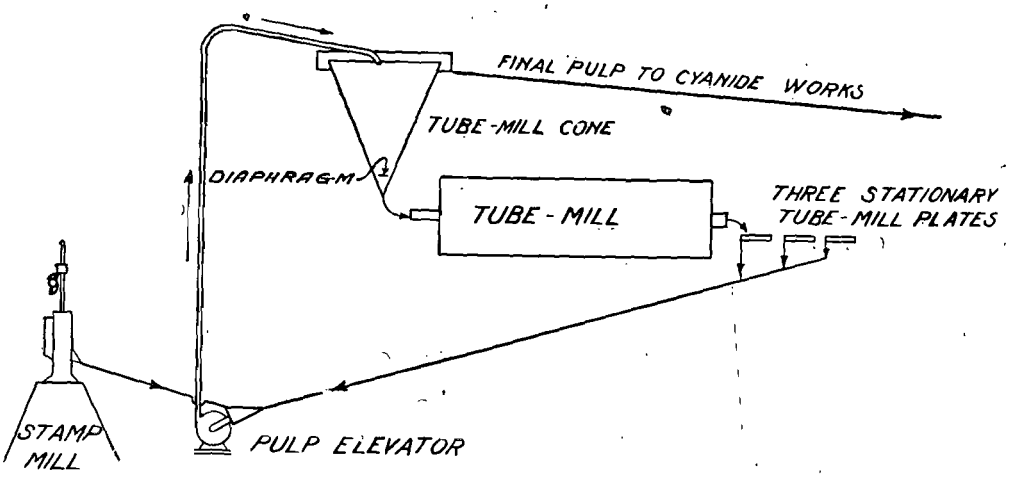
(C). Amalgamation of the mixed pulp after re-grinding of the coarse portion by tube-mills, as at recently erected plants.

After the failure of various efforts to eliminate tube-mill plates, it may now be accepted that there is general agreement amongst metallurgists as to the advisability of amalgamating the tube-mill pulp, since otherwise considerable concentration of coarse gold by the classification of both stamp and tube-mill pulp takes place in the tube-mill circuit. Tube-mill pulp has only to be examined for metallic iron from the wear of shoes and dies and from other sources to illustrate how heavy particles will persist in the circuit till crushed or abraded fine enough to overflow the classifiers.

With efficient tube-mill classification, all but the finest gold released by the stamps is brought into the tube-mill circuit and recovered on the plates, together with that released by crushing in the tube-mills. This being the case, the problem narrows itself down to determining whether it is necessary or advisable to amalgamate the final pulp, leaving the crushing plant as an overflow of the tube-mill classifiers. The considerations which enter into this problem are the extra capital outlay and buildings, the capital locked up as amalgam as setting to the plates, the increased area of amalgamated surface to guard, the consumption of mercury, the labour of dress-



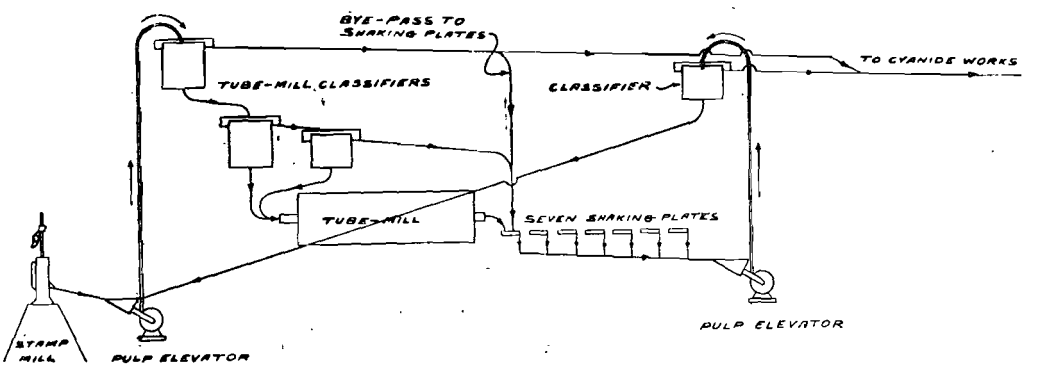
A.—Randfontein Central Arrangement
Six small stationary tube mill plates and eight small stationary secondary plates for final pulp.



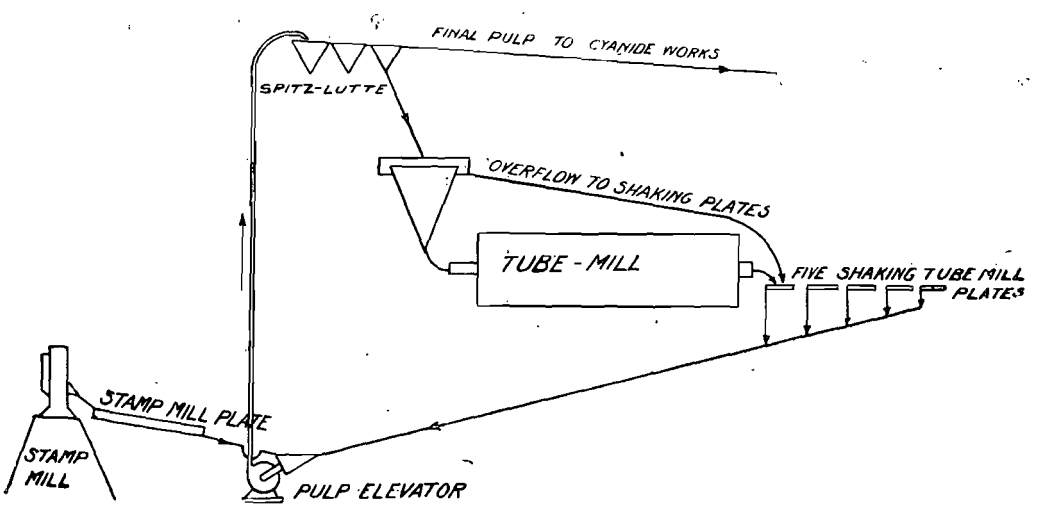
B.—Simmer and Jack Arrangement.—Three stationary plates in tube mill circuit.

ing, and the additional elevation of pulp necessitated by its fall during this amalgamation process. Even if mercury traps are used at the foot of amalgamated plates outside the tube-mill circuit for the pulp overflowing the tube-mill classifiers, it is possible for fairly coarse mercury and amalgam to escape to the cyanide plant.

On reference to the sketches, it will be noted that alternatives (A) and (C) have two elevations of the total pulp, plus the tube-mill pulp, whereas (B) has only one elevation of the total pulp, plus the tube-mill pulp. Alternative (B) therefore does not involve any extra charge for re-elevation, and remains the same as in the usual present-day practice, as represented in sketch (D). Unless increased recovery counterbalances these disadvantages, it is preferable to omit this additional operation. The use of very coarse battery screening naturally reduces the amount of metallic gold set free in the stamp-mill, and where the bulk of the crushing is done by the tube mills, the quantity of very fine gold which might reach and overflow the tube-mill cones is correspondingly reduced, as such gold is retained on the tube-mill plates before the elevation of the tube-mill pulp. Even where the coarseness of crushing in the stamp-mill does not preclude the use of plates there, their elimination, when conditions permit, is followed by the various advantages above mentioned, and the rendering available as immediate profit the gold on such plates, which would otherwise not be realised until the end of the life of the mine. Plates thus taken out of the stamp-mill may carry from £50 to £300 worth of gold per plate, according to their condition, and in the aggregate this may render some £20,000 worth of gold immediately available in the case of a large mill. The elimination of the final pulp plates, however, can only be considered safe where good tube-mill classification obtains, and an equally efficient cyanide treatment follows. In the cyanide plant



C.—Recent Arrangement.—Seven plates amalgamating total mixed pulp.



D.—Common Arrangement.—One plate for five stamps and five shaking tube-mill plates for one tube-mill.

clean separation of sand from slime is essential, such as is possible with large diaphragm cones and continuous sand collection, producing on the one hand freely leachable sand, and on the other a large percentage of slime free from coarse sand. With tube-mill classifiers liable to choking of small underflow outlets and consequent overflow of coarse sand and gold to the cyanide plant, and with slimy semi-permeable sand charges, the wisdom of allowing any fine amalgamable gold to escape from the crushing plant is doubtful, in spite of the economies above mentioned. In the case of the Simmer and Jack mill, where there are plates in the tube-mill circuit only, though the cyanide residues remain normal, the amalgam recovery dropped from 64% to 55% on a $6\frac{3}{4}$ dwt. ore, owing to the fact that the fine sand and slime overflowing the classifiers are not amalgamated at all. If necessary, and as shown later in the case of the Simmer Deep plant, finer crushing and larger relative tube-mill cone area could be employed to increase the percentage of gold recovery by amalgamation. Whilst the collected sand is slightly enriched by some very fine free gold, there is no difficulty in dissolving this and obtaining a complete recovery. The popular theory that only 85% of this gold is recovered by cyaniding, instead of 100% by amalgamation, has not so far been borne out either in the laboratory by residue assays or regular panning of residues. Any fine metallic gold overflowing the tube-mill cones is necessarily smaller and more easily dissolved than the average of partly-encased particles in sand which are dissolved by cyanide. The amount of fine metallic gold overflowing the tube-mill cones is much affected by the number and area of the latter. Where a large ratio of tube-mills to stamps exists, such gold, more often than not, cannot be detected in the final pulp by panning or in the sand before treatment. This is the case at the Simmer Deep plant, where, before the removal of the stamp-mill plates, the total plates in operation numbered 82, equivalent to 5,276 sq. ft., of amalgamating area, and the amalgamation recovery was 56·8% of the ore value. After the removal of all the stamp-mill plates and the reduction of those in the tube-mill circuit to 30 stationary plates, presenting an area of 1,700 sq. ft., the amalgamation recovery was 57·7%, and the value of sand and slime before and after treatment has not increased. The total extraction by amalgamation and cyaniding was 93·4% before and 93·5% after removal of the stamp-mill plates. There was slightly finer crushing of the ore in the latter case, there being 77·6% of – 90 mesh (0·006 in.) product in the final pulp before removal of the mill plates, and 81·0% of – 90 mesh (0·006 in.) product after the removal. Where the ratio is lower, as at the Simmer and Jack, a trace of very fine gold can be detected by panning the final pulp and collected sand, though the assay value of the sand residue remains normal. The slime is also enriched, though this may be off-set by better washing, due to the lower percentage of moisture on settlement of well-classified slime, containing a higher percentage of fine (– 200 mesh) sand. For instance, assuming that 1·5 dwt. and 1·8 dwt. slimes are treated, settling to 42·5 and 35% moisture respectively, by two 4 : 1 washes in both cases, of which the first wash only is precipitated down to 0·01 dwt. per ton of solution. With 0·08 dwt. undissolved gold in the first residue and 0·10 dwt. in the second, the total extraction will be 90·4% (0·144 dwt. total residue) from the 1·5 dwt. slime, and 92·2% (0·14 dwt. total residue) from the 1·8 dwt. slime. Where Butters' or other vacuum filters are used, the value of the original slime should be immaterial, as, providing the dissolving of the gold is satisfactory, the residue should contain little more than the trivial amount encased. It is to be hoped that detailed results and working costs of the recent work in this direction at the Crown Mines will before long be laid before our Society.

The author is indebted to F. A. G. Maxwell for much information on regular work at the Randfontein Mines, where plates were eliminated from the stamp-mill, and hopes that he will bring forward his results as a discussion to this paper. In the meantime, it may be said that during 1908, in the four 100-stamp sections of the South Randfontein, when there were amalgamated plates both in the stamp-mill and the tube-mill circuit, the average amalgamation recovery was 54·46%, and the total recovery 92·06%; whereas in 1910, when there were plates in the tube-mill circuit only, the amalgamation recovery was 48·60% and the total recovery 92·94%.

As mentioned in a note lately presented by the author to this Society (8), it has generally been considered necessary for the satisfactory amalgamation of the large volumes of thick tube-mill pulp that the plates should shake. Tracing back the history of this belief to the pioneer work done by J. R. Williams on the Glen Deep in 1904, it seems that in order to prevent the banking of sand on the stationary plates installed with 10% grade, it was found necessary to mount the plates on shaking vanner frames. Not only has the shake been subsequently adopted by all the mines without question, but the number of plates has gradually increased from the two originally installed to five and even six later. When the Knights Deep had five tube-mills, equivalent in cubic capacity of shell to 3·91

standard tube-mills of 22 ft. × 5½ ft., sixteen shaking plates 10½ ft. × 4¾ ft. were in operation. As there was no convenient space for more plates when another tube-mill was erected, it was decided to make the sixteen plates serve. The number has now been reduced to fourteen. The ratios are :—

For 3·91 tube mills 16 plates = 4·09 plates per tube-mill.

For 4·91 tube-mills 16 plates = 3·26 plates per tube-mill.

For 4·91 tube mills 14 plates = 2·55 plates per tube-mill.

The amalgamation recovery was just as efficient with the smaller ratio as with the larger.

In the course of the experiment mentioned in my note, and after having thoroughly tested the stationary five plates system, initiated originally by F. A. G. Maxwell at Randfontein (9), it was decided on W. A. Caldecott's suggestion to perform the whole amalgamation of the tube-mill pulp on two stationary plates per tube-mill. The Simmer East plant has three 22 ft. × 5½ ft. tube-mills. The pulp of each mill has been regularly run over two plates only since December 19th, 1910. The dressings have been at six hour intervals and the scrape taken six inches lower than formerly. The daily scrapes have been normal, yielding the amalgam called for. The two plates of one of the mills were steamed at the beginning of the run and again at the end of a month's run to determine whether the yield from this source was affected in any way. The previous practice in this plant was to steam one-third of the total plates each month so that each plate ran for three months before steaming. It was found that one month's run of the two plates yielded more amalgam when multiplied by three, than the average of the three months of five shaking plates previous to starting the non-shaking experiments. However, to be on the safe side, it was assumed that the plates would not accumulate steam amalgam at the same rate during the second and third months, and it was decided to multiply by only two and a half. This calculation gave a figure for steam amalgam just about equal to the average of the three months of five shaking plates. Adding this to the daily scrape and other sources the recovery is fairly arrived at, and works out at 21·54% of the screen value and 34·17% of the tailing value, compared with 21·80% and 34·04% of the previous five-shaking-plate period.

To ensure successful and economical work the ratio of water to solid should be reduced to the minimum, as not only does this decrease the velocity of the stream over the plate, but the cost for re-elevation is also less. The volume of pulp of 400 tons of sand per tube-mill plus 480 tons of water, allows about 10,000 cubic feet of pulp per 24 hours for each of the two plates, or about the same as passes over the plate of a five-stamp battery with a 9-ton duty per stamp, and a 6·5 to 1 ratio of water to ore in the screen pulp. The tube-mill circuit is the only part of the crushing plant where a thick amalgamable pulp is available without the installation and operation of special classifiers for the purpose. Distribution of the stream across the full width of the plate is also of considerable importance, and may be obtained by using the usual box at the head of the plate perforated with about ¾ in. holes along the bottom of the side facing down the plate. These holes should be of such capacity that the pulp attains some head in the box. In the event of any of the holes becoming temporarily choked, a further row of holes an inch or two higher up the front side of the box will serve to carry over the stream, and still maintain a fairly good distribution. The fall given to the stationary plates is 18%. This is the fall arrived at by Mr. Maxwell and since adopted here. With the fall of 18% it is found that the minimum percentage of water in the sand is 55%, and it appears that should it be desired to reduce the fall, the moisture would have to be increased, fall and water ratio being convertible terms within limits. The point to be reached in water ratio and fall is that just short of banking of the sand on the plate. Even with a low water ratio the rush of pulp over two plates appears to the eye to be too great, but this is not really so. As long as the velocity of the pulp stream is such that particles in suspension may come into contact with the plate and not be carried on mechanically without touching the plate, the gold will be caught. It might be considered that pulp carrying only 55% of moisture is too thick to allow the gold to sink through and reach the plate. As, however, amalgamation has been proved efficient with this ratio, this does not appear to be the case. In this connection, it might be mentioned that in pan amalgamation the pulp is kept very thick, although in plate amalgamation a fluid pulp was heretofore considered necessary for good recovery. On a stationary plate there appears a tendency for the gold to amalgamate further down the plate than when shaking; there is hence a liability for the scraping to yield less amalgam and the steaming more with stationary plates as compared with shaking plates. In connection with the satisfactory results obtained in the operation of the two stationary plate method, much credit is due to the practical skill and progressive attitude of Mr. A. J. Herald, the Knights Deep mill foreman, and to his assistant, Mr. Peter Wilson.

The removal of plates from the stamp-mill has had the good result of introducing on these fields the separate plate-house. The Homestake mine and the Waihi (10) set the example many years ago, but the Witwatersrand has been very slow in following the lead and accepting the suggestion made before this Society in 1903 by H. R. S. Wilkes (11). The arguments advanced by some mill-men that a plate attached to each battery makes for good recovery due to better distribution of the pulp by the stamps over the width of the plate, and that even vibration assists lacks confirmation. It appears to the writer that this method is just one of the many practices taken over from other countries without investigation. Tube-mill plates have now been running long enough to prove that most efficient amalgamation can be performed with a pulp sub-divided from a launder and distributed over the width of the plate from a box pierced with holes. Tube-mill pulp carries heavier particles and contains less water than mill pulp, and is therefore a more difficult pulp to distribute. The advantages of a separate plate-house are increased running time, since stamps need not be stopped for plate dressing, and greater attention to the crushing machinery by the attendants, thus increasing the crushing capacity. Again, better amalgamation should be obtained by the greater attention to the work by specially chosen and trained men. The whole operation is confined to a smaller area and lends itself to closer supervision so that the risk of loss by pilfering may be reduced to a minimum. In connection with the matter of conveying pulp the author considers that sufficient importance is not attached to launders. All launders should be as smooth as possible and free from obstruction at the joints. Of all material used for the lining of launders there does not seem to be anything better than cement. This provides an ideal surface presenting least resistance to the flow of the pulp and affording very little lodgment for amalgam and still less for gold in unamalgamated pulp. On those mines here, where separate plate-houses or combined plate and cyanide extractor houses have been built, the arrangements have not gone far enough or else too far. The erection of zinc-lathes, elevator pumps and tube-mills in the same building as the plates and the extractor boxes means the introduction of gangs of men and natives for repairs, other than those engaged on the gold recovery work. If this machinery is placed in the same building because the launders may carry gold, then the whole tube-mill plant including circuit launders and elevators should be included. The best plan is to include in the building only such machinery as is actually connected with recovery work, so as to limit the men entering the building to those directly engaged in the work. The accumulation of amalgam in the tube-mill launders should be reduced to a minimum and the launders should be closed to protect what does collect. Prior to the introduction of tube-milling amalgam was to be found in the launders to the cyanide plant, and some found its way as coarse particles to the concentrate vats, entailing loss in the residue. Where efficient tube-mill classifiers are installed no amalgam is found to-day reaching the cyanide plant except in an impalpably fine condition, since heavy particles of amalgam pass into and are retained in the tube-mill circuit. There is always a loss of mercury in amalgamation mainly by abrasion, and where this enters the tube-mill amalgamation of the gold liberated by crushing takes place. Where this amalgam does not reach the plates it is found in the launders between the discharge of the tube-mills and the plates. Such impalpably fine mercury or amalgam as enters the cyanide plant dissolves in the same way as gold, and like dissolved lead salts is either wholly or partially precipitated by soluble sulphides in the working solution, or later in the metallic form by zinc shavings in the boxes.

Where plates are eliminated from the stamp-mill the water ratio may be considerably reduced. The clearing of a long plate set at a fall of 9% or 10% by avoidance of any banking of sand from the pulp has been the factor determining the water ratio in the past, and this requires six to seven parts by weight of water to one of solid using average battery screening. It has been found that in the absence of mill plates the ratio may be reduced to 4·5 without serious ill effects provided the mill launders have sufficient grade to carry pulp of this consistency. A coarse screen with less water will give the same result in duty and grading as a finer screen and more water, water and screening being convertible terms within limits. The reduction of water has considerable influence on the cost of pulp elevation and of running water pumps, the reduction of weight from 6·5 to 4·5 of water to 1 of solid being equivalent to about 27%. The saving in power in a large plant is material when working on the reduced ratio, and there is likewise a smaller load on the classifiers and slime collectors. There would, however, be no saving in the consumption of water as much the same volumes are exposed to loss by evaporation, seepage in dams, and moisture in residues.

Although there is no rule for the area of amalgamating plates there is no doubt that the number and area of plates in common use is unnecessarily large and when these are placed in a separate plate-house the total area may be materially reduced. The author is of opinion

that much of the amalgam found at the lower end of long battery plates is largely worked down by amalgamators wishing to present a uniform bright surface all over the plate. The old practice was to instal plates about 15 ft. long × 5 ft. wide per battery of 5 stamps, which equals 15 sq. ft. per stamp. Where say one standard large tube-mill for 30 stamps is erected, and assuming that the tube mill is equivalent to 30 stamps, five plates of 11 ft. × 5 ft. would probably be erected. The combined plate area is then :—

30 stamps = 6 plates 15 ft. × 5 ft. = 450 sq. ft. = 15 sq. ft. per stamp.

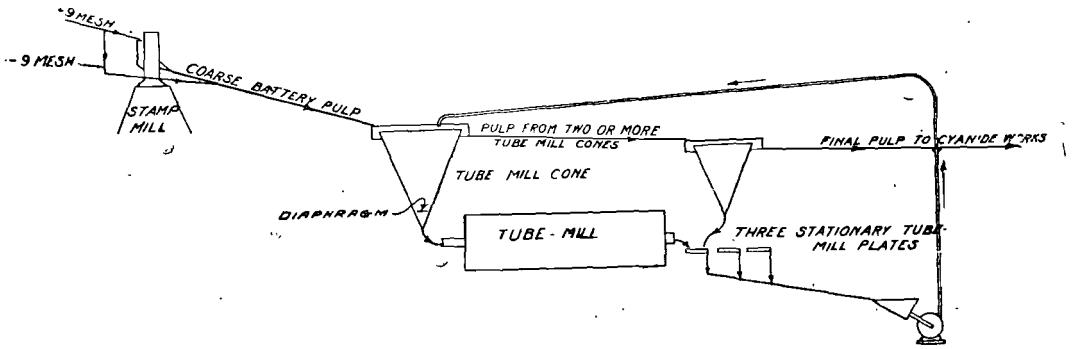
1 tube-mill = 30 stamps – 5 plates 11 ft. × 5 ft. = 275 sq. ft. = 9 sq. ft. per stamp.

Total for 60 stamps units = 725 sq. ft. = 12 sq. ft. per stamp unit.

At the Randfontein Central the total plates erected amount to 9·6 sq. ft. per stamp unit but it is now proposed to use only 4·8 sq. ft. per stamp unit, and there is little doubt that the latter area will be quite sufficient. In the Simmer and Jack re-arrangement the plate area is about 2 sq. ft. per stamp unit. With the high stamp duties of recent years the same plate area is doing nearly double the work it used to do in the past, and is capable of doing a great deal more. Conversely, in regard to the usual 10% fall, where this is increased more crushed material with smaller water ratio may be amalgamated with equally good results. H. W. McFarren in his recent interesting little book (12) says the fall of plates varies from 1½ in. to 3 in. per ft., and should not be less than 2 in. or 2½ in. A fall of 2 in. per foot is equivalent to 16·7%, and 2½ in. to 20·8%, whilst the average is 18.75%. Among the advantages due to reduced area of plates are reduced cost of installation and operation, and especially a reduced cost of mercury the consumption being in proportion to the area of plate. In the Simmer and Jack plant the consumption of mercury has been reduced to one-sixth of the former figure. Where a large area is exposed in amalgamation a proportionate amount of gold is taken up to set the plates, and is not available for realization until the end of the life of the mine. The gold held by well-set plates may be taken at 1 oz. per sq. ft. of plate. A 200 stamp-mill having 40 plates 15 ft. × 5 ft. will thus absorb 3,000 oz. of gold worth £12,000. The future points to the reduction of plate area to the point where only the gold too coarse for cyaniding will be caught by amalgamation. This point has probably been reached in the Simmer and Jack plant which has now only eighteen stationary plates in the tube-mill circuit in place of the former 64 battery plates and 30 shaking tube-mill plates. It will be noted that the Simmer East with plates in the stamp-mill have retained two stationary plates per tube-mill, whereas the Simmer and Jack and the Simmer Deep-Jupiter joint plants without plates in the stamp-mill have each retained three plates per tube-mill. The additional plate in the last two cases is a measure of safety to deal with the richer pulp and to have not less than two plates per tube-mill in action during dressing operations. Diagram (E) shows a proposed arrangement of crushing and amalgamating with three stationary plates only per tube-mill and no plates in the stamp-mill, and with safety cones for the common streams of pulp overflowing the tube-mill classifiers (13).

The author trusts that the foregoing account of some considerations affecting the methods of gold recovery responsible for some £20,000,000 per annum, two-thirds of the Rand's total output, may be of interest to members, including those who have visited the plants where the changes described have been effected, and he trusts during the course of his reply to the discussion to furnish future comparative details of future results.

In conclusion the author wishes to thank Mr. G. A. Chalkley, the Manager of the Knights Deep, and Mr. C. D. Leslie, the Superintending Engineer of the Consolidated Goldfields of South



E.—Proposed Arrangement.—Three stationary tube mill plates.

Africa, for access to information of the various mines of the group, and for permission to read this paper before the Society.

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(10) C.M.M.S. "Notes on Waihi Ore Treatment," by Ralph Stokes, July, 1907, Vol. VII, p. 12; also "Mines and Minerals of the British Empire," by the same author.
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(12) "Practical Stamp Milling and Amalgamation," by H. W. McFarren, p. 91.
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The President: It gives me very great pleasure to propose a hearty vote of thanks to Mr. Dowling for his excellent paper. Not the least interesting part of this paper is, I think, the excellent historical review he gives at the beginning, which is very useful to those of us who were not here before the war. I think it is a precedent which others who had the privilege of being here before the war might follow.

Mr. A. F. Crosse (*Past-President*): As Mr. Dowling was my assistant for some years, I should like to take this opportunity of mentioning my appreciation of his paper this evening. I am sure it will lead to a very interesting discussion, and it is a good thing we have a milling paper to discuss as they have been rather scarce in the past.

Mr. K. L. Graham (*Member of Council*): I have very great pleasure in seconding the vote of thanks to Mr. Dowling for his paper. His contributions to the Society are always interesting, and this one is no exception to the rule. It opens up an enormous field for discussion and will probably occupy a lot of time at our next few meetings.

AIR LIFT AGITATION OF SLIME PULP.

By ROBERT ALLEN, M.A., B.Sc., M.I.M.M.
(Member of Council).

One of the principal causes of the fall in the price of silver in recent years has been the increased production of the metal, due to the impetus given to the industry by the great improvement that has taken place in the methods of cyaniding ores. Formerly these ores, when treated by the cyanide process, were first milled, then generally concentrated, the separated sand being leached, whilst the slime was mechanically agitated, the treated pulp being decanted as on the Rand. The leaching process was very protracted, and required a plant of very large capacity, and it was found that to obtain satisfactory results it was necessary to grind the ore very fine—the finer the better—owing chiefly to the fineness of dissemination of the principal silver-bearing mineral, argentite, throughout the gangue. The result of this was that the percentage of slime to be cyanided was gradually increased, whilst that of the sand was at the same time lessened, until in some plants the whole of the ore was "slimed."

The treatment of silver-bearing slime pulp in mechanical agitators has been always an expensive and unsatisfactory method. With the larger sizes of vats the stirring gear is generally of the clumsiest construction, due to the conditions imposed upon the apparatus, and a great waste of power occurs. Also in most cases after a few hours' agitation—less than six—unless fresh air is introduced into the pulp, the cyanide action on the silver-bearing pulp (which requires more oxygen than a gold-bearing pulp) ceases.

Early in 1902 F. C. Brown, engaged in the treatment of silver-gold ores on the Komata Reefs Mine, in New Zealand, in conjunction with S. D. McMiken and others, began experimenting with a cone bottom vat, 7 ft. 6 in. diameter × 37 ft. deep, for agitating slime by means of compressed air, the cone having an angle of slope of 55°. Later on in the same year he agitated charges in conical bottom vats, 10 ft. × 39 ft., the cones having an angle of slope of 60°. In 1903 he attempted to agitate finely-ground sand, of which 93% passed a 200 mesh, in a 10 ft. × 39 ft. vat with a jet of air only, as before, but found that the air moved up through a channel in the centre of the vat, and

the bulk of the sand received very little agitation. Early in 1904 the central air lift tube was successfully used in these tall vats, and the arrangement became a standard one for agitating finely-ground sand, and concentrates as well as slime.

As the result of much experimenting, it was found that :—

1. The higher the vat compared with the diameter the less was the power required.
2. The consumption of cyanide was less in tall than in shallower vats, owing to the fact that the necessary quantity of compressed air (and therefore the amount of carbonic acid passed into the pulp) is less per ton of ore in the former than in the latter.
3. That although good agitation could be effected with slimy material in shallow vats, yet, as the coarseness or the density of the material, or the specific gravity of the pulp to be agitated, increased, the ratio of height to diameter should be increased likewise.
4. The diameter of the central air lift pipe should be $1\frac{1}{4}$ in. for each foot of diameter of the vat.

Incidentally it might be noticed that the cyanide consumption with these agitators was found to be slightly less when potassium cyanide was used than when sodium cyanide was used.

The following table gives the average results of experiments made by F. C. Brown in New Zealand, and averaged over several months each, by agitating different classes of material in different sizes of vats :—

Size of vat.	Material agitated.	Tons (dry) of material.	Cub. ft. of free air required per minute.	Pressure of air required lb. per sq. in.	H.P. required.
10′ × 40′	Sandy slime	30—40	10—13	25	$\frac{3}{4}$—1
7′ 6″ × 37′	Slime	15	4—6	22	$\frac{1}{3}$—$\frac{1}{2}$
7′ 6″ × 37′	Finely ground concentrate	15	15—20	26	$1\frac{1}{2}$—2
13′ × 55′	Slime	75	32	32	$1\frac{3}{4}$

These tall vats, the use of which has spread to other parts of the world, are known in New Zealand and Australia as Brown and McMiken vats, in Mexico and U.S.A. as Pachuca tanks, and in other places generally as Brown vats or Air Lift vats. They were introduced into Mexico early in 1907 by Albert Grothe at the Hacienda San Francisco, at Pachuca, for agitating an all-slimed silver ore, and with such successful results that their use has since become general in that country, very few silver-treatment plants being without them, the mechanical agitator having been practically superseded. Now in Pachuca alone there are nearly 80 large vats of this kind. A large number is also in use in U.S.A.

Fig. I. is a plan of Grothe's design of the vat he first installed. It will be noticed that it consists of a cylindrical shell stiffened by horizontal angle iron rings and terminating at the bottom in a double angle iron ring, which rests on a substantial concrete foundation. The cone, which

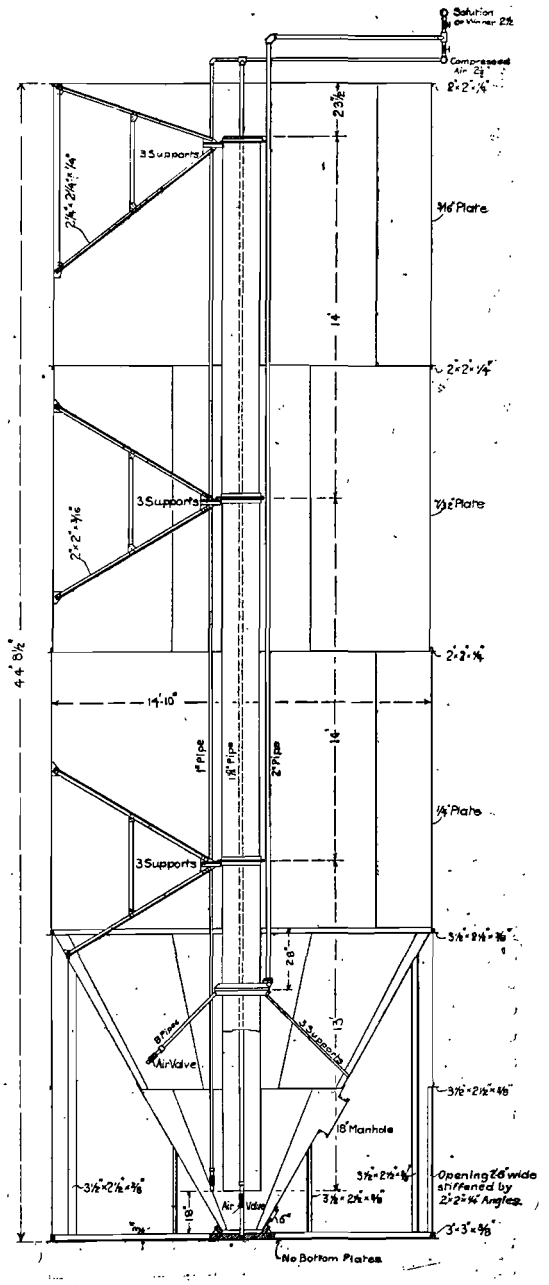


FIG I.

has an angle of 60°, sits upon a heavy cast-iron bottom, which, it will be noticed, supports one-third of the weight. A door-way in the lower part of the shell, having an opening 3 ft. 6 in. high × 2 ft. 6 in. wide, gives access to the space inside the shell below the cone to a manhole and a discharge cock. Openings in the shell give light to the space below the cone. Fig. II. is a

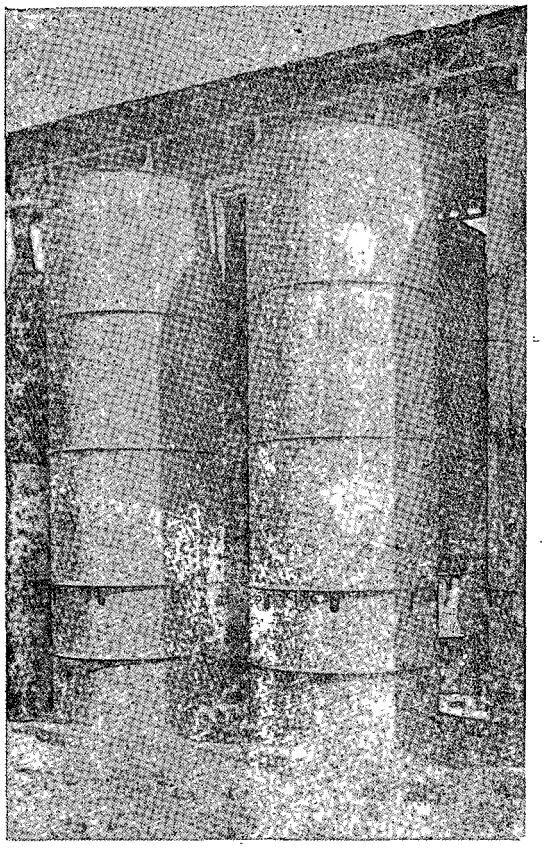


FIG II.—Brown Vats at San Ra Fael Mill, Pachuca.

photo by Edmund Girault of two of these vats at the San Rafael mill, Pachuca, which shows these openings. The vertical joints of the shell below the cone are stiffened by angle irons. The vertical air lift pipe, in the centre of the vat, has its ends usually 18 in. below the top and 18 in. above the bottom of the vat. The central air pipe is generally the only one used in agitation; but the vertical air pipe external to the air lift pipe is useful in keeping a clear channel through the pulp during the process of filling a vat. When a vat is full it usually only takes a few minutes after the central air supply is turned on to get the whole contents into agitation. The air required at first is, however, much above the normal requirement, generally double. As soon as the contents of the vat are in circulation and the cone is free from banked-up material the air supply should be reduced, to avoid a waste of air, until the cone just keeps clear. This is ascertained by feeling the surface of the cone with a lead weight attached to a string.

The eight radially arranged air pipes are useful to start the agitation of a charge which has stood for some time. After several weeks' stoppage it is usually possible to get a charge into perfect agitation within an hour. (It might be noted here that, in the case of the stoppage of a large diameter mechanical agitator for a short time only, it is necessary to dig out its contents to free the paddles).

In the case of ores which have a tendency to scum when pulped, it is best to fill the vats not so full as usual, so that the falling pulp can be made use of to break up the scum.

For the air valve the delivery pipe is usually perforated in several places all around for a length of about 4 in., and a short length of rubber tubing of good quality, strengthened at each rim by an extra ring of rubber, is slipped over the pipe, the valve behaving similarly to that of a bicycle. This rubber sleeve is practically the only thing that ever needs renewal, and the repairs on this type of agitator are almost nil.

The air from this valve in the air lift escapes into the pulp in a thin spray around the air pipe, and mixes intimately with the pulp. Large bubbles at the discharge of the air lift occur when the agitation of a charge is commencing, but in normal working these generally indicate a waste of air. When the air is properly delivered to the pulp, the latter is saturated with it, and it is this air in a fine state of division, and not that of the large bubbles, that assists the chemical action.

It has been urged against the air lift agitator that an unnecessarily large consumption of cyanide occurs with its use, due to the carbonic acid in the air, and also that atomized oil from the compressor, introduced into the pulp, interferes with the chemical action. With regard to the former, the air certainly could be deprived of its carbonic acid before compression by means of lime, but nobody so far seems to have thought it worth while to do this. In the case of the latter, with ordinary care, the amount of oil atomized is so small that it is negligible. Compressed air of sufficient pressure and free from oil might be produced by means of turbine blowers worked in series, and the economy of this method of production of compressed air, for agitation purposes, might be looked into.

The consumption of cyanide in a Brown vat is not usually great in practice; in fact it is usually

less than with other agitators when treating the same pulp. This is mainly due to the fact that a charge of slime is much more rapidly treated in a Brown vat.

At Pachuca, almost side by side, are two plants, the Loreto mill and the Hacienda San Francisco, both treating "slimed" and concentrated ore, of almost identical character and value. In the former are 30 ft. mechanically agitated vats, each taking a charge of 57 short tons in a thin pulp, whilst in the latter, as mentioned above, are Brown tanks 15 ft. × 45 ft., operating upon charges of 112 short tons in a thick pulp. The cyanide consumption per ton in the former mill is greater than in the latter, whilst its power consumption is at least ten times as much.

Some years ago at the Waihi Grand Junction Mine, N.Z., F. C. Brown made some comparative experiments with different types of agitators. The agitators were:

(A) Shallow vats with ordinary stirring gear.
(B) Brown vats, 13 ft. × 55 ft.
(C) Brown vats, 7 ft. 6 in. × 37 ft.
(D) Shallow vats each with a central tube and a high speed screw propellor. (These last are known in Australasia as the A.Z. agitators: the Hendryx is of the same type.)

In (A) and (B) for the same period of two months the same class of slime pulp was agitated, the average consumptions of sodium cyanide being respectively 2·6 and 2·4 lb. per short ton. In (C) and (D), also for a period of two months, the same class of ground concentrate pulp was agitated, the average sodium cyanide consumptions being 2·4 and 3·1 lb. per short ton.

In August, 1908, J. Leslie Mennell, of Mexico City, first proposed the series or continuous system of treatment in Brown vats as an improvement upon the ordinary intermittent treatment system. In the latter system, which until recently has been that commonly in vogue, the agitators are each worked on the charge system, each vat being filled separately, its contents being agitated and then discharged. In the former system, now largely superseding the intermittent, he proposed to feed the pulp into one vat only, the head of a series of vats, and connect the successive vats so that the pulp flowed proportionately and at the same rate through the series. In conjunction with A. Grothe, in a series of small vats 18 in. × 54 in. exactly on the Brown model he made a series of tests to see if the system would work in practice. The vats were placed on the same level. Some difficulty was encountered at first in finding a suitable form of connection between the successive vats which would prevent segregation of coarser material in the vats at the head of the series. After some trials the connections between the vats were arranged as shown in Fig. III., and a uniform quality of flow from vat to vat secured. Then a silver ore somewhat complicated by base metals and requiring a long time of agitation for commercial treatment was selected for experiment and finely ground. Two experiments were made upon this ore, one with a single vat, which was charged with 154 lb. of ore and 220 lb. of solution, and the other with four vats in series as shown in Fig. III. In the latter case ore and solution were fed into the head vat every fifteen minutes for eight days, at the rate of 154 lb. of ore and 220 lb. of solution per 24 hours. After five days, when the plant seemed to be working regularly, samples were taken from all the vats. Sampling was continued to the end of the eight days and values were found to be uniform. The

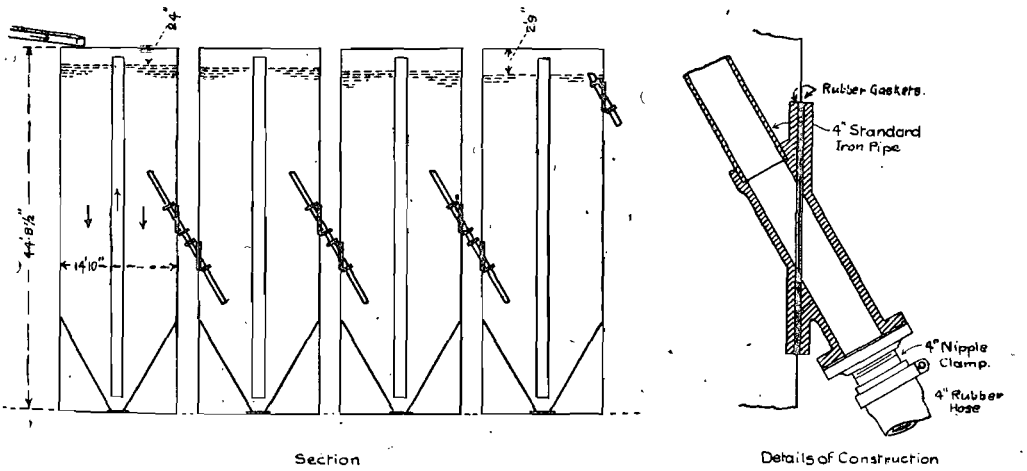


FIG. III.—Grothe and Carter Vat Connection for Continuous System.

contents of the single vat were also agitated for 48 hours, and were sampled at intervals.

The results of the two experiments are here shown side by side in tabular form :—

Hours of Agitation.	Strength of Solution. %KCN.		Cyanide Consumption. Lb. per Short Ton.		Silver Value. Ounces per Short Ton.		% of Extraction.	
	Single Vat.	Series Vats.	Single Vat.	Series Vats.	Single Vat.	Series Vats.	Single Vat.	Series Vats.
0	·39	·39	0	0	10·82	10·91	—	—
12	·26	·30	4·31	2·84	3·66	4·26	66	58
24	·20	·26	5·96	4·09	3·15	3·56	69	68
36	·13	·20	8·27	5·96	2·86	2·95	71·2	71·3
48	·12	·18	8·49	6·50	2·77	2·34	72·7	77·3*

In the above sets of figures those of the series vats refer to the effluents from the four vats respectively. These experiments and the advantages of the new system were described by Mennell in the *Mexican Mining Journal* of February, 1909. Briefly put, the advantages of the continuous system are that :—

1. The time of emptying and filling vats can be employed more usefully in continuously agitating the pulp.
2. Individual attention to the operation of each vat being no longer necessary, all that is required is a watchman to see that all the vats are properly supplied with air, and that there is no wastage of air.
3. The cost of discharging the individual vats is done away with.
4. The loss of head is practically saved, the level of discharge being only about three feet below the top of the last vat. Thus the treated pulp can flow by gravity to the filter plant instead of being pumped to it, as is usual with the intermittent system.
5. In case a sudden extra demand is thrown upon the cyanide department of a mill there is more chance of the residues from a continuous system being lower than those from an intermittent system.

Soon after the publication of the results of the Mennell-Grothe experiments, several mining companies in Mexico arranged to try the system practically, amongst them the Esperanza G. M. Company of El Oro, State of Mexico. This company had formerly six vats in use on the intermittent system, each 14 ft. 10 in. × 44 ft. 8½ in., for treating charges of 90 short tons of finely ground battery sand (without true slime). The pulp, consisting of 1 part of sand to 1·8 of solution, was agitated for 33 hours, this including half the time of filling and discharging, viz., 5·1 hours. If the pulp had been treated continuously, the time of agitation would have been increased to 38·1 hours, an increase of 15·3%. This loss of time in agitating and the great expense of the elevator system incurred in discharging the vats made the company decide to adopt the continuous system, and the vats were connected by 6 in. pipes. The result was quite successful, the transfer of pulp from vat to vat being perfectly regular, whilst, in addition to the saving of labour and cost of elevation, there was an actual saving of cyanide of $\frac{1}{20}$ lb. per ton, while the increased extraction of gold was 1·3% and of silver 1·5%, the original gold and silver contents being 0·196 oz. and 2·34 oz. per short ton. This modified plant has been in successful operation since January, 1910.

Fig. III shows the method of connecting up a series of vats for the continuous system in the way designed by Grothe and Carter, of Mexico City. The connecting pipes are at an angle of 60°, at which angle they are kept clean. The flexible pipe connection outside the vats is introduced to do away with the effect of vibration and unequal expansion.

It will be noted that a small loss of head, due to friction, occurs with a series of vats, the level of pulp in each vat being slightly lower than that of the vat immediately preceding it. In the six vats of the Esperanza mill the friction drop from vat to vat was about 6 in., or a total drop of about 30 in. for the six vats : in the last vat, however, the requisite portion of the pulp, discharged by the air-lift, was cut out for final discharge by a box, 12 in. × 12 in. × 24 in. placed above the level of the pulp in the vat : the loss of head with the six vats was thus reduced to 12 in.

In designing an agitation plant on the continuous system it is advisable to experiment upon the ore in a single vat first. By plotting the percentage extractions, say at the end of every hour of treatment, it will be seen when it would cease to be profitable to further agitate the ore. On dividing the curve by a series of equidistant lines, parallel to the axis of the curve representing the extractions, into as many spaces

* This was confirmed by the bulking and re-assay of the whole of the residues.

as the number of vats it is intended to use, it will be seen what extraction in each vat may be expected.

The air lift system of agitation is not only valuable for the cyanide treatment of current material, but it is also an excellent piece of apparatus for the preparation of accumulated slimes for their subsequent treatment by cyanide.

In the *Journal* of this Society of July, 1897, Dr. W. A. Caldecott, writing upon the chemical conditions existing in accumulated slime, showed how the neutralization of cyanicides in it would be effected by their gradual oxidation through exposure to air, and in March, 1898, in conjunction with John Kelly, he took out a Transvaal patent, No. 1559, for an air lift apparatus for the agitation and aeration of slime pulp or other gold-bearing material. This method of sweetening accumulated slimes by means of atmospheric air is utilized on the Geldenhuis Deep G. M. Co.'s plant and also on that of the Luipaard's Vlei Estate & G. M. Co.

The first Company in the Transvaal to use the air lift principle in the treatment of accumulated slime was the Nourse Mines, Ltd., a flat-bottomed vat 30 ft. in diameter and 12 ft. deep being used. The Geldenhuis Deep G. M. Co. to oxidize accumulated slime uses two conical bottomed vats each 20 ft. in diameter and 20 ft. deep, the vertical depth of the cone being 5 ft. These vats have air lift pipes 16 in. in diameter, in the lower portions of the vats, of one-half the depth of the vats only. Pipes were formerly used in length equal to the full depth of the vat, and then afterwards tried of shorter length with much more satisfactory results. The slime is pulped with 2 parts of water in a knife box mixer and then pumped into the air lift vats, a charge for each vat being 50 tons of dry slime. Each vat requires 50 ft. of free air compressed to 10 lb. pressure per minute.

At the plant of the Luipaards Vlei Estates Co., the accumulated slime after being suitably pulped (lime and lead acetate being then added), is pumped into a conical bottom vat of diameter 30 ft. and total depth 30 ft., fitted with an air lift pipe 18 in. in diameter, 26 ft. high and 22 in. above the bottom of the vat.

The air supply to the air lift is delivered through a $\frac{3}{4}$ in. air pipe, centrally placed inside the lift pipe. This supply is utilized only when the vat is full and its contents are in agitation. But, when the vat is being filled or discharged, the air lift is not operated, the pulp being agitated by air supplied through four $\frac{1}{2}$ in. air pipes outside the air lift tube. The pulp receives a total aeration of $22\frac{1}{2}$ hours, made up as follows:—Filling 9 hours (partial aeration), air lift agitation $6\frac{1}{2}$ hours, discharging 7 hours (partial aeration).

The charge of the vat is 150 tons of slime and 300 tons of water, and this charge is probably the largest single charge of slime that is agitated in the world. The vat requires about 85 cub. ft. of free air compressed to between 25 lb. and 30 lb. per sq. in. The small amount of sand, which is always present in accumulated slime, is drawn off from the bottom of the cone, as in the separatory funnel of the laboratory, and sent to the tailing wheel of the cyanide plant.

One of the most useful parts of the treatment vat is a conical diaphragm surrounding the air lift pipe, whose horizontal base is 3 ft. $4\frac{1}{2}$ in. in diameter and placed 2 ft. $5\frac{1}{2}$ in. above the bottom of the vat. This leaves an annular space about 9 in. wide between the diaphragm and the sloping side of the vat, through which the pulp passes to the air lift. This arrangement entirely prevents the banking up of material around the mouth of the air lift, and reduces the chance of fine sand collecting on the sloping side of the vat above the diaphragm. In general it enables a vat of larger diameter, and consequently of less capital cost per ton treated, to be used than without its use.

The accumulated slime after aeration treatment is pumped to the current slime plant for cyanide treatment. The following average figures from a three months' run show such highly satisfactory results as to warrant more attention being paid to this method of treating accumulated slime:—

Value of Charge... ...	2·460 dwt.
„ „ Residue	0·278 „
Percentage Extraction ...	88·7%
Lime consumed per ton ...	18·079 lb. at 50% CaO
Potassium cyanide per ton	0·233 lb.
Lead acetate per ton ...	0·121 lb.

The total working costs in connection with the treatment of this slime is 3/8·1d., which includes 1/5·3d. for collecting and pulping.

The Brown vat is also successfully used for the treatment of blacksand. At the plant of the Simmer & Jack Proprietary Mines, Ltd., an air lift vat, 13 ft. in diameter and 18 ft. deep, with a central pipe 16 ft. 6 in. long and 10 in. in diameter is used for treating 12 ton charges of black sand. The black sand is previously finely ground in a small tube mill. After the pulp of sand and water has been got into circulation, lime is added, and later on lead acetate also. Agitation is continued until all soluble sulphides are decomposed, a period of about 16 hours. Cyanide solution is then introduced under pressure into the bottom of the charge, and the solution of the gold is commenced, periods of agitation, settlement and decantation following one another alternately until

the value of the decanted solution indicates completion of the treatment. After each decantation, a centrifugal pump, connected with the bottom of the vat, and discharging externally into the top of the vat, is used to withdraw any sand which may have settled at the bottom of the cone below the air lift and put it into general circulation again.

The head samples assay about 10 oz. gold per ton, and over 97% extraction is obtained on this value of sample with a cyanide consumption of only 2 lb. per ton.

The air lift has been in use for the elevation of various liquid materials since 1892: in Western Australia it was used for pumping battery tailing in 1902; and, although F. C. Brown, as mentioned before, introduced the central air lift tube into his tall agitation tas in 1904, Mr. A. F. Crosse actually used the air lift on the Rand for the experimental agitation of slime in 1903.

The following is a list of published articles, bearing directly or indirectly upon the subject of the cyanide treatment of pulp :—

The smaller sizes of these mechanical agitators are often arranged so that the stirring gear can be raised out of the pulp, when a stoppage is imperative.

The Blaisdell agitator is one with a stirring paddle gear, driven by electro-motor and supported on a moving bridge, which can be moved over a row of agitation vats. The stirring gear can be raised out of the vat so that it can be transferred to another of the series of vats, the agitation of charges in this system being intermittent. A centrifugal pump on the bridge, also motor driven, is used to withdraw the pulp and then aerate it by spraying it back into the vat again. This system is very expensive to instal and operate, and is not suitable for a pulp that requires continuous agitation.

The Hendryx agitator has a conical bottom vat and a central tube, through which the pulp is raised by a high speed screw propellor, driven by a vertical shaft in the tube. It is expensive to operate, 18 h.p. being necessary to agitate a pulp containing 100 tons of solids. (Ref. No. 8.)

Ref. No.	Title of Article.	Author.	Publication.	Date Published.
1	Agitation by Compressed Air	F. C. Brown	*Mining & Scientific Press*	Sept. 26, 1908
2	Continuous Cyanide Treatment	J. Leslie Mennell	*Mexican Mining Journal.*	February, 1909
3	Cyanide Lixiviation by Agitation	Walter M. Brodie	*Engineering and Mining Journal.*	April 3rd, 1900
4	Treatment of Ore Slime	Andrew F. Crosse	*The Journal of The Chemical, Metallurgical and Mining Society of South Africa.*	November, 1909
5	Hydraulic Agitation in Cyaniding		*Mining & Scientific Press.*	April 20, 1910
6	Cyanidation of Concentrate	A. E. Drucker	*Mining & Scientific Press*	March 19, 1910
7	Continuous Pachuca Tank Agitation at the Esperanza Mill	M. H. Kuryla	*Mexican Institution of Mining & Metallurgy*	April 1910
8	Methods of Pulp Agitation	Lloyd M. Kniffin	*Mining & Scientific Press.*	June 4, 1910
9	Notes on Cyaniding in Pachuca Tanks and the Continuous System	A. Grothe	*Mexican Mining Journal.*	August, 1910
10	The New Esperanza Mill and Milling Practice	Chas. Hoyle	*Mexican Mining Journal.*	August, 1910
11	A new Cyanide Device	Lee Fraser	*Mining & Scientific Press.*	Oct. 15, 1910
12	A Modification of the Pachuca Tank	Amos J. Yager	*Mining & Scientific Press.*	Dec. 22, 1910
13	A Cyanide Plant constructed of Masonry	John H. Eggers, jr.	*The Pacific Miner.*	Jan., 1911
14	Mechanical and Air Agitation for Slime Treatment	A. W. Warwick	*The Mining World.*	Jan. 1911

Reference has been made before to the use of mechanical agitators, viz., those in which the pulp is stirred by rotating paddles. In these it is not possible to use a thick or very sandy pulp, and in the majority of them air is introduced into the pulp generally by an air jet at the bottom of the vat, but, in some cases, in small jets along the agitator paddles; it having been delivered through the hollow shafting of the agitator gear.

In the Just system of agitation recently introduced and in use in Guanajuato, Mexico, air from a blower is introduced through a special porous brick floor in the bottom of a vat, agitating 9 ft. of pulp above it. It will probably be found in practice that the bricks will become choked with lime salts, and, if so, cleansing would be troublesome.

Mr. A. F. Crosse's method of decanting an overflow from an agitation vat seems to have been useful to inventors.

In the Paterson vat this principle is made use of as well as that of the Brown vat, the decanted solution, free from sand, being passed outside the vat and drawn through a centrifugal pump. This delivers the pulp again to the bottom of the lift pipe, taking the place of the air of the Brown vat. The makers give $4\frac{1}{2}$ h.p. as the probable power required to agitate 50 tons of (dry) slime in a 15 ft. × 45 ft. vat. By way of comparison it may be noted that in a Brown vat of the same size at the Hacienda San Francisco, at Pachuca, 112 short tons of (dry) "slime" are agitated with about 2 h.p. (Ref. No. 5.)

Mention has been made above to the use of a shortened air-lift pipe in the vats of the Geldenhuis Deep G.M. Amos J. Yager (Ref. 12) has recently described some experiments made by the modifying of the air lift pipe of a Brown vat, as shown in Fig. IV., No. 1 showing the normal

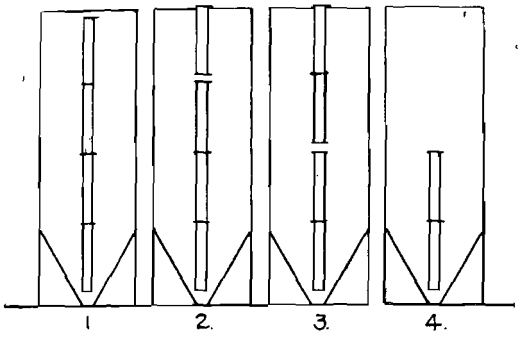


Fig. IV.

arrangements. He obtained better extractions on silver ore successively with No. 2, No. 3 and No. 4 arrangements, and finally adopted No. 4 as standard. A. W. Warwick (Ref. 14) also alludes to the shortening of the air-lift pipes in Brown vats at the Stratton's Independence Mill, at Cripple Creek, to two-thirds of their former length. The result of shortening of the air-lift pipe is the spreading out of the rising pulp above the pipe as is shown diagrammatically in Fig. V., and consequently more use is made of the air from the air lift; but it is doubtful if this modification would be used satisfactorily in the continuous system of agitation.

Vertical circulation pipes in a Brown vat, as shown in Fig. VI., with their upper orifices, which are protected by goosenecks, just below the surface of the pulp, and lower orifices near the mouth of the air lift, were suggested by W. M. Brodie. Their use was mainly the pre-

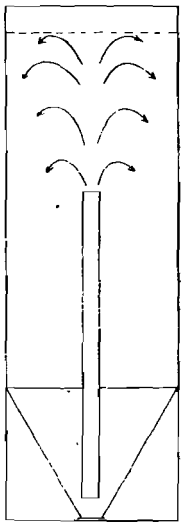


Fig. V.

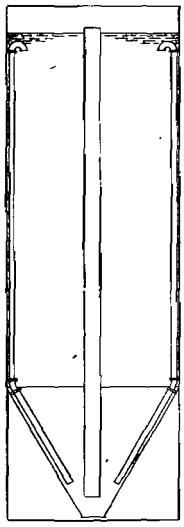


Fig. VI.

vention of choking by heavy sand around the opening of the air lift, and with them less air is required to start agitation after a shut-down. (Ref. No. 3.)

The use of air-lift vats built of masonry at Zambma, Mexico, each with two air-lifts and continuous decantation and constructed as shown in Fig. VII. has been described by John H. Eggers Ref. No. 13).

In conclusion, I beg to thank Mr. C. D. Leslie, Superintending Engineer of the Consolidated Goldfields of S.A., Ltd., for permission to pub-

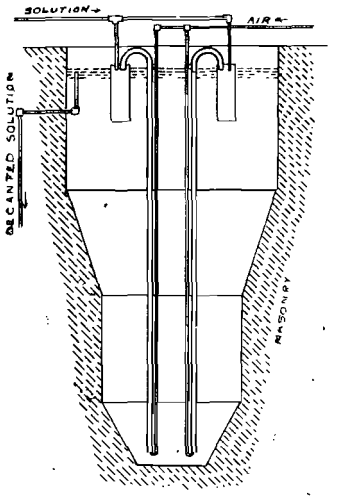


FIG. VII.

lish the working data of the Luipaard's Vlei Estate and the Simmer and Jack treatment plants.

The President: This is another very good paper indeed, and I am sure you will accord a very hearty vote of thanks to Mr. Allen. I believe it is his maiden effort in this Society, but you can see he is already an expert, especially at diagrams.

NOTES ON THE MATTE ASSAY.

(*Read at December Meeting, 1910.*)

By L. J. Wilmoth, A.I.M M., M.Am.I.M.E.

DISCUSSION.

Mr. T. Graham Martyn (*Member*): Mr. Wilmoth has made a very useful contribution to our methods of assaying cupriferous materials, but referring to the literature of the subject he says, "I . . . . as yet have found no mention of matte except as an unintentional product in the assay of such material as pyrite. In all cases the production of a matte under these circumstances is condemned, and rightly so, as it is always accompanied by unreliable results."

In my opinion the last clause is too sweeping as I have found that provided the slag is sufficiently acid and fluid to permit of clean separation of the matte from it one gets, if anything, a better recovery of precious metals from the slag when the lead button is sulphury, or a matte is formed, than when a soft button is obtained. The reasons why the sulphury button is objected to, are, I take it, first because a further operation is required for purifying the lead before cupelling which requires time and attention and involves some risk of loss; and secondly because if one is not on the look out for it there is great risk of losing part of the brittle lead or adherent matte in cleaning off the slag. It is not at all surprising to me therefore to find that, when care has been taken to produce a slag, a matte and a lead button easily separable from each other, the slag has been unusually clean from precious metal and the matte has carried no more or perhaps less than would ordinarily have been lost in the slag. The surprise to me is that it has not come under one's notice before, as an alternative to the unsatisfactory or troublesome methods in use for the assay of cupriferous materials for gold and silver. The author's results should have led him I think to carry his method a stage further, and I would suggest the retreatment of his matte by his own method instead of his resorting to scorification which in my own experience confirmed by his results, is much less reliable.

Taking the first results given in his paper and comparing the matte assay with the scorification assay a little more in detail than he has done:—

Assume that the result of the "combination" assay gives the true value, viz., 10·49% then the uncorrected matte assay recovered 99·24% of the fine gold, leaving 0·76% in matte and slag, whereas the scorification method only recovered 98·29%, leaving 1·71% in the slag or lost by volatilisation. The above 0·76% was partially recovered by scorification; taking the actual weights instead of percentages out of 7 6 mg. lost in the matte assay, 2·8 mg. were recovered from matte by scorification and 0·17 mg. from the slags. Had he treated this matte as in his later assay of copper matte I am of the opinion he would have obtained a better recovery and might very possibly have improved on the "combination assay" result. I hope that before he replies to the discussion he will find an opportunity of trying the experiment. It would probably improve the gold assay in both stages to add silver before hand if not already present in excess, say, to the extent of at least three times the gold. I have usually obtained a higher result by running down rich sulphide ores with excess of litharge or red lead, a little soda carbonate and silica or borax, if necessary, than by scorification, the latter being a method which in consequence I avoid except for reducing the size of buttons or for assaying rich alloys. It appears that the author has devised a method which may render scorification entirely obsolete and the results of experiments on the lines indicated above would be most interesting.

NATIVE FOOD SUPPLIES AND THEIR QUALITY.

(Read at January Meeting, 1911.)

By F. W. Watson, B.Sc. (Lond.), F.I.C., F.C.S. (Member).

DISCUSSION.

Dr. J. McCrae *(Member)*: With Mr. Watson's general contention that the food stuffs supplied to compounds should be subjected to control by examination, I have the greatest sympathy, but with his idea of purchasing foodstuffs (even excluding meat and fresh vegetables) according to proximate analysis, I cannot agree. The author says that up to the present very little has been done to test the articles of diet supplied to compounds. This statement reflects on those responsible for the conduct of compounds, and to some extent on the Inspectors of the Native Affairs Department. I cannot speak for the compound authorities, but I may say that samples are frequently taken by officers of the Native Affairs Department for examination. I think that if the author were fully conversant with the facts he would modify his view that little has been done; indeed, he would find that many people are alive to the question of the quality of the foodstuffs supplied to native labourers.

The author quotes two analysis of maize, and for comparison, one analysis of flour. These analyses do not teach us anything new, and from such proximate analyses it would be difficult to deduce which is the most valuable foodstuff. It is probable that the whole maize might work out as of the highest monetary value if the valuation were made on the amounts of fat, protein and carbohydrate contained in each foodstuff; but surely a great advantage is achieved by removing the large-piece fibre (in the husk) from the whole maize, even if in the process fat-content is simultaneously reduced owing to removal of the embryo. It is for reasons such as this that I regard it as inadvisable to purchase foodstuffs in general on a proximate analysis. In purchasing cyanide, it is only cyanide which is wanted—anything else present is useless or harmful. With maize meal the case is entirely different: the carbohydrate is the valuable constituent, the protein and fat, too, are useful; but, for its mechanical action, may not the fibre also be of appreciable value (provided it is not in large coarse pieces), and who would be so bold as to assert that the mineral matter is of no use? I suggest that what is of use in the examination of flours and meals is to ascertain that what is required is supplied, and supplied without the addition of other material, and that the flour or meal is in a condition fit for human consumption.

The author tells us that he has not yet examined any samples of maize meal on which any suspicion could rest. I can only think that his experience in the examination of maize meal found in compounds is not extensive. Perhaps in time the author may come across some mouldy weevil-eaten samples!

I would like here to make it perfectly clear that I am no opponent of proximate analysis: on the contrary, I would advocate the frequent analysis of foodstuffs—not in order to decide the price to be paid, but to ascertain that the foodstuff is not overloaded with water (for even on the Rand, water is a cheap adulterant), and that it approximates to the usually accepted average composition.

The author quotes five analyses of ground-nuts. Clearly samples D and E, which were adulterated, should be rejected: starches and salt are cheap adulterants for ground-nut, and the analyses quoted show, as the author rightly says, the necessity of testing supplies in order that only pure products may be accepted. I cannot, however, agree that the price of ground-nuts to be used as food should be regulated by the fat-content, freshness, freedom from disease (plant), and general condition are important factors. Although a definite amount of ground-nut may be specified in a diet-scale, it is not essential for the human frame that precisely this quantity down to the veriest grain be consumed. The case is wholly different when ground-nuts are purchased for oil extraction: here it is the oil, and the oil only, which is wanted, and the oil maker's return depends entirely on how much oil he obtains for re-sale (or for manufacturing processes). Would the author advocate that the price of milk for ordinary use be regulated by its fat-content: The butter-maker in buying milk purchases on fat-content, because it is the fat which he wants, and it is the fat which he re-sells.

The author makes reference to ground-nut cake obtained after expression of the fat. This is in itself a valuable foodstuff, because it contains all the original protein, and the fat is never fully expressed. The following analysis of a locally produced ground-nut cake will show how useful such cake is:—

Water	8·6	per cent.
Protein	42·6	,,
Fat	8·4	,,
Fibre	4·3	,,
Carbohydrate	31·0	,,
Ash	5·1	,,

Clearly a cake so rich in protein could be employed for increasing the protein-ratio in a diet,

and if the nuts are cold-pressed the digestibility of the protein will probably not be reduced.

In reference to cocoa the author quotes Dr. Hutchison. The quotation appears to me to be inapposite. The "impossible quantity" is not of cocoa, but of water. I do not think that anyone imagines that they derive any large amount of nutriment by drinking a cupful of the beverage, yet for the quantity taken (and surely that must be strictly borne in mind) cocoa is, undoubtedly, a valuable foodstuff.

In his section on Kafir beer the author says that "three per cent. of alcohol by weight is the maximum allowed." It is regrettable that the Ordinance (No. 32 of 1902) is not so specific, for it only states "three per cent. of alcohol," and leaves the question of how the alcohol is to be measured quite open. Personally, I would prefer that the measurement be by volume, because I think that that form of statement has more significance, and because alcoholic beverages are both purchased and consumed by volume.

Evidently, the Kafir beers which have come under the author's notice have been very mild. I have been less fortunate! It has been my lot to examine Kafir beers from many compounds, and I have frequently found as much as 8 per cent. by volume of alcohol, and even 11 per cent. has not been unknown. I should be glad to know if the author followed the increase in the acidity of that sample of Kafir beer in which he made observations on the increase of alcohol during 21 days. It would be of interest to know why fermentation ceased when the alcohol had reached only 3·84 per cent. by weight. One might hazard the guess that acetification was proceeding almost as quickly as fermentation; possibly the acetic acid formed had inhibited the growth of the yeast, or possibly there was some interference with the saccahrification of the starch. There can be no question that Kafir beer ordinarily has the potentiality for becoming as strongly alcoholic as unfortified wine. So long as acetification is prevented, fermentation will normally proceed until the growth of the yeast is inhibited by the alcohol formed, which takes place when the proportion reaches approximately 15 per cent. by volume. I think, however, that the author is very near the mark when he says that Kafir beer remains below the maximum allowed by law (even assuming that the law is to be read as 3 per cent. by volume) up to four days after commencing to brew.

The question of the examination of the foodstuffs employed in compounds is, undoubtedly, of vast importance, and I trust that the author's communication will do much towards directing attention to the subject. Such examinations by leading to the detection of spurious goods can result only in benefit to the native labourers, and it can do much for honest trade.

It would, perhaps, be of interest, and might stimulate investigation if the author could lay before the Society some statistics showing the extent to which adulteration of foodstuffs supplied to compounds is practised. I may say that, during the past five or six years, of the foodstuffs examined under Municipal Bye-laws in the Transvaal, the samples which have been found to be grossly adulterated, or have been condemned as unfit for human consumption, have varied between 15 and 20 per cent. of those examined.

The President: I am sure we should thank Dr. McCrae for his valuable contribution, and I am sure you will all agree with me that his laboratory note books are probably full of other data valuable to the mining industry which he might give us. I have a suggestion to make myself concerning the composition of mealies. Some two or three years ago the *Journal* published an abstract to the effect that the proteid of mealies (namely *zein*) is an unsuitable diet when taken by itself for human beings.* This may have something to do with the inefficiency of the Kafir we hear so much about.

Mr. F. J. Pooler (*Associate*): Mr. Watson's paper has opened up for discussion one of the most debateable questions with which chemistry deals, and while one is struck with the care and industry displayed in his analyses, it seems doubtful whether all such work, even where a fair standard of accuracy is aimed at, as is evidently the case here, has more than an academic value, and such remarks as I have to make will be not in the nature of criticism of results of Mr. Watson's work—such presumption, I hope, is not mine—but a general criticism of the chemical methods at present in vogue, which methods purport to give an indication of food-values of various food-stuffs.

One would ask of what importance is it to know the percentage of fats in various nuts unless one knows the average amount of energy derived from those nuts by an average human system? The human body is not a retort or a distillery, and while it may be extremely interesting to know the percentage of nitrogenous matter in maize, it is of little apparent practical importance without the additional knowledge that 1 gm. of such proteid matter represents so much actual flesh-forming power to the average man. Then the question arises, "What is the average man?" from a dietetic point of view, and it is of importance to know if 1 gm. of albumen forms the same quantity of flesh in a

* F. G. Hopkins, *The Analyst*, Dec., 1906, p. 386; this *Journal*, Vol. vii., May, 1907, p. 386; also *Journal Chemical Society* (London), Abstract, 1909, ii., 594; 1910, i., 598, and ii., 625.

head worker as in a hand and muscle worker. (It is stated in some cases that this is so—has it ever been proved experimentally?)

Again, of what practical value is "calorific values of foods," expressed in kilogram calories per 1 gm. of food-stuff burnt, unless it be first shown that in the human system the power of deriving benefit from the combustion of the foodstuffs is any way a function of the relative potential energies so determined? Quoting from "Hygiene Advanced," R. A. Lyster (M.D., D.P.H.), M.O.H. for Hampshire, I find the following on such determinations :—"The figures thus obtained are not of any great practical value, because the different food-stuffs have different uses, and are not mutually replaceable; moreover, they are not in all cases broken down in the body in the same way as they are experimentally in the laboratory." One might add: The question whether they are broken down in the body at all is an important one, and also, is the metabolism in the intestinal organs of the Kafir the same as that in the white man? This question is not, perhaps, as foolish as it seems, when we consider that the Kafir can live almost entirely on mealie pap, supplemented by a little fat, which is not generally accounted a perfect food by white authorities, and which is said to be unsuited to the needs of our Indian passive resisters.

Further, can it be shown that the human system is able to derive energy from, say, nut oil in proportion of 9·21, while an equal weight of sugar, digested by an entirely different process, gives to the system a quantity of energy proportional to 4·02 on the same scale? Again, it seems scarcely to follow that because the percentage of nitrogenous matter in mealie meal is found to be 10·28 and in pea nuts 29·69, that these values represent the flesh-forming powers of the respective foods on the same scale; for, though in general the digestive processes will be the same, it is not yet apparent that the body derives equal benefit from similar food constituents which exist under different conditions and mixed with the different proportions of other constituents, all to be digested together in the various foods.

It seems as though the chemist, working alone, is rather wasting valuable time in attempting to fix food-values by analysis, for since the vegetarian and fruitarian differ from each other and from the mixed diet man in their ideas of relative food values on the human system, and all three differ from the cannibal, of what value is the fat-extraction apparatus and the Kjeldahl estimation except that (in so far as we can accept current ideas as approximately correct) the analyst is enabled to satisfy himself that certain foods are up to average standard? Probably before any considerable advance can be made in the science of dietetics—which at present is an art of guesswork indulged in by all with a delightful empiricism that would be amusing if its results were not so disastrous on the white races to-day—it will be necessary for the chemist and the physiologist to work hand-in-hand, the former analysing and classifying proteids, sugars, &c., into various sub-branches (for it is unthinkable that "nitrogenous matter" is sufficiently informative), while the latter feeds his subjects and measures gain in weight, heat changes, waste excreted, working capabilities generated, and so on, such as, I believe, is now being done in Europe and America. Then, working together, it may be possible to tabulate conclusions of real scientific value, but even such results can only apply to the "average system," where such exists, and the results as determined must in general be subjected to immense variations depending on the particular ferments that find most congenial ground for propagation in the digestive organs of the particular subject. But it is also necessary that food-stuffs so experimented upon shall be of the same stock, for, as Mr. Watson has realised, values as quoted elsewhere differ greatly from those in use here. A comparison of Mr. Watson's results with those given elsewhere, shows considerable variations, some of which may be worth while noting, *e.g.*:

MAIZE.

Mr. Watson.		R. A. Lyster.	
Nitrogenous matter	9·54%	Proteids	14%
Fat	5·52%	Fat ...	3·5%
Carbohydrates ...	70·64%	Sugar	2%
		Starch	65%

If this deficiency in nitrogenous principles is characteristic of S. African mealies, the need for pea nuts or some other addition to a mealie meal diet is obvious (accepting current theories as correct) and the flesh-forming powers of our local mealie meal must be considerably less than those of the commonly-accepted standard in Europe. It would be instructive to find out to what these fairly large variations are due, especially in the case of our local staples—it may have something to do with local manuring, or the absence thereof. The local maize, however, has an advantage as far as fats are concerned.

A similar variation is noted in the case of cocoa, thus:—

Mr. Watson	(C).	(E).	R. A. Lyster.	
Water ...	3·28%	1.36%		3·5%
Fat ...	24·73%	40·61%		50·0%
Starch ...	7·10%	20·55%		13·5%
Nitrogenous matter ...	18·81%	17·87%	Proteids ...	15·5%
			Theobromine	1·6%

The differences in the fat contents are noticeable, though all Mr. Watson's samples fall within the rule that "No cocoa shall contain less than 20%* of fat." Is this an example of an old saying in England, that "Anything is good enough for the colonies?" The point I would make here is that it is, on the face of it, absurd to say that such and such a food-stuff is of greater value than another, unless, as Mr. Watson states, "some guarantee from the seller" is required, and yet how such can be given is not obvious when such great variations are shown by the analyses, and, finally, if such guarantee is given, what guarantee is obtainable that the food so obtained is going to do the work it is popularly supposed to do, for "one man's meat is another man's poison" is probably as true for the Kafir as for the white man.

In apologising for so lengthy a criticism of certain general principles, my excuse is that this is a matter that concerns us all, not only from a chemical but also from the every-day point of view of general health and working powers, which are, after all, closely allied to working costs.

THE SHRINKAGE METHOD OF STOPING AT THE FERREIRA GOLD MINE.

(*Read at November Meeting, 1910.*)

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

REPLY TO DISCUSSION.

Mr. G. Hildick Smith (*Member*): I am sorry to notice that except for several practical points brought forward by Mr. Johnson, nothing worth noting has been mentioned with regard to shrinkage stoping, although there are many points which could have been improved upon in the actual case described. It is very true that often the best way to learn something about anything is to start by making some foolish remark about it and then waiting for the inevitable "rise." I certainly got "rises" in three places over my remark concerning the novelty of shrinkage stoping on the Rand. On this point I stand corrected by the older and more experienced members, but it is certainly unfortunate, in the interests of mining men, that the valuable experiences of such members as Mr. Saner and Mr. Coombe are in danger of being buried with them, instead of being a guide to others who are less fortunate in the way of experience.

As to Mr. Saner's remarks concerning the question of keeping the mill going at any cost I would suggest to him that when "pushed" the best thing to do would be to mill waste, just to keep the ball rolling, but this of course is not new either, and was probably resorted to extensively in the "early days." Poor unfortunate old workers they must by this time know all the moves on the board, but apparently they do not always do the right one. As to keeping the mills going it would certainly be necessary that a certain reserve of broken ore should accumulate underground before milling was commenced, but once having started milling it would not be more difficult to keep breaking ground at the required milling rate by the shrinkage method than by the usual underhand method. As to the question of shortage of labour, circumstances might easily arise in which a good reserve of broken ore underground, broken perhaps at a period of sufficient labour, might help the mine in tiding over successfully a period of shortage of labour.

With regard to Mr. Coombe's remarks as to setts being unnecessary, it was found that setts were much handier, and made a much better job than poles and lagging, and in addition, when the ore is finally run off, the sett planks may be taken out again and used for lagging, shaft setts, etc. Throughout the building up of about 550 ft. of setts not more than two complete setts were destroyed by blasting or what not. A method far preferable to setts or poles and lagging is that employed at the Treadwell mines, where winzes and raises are so placed that by leaving pillars along each side of them the winzes and raises form the pass ways, and require no timber at all. This is one of the points which might have been improved upon. Another improvement I might mention is the use of $\frac{3}{4}$ in. round iron clamps spaced at convenient intervals, in preference to the ladders described in the original paper. Still another improvement is the replacing of the iron doors of the ore pass boxes (see Fig V. of the paper), by iron straps and planks (as shown at *e*, Fig. II. of the paper. Again, with regard to jumpers it was found that a line of pipes down which jumpers could be let fall became too easily jammed with fine stuff to be satisfactory. I just mention these points *en passant* as they may be of interest.

Mr. Coombe seriously questions my costs of box installation, but as he omits to say whether he thinks the costs too high or too low I cannot comment upon it.

Referring to Mr. Johnson's remarks with regard to the ore passes being run up with as little bend as possible, this is important from the point of view of the wear of the wall plates, which quickly become cut through by falling rock wherever an appreciable variation in the

* Ikin and Lyster, "Hygiene (Advanced)."

angle of dip occurs. Without going into much detail I do not see how Mr. Johnson makes the cost per machine shift for air, etc., come to 9s. From the data he has before him, *i.e.*, 10·48 tons per machine shift, cost of explosive 8·06d. per ton, and a total cost per ton of 3s. 1d., the cost per machine shift might be made up to 12s. 6d. for air, etc., and still allowing a sufficient margin for sundries and white and native wages.

With reference to Fig. VIII. of the paper Mr. Johnson has taken it that this figure with the dimensions given refers to an actual average bench in the stope. This was not what I intended it to represent, the idea being simply to show the position of the holes relatively to one another on any assumed size of bench, in this case 7 ft. × 5 ft. The distance between the top and bottom positions of the machine is correct, as shown, *i.e.*, 2 ft.

As to the question of using an arm on a horizontally rigged bar, it will be found better to get a smaller bench perhaps, but a bench every shift, than to try and get a larger bench and fail to get through with the drilling in one shift, due to time wasted and trouble with an arm on a horizontal bar, which is a very awkward thing to handle. With regard to breaking benches from the bottom of the stope to the top it all depends in which way the scheme is looked at, whether we consider we are working from bottom to top or top to bottom. In reality unless there is only one bench left we must follow the bench along and of necessity work from top to bottom. A combination of underhand and overhand stoping might be all right in a sound new mine, but with caved ground all around and in free milling rock as well, and also from past experience in this method as disclosed by blocks shown on the plan in the form of V's in horizontal positions, with their points together, which have been abandoned in the past having presumably become too dangerous to work, it is safer and better to break everything by the overhand method entirely. Although a little less tonnage is obtained at first yet eventually all ground broken can be run as required from a place of safety in the levels, and nothing is lost on the stulls, etc.

In conclusion, my thanks are due to those who were sufficiently interested in my paper to have commented on it.

THE MINE DUST PROBLEM.

(*Read at January Meeting, 1911.*)

By Dr. J. L. Aymard (Associate).

DISCUSSION.

Mr. E. J. Laschinger (*Member of Council*): Although Dr. Aymard's paper is a rather rambling treatment of the many factors entering into the problem of health conditions underground, the thanks of the Society and the mining world in general are due to the author. As the author is a qualified medical man of no mean standing, I read into his wide treatment of the title of the paper "The Mine Dust Problem," and more particularly because of his statements :—

> Page 312, "The medical aspect of the dust problem remains to-day very much as it did ten years ago," and "I fear we have all taken the pathology of the disease as definitely settled whereas it is quite clear that it is not so,"

that in the present state of knowledge even medical men have great strides to make in order to arrive at a rational and scientific settlement of the many problems that affect the health of miners and other underground workers.

Underground hygiene, if one may be permitted to use the expression, does not appear to be by any means a simple and well understood branch of the science of health. If this statement be accepted as true, it surely seems absurd that legislators and Government officials should treat the matter by Acts of Parliament and regulations as if the problems involved had been definitely settled by scientific research and the exact determination of natural laws. In the present state of science on this question it will not be presumptuous on my part, even as a layman, to venture on a few remarks regarding underground health conditions, not from the point of view of discussing complex or minute details, but on general aspects. As an engineer I do so all the more readily because it is the engineer's peculiar province to carry out on a large and practical working scale the recommendations of the scientist. I think the time has almost, if not quite, arrived when the recommendations of the medical man with regard to underground labour conditions should be passed on to the engineer to carry out, in the same manner as the recommendations of the metallurgist are, with regard to metallurgical and ore reduction operations. The chief difficulty is that the medical man does not seem to be in readiness to give his recommendations. The medical profession seems to be in a state of unpreparedness, but by this I do not mean to cast any slur on that profession. The blame, if any, rests on those who are responsible for the health of any section of the public in proportion as that section is numerically important. As long as any subject is not known to be an exact science anyone's opinion may be of value, whereas after science has determined laws it is generally rash for the uninitiated in these known laws to enter into discussion with those who have passed the neophyte stage.

With this preamble I will now discuss various points that arise from a perusal of the author's paper, confining myself to matters that affect efficiency in any way connected with the state of the air which the miner has to breathe.

(1) The problem is first and foremost an economic one, and as such deserves the earnest consideration of the employers of underground labour, as affecting the efficiency of an expensive machine—the man. The health point of view in industrial undertakings is only of importance in so far as it affects efficiency in production. This being the case, it follows that any recommendations regarding health conditions must be capable of being carried out in practice without excessive cost.

(2) The health point of view is of first importance to the state in maintaining the happiness, virility and general efficiency of its citizens. If any trade or occupation involves conditions which amount to rapid and unavoidable premature death to citizens it is no more than just that such occupation be absolutely prohibited by law. This is an accepted axiom of modern thought. It is, however, the belief of those competent to judge that the ordinary occupations of mining are not such that they should fall under this extreme ban of the law. Indeed, we find in certain Government Commission Reports that it should be possible to make the miners' occupation as healthful as any ordinary surface work. The very sheet anchor of progress in South Africa is the successful exploitation of the mineral wealth of the country. The vast majority of residents here live and trade on the belief that occupation in mining is not and should not be a prohibited one. I make these general remarks because it is surprising to become fully aware of the intense feeling there is in South Africa generally regarding miners' phthisis and the absurd remedies that are suggested by some people who do not even look the question squarely in the face.

(3) Since in Rand gold mining, poisonous gases from the earth or poisonous minerals are practically unknown, and since sanitary science is a more or less exact science, the serious problem to solve is to maintain the air pure and free from harmful contamination from certain well-known sources. The object should be to maintain mine air in as nearly the same condition as the air off the open veld. What I wish to point out is that the dust danger is not an isolated problem. Although an absolutely dustless mine is a practical impossibility, still a dustless mine might be more dangerous than another with dust. The question seems to be, what characteristics in dust are harmful, what amount of such harmful dust is permissible as a maximum, and by what means can this dust be rendered harmless or comparatively so. That the human system is capable of throwing off dust inspired with the air is self-evident, for any miner who has died of the effects of dust must have thrown off many thousands of times as much dust as was found in his lungs. Even if "fibrosis" is the effect of lacerations of lung tissue by dust particles, this effect is the result of but an infinitesimal proportion of the inhaled dust particles. There must be certain conditions external to the presence of the dust itself which predispose the human system to harm from dust. The chief harmful ingredients in mine air are carbon dioxide and organic emanations from the body, excessive humidity, and the products of combustion of lights and explosives. The "smell of the upcast" is wonderfully uniform all over the world. Again, very few diseases are simple in their origin and pathology and the author's paper together with other treatises on the same subject form an admission that miners' phthisis is the result of a complexity of causes. Perhaps too much has been made of the dust effects. Dust being solid is found to some extent in the lungs of the patient, gases or vapours which may by as much or more have been responsible for the disease are naturally not found in the *post-mortem* examination.

(4) The author touches upon ventilation and makes certain suggestions. Since he admits that all dust found in the lungs is capable of suspension in air currents, and since ventilation, as he refers to it, is change of air by displacement, *i.e.*, currents of air, it follows that the stronger the ventilation the more the dust is carried through the mine. As far as the dust problem is concerned, strong ventilating currents during working hours are only an additional means of spreading the danger instead of minimising it; they have also other harmful effects; it is not healthful to work in strong draughts.

(5) The dangers and expense of over-ventilation bring one to the thought of corrective measures outside the crude and obvious process of simple replacement of air. Why not correct the composition of the air in mines by artificial means? Dust and dangerous gases may be washed out or eliminated by chemical means, excessive humidity may be corrected by artificial precipitation of moisture and excessive temperature may be corrected by artificial cooling. Such expedients in combination with a moderate displacement system of ventilation seem to offer the correct solution, especially for very deep level mines with extensive workings. It is a matter of relative cost and efficiency as to how far the orthodox mechanical or natural ventilation method should be supplemented by means more scientific and complex.

Without going into a lengthy discussion as to the harmful effects of excessive humidity I might point out that it is generally agreed by competent authorities that a wet bulb temperature of 80° F. approaches the limit when a man can not do his day's work efficiently or continue to maintain his bodily health. From an extract in our *Journal* of July, 1910,* it appears that the Socialist Parliamentary Party of Belgium even brought forward the proposition that with a wet bulb temperature of only 67° F. it was proposed to limit a day's work to six hours. In order to avoid excessive humidity the practice in ventilation of the Comstock Mines† is to deliver dry air at or near the working faces in metal pipes so as to prevent saturation by premature contact with wet faces; also nearly all drains are covered in to prevent evaporation of water and saturation of the air.

(6) Over a year ago two of our local workers, Messrs. Whitford and Mills, proposed a system of removing humidity from mine air by means of mechanical refrigeration plant underground, and also the removal of carbon dioxide by a spray of alkaline solution (caustic potash or the like). The spraying and precipitation of moisture would also remove to a considerable extent dust and other dangerous gases and organic emanations.

There are, of course, other means of removing moisture and of treatment for gases, and although these mechanical and chemical treatments would involve expense it is quite probable not only that such schemes will be practical, as the alternative to over-ventilation, but also that they may have to be resorted to in the very deep levels as the only solution of the problem of keeping down excessive temperature and humidity.

(7) In view of the remarks made by Mr. Weston at the last meeting regarding the mines at Pzzibram providing up to 200 cub. ft. of air per man per minute, this would be an almost economic impossibility as applied to deep level mines on the Rand, and is only practicable in mines where the number of workers underground is small. To adopt this scheme on the Rand either extremely large special ventilating shafts would have to be provided or else the air forced underground and distributed at very high pressures at great cost.

Underground health conditions would not be improved by having a hurricane of wind blowing through the mine. What is needed in such a contingency is local purification and correction of the air to make it nearly like the open air above. According to an estimate which I have prepared we can take it that in order to provide each worker with the amount of air required by law (70 c.f. min.), the total cost is under 1d. per man per shift. The details of this estimate are as follows :—

	d.	
Power cost	0·35	
Attendance above and below ground	0·15	
Maintenance	0·10	
Working cost ...	0·60	per man shift
Interest and redemption on plant 1% per month...	0·25	
Total cost	0·85	per man shift

The cost per ton of ore mined will of course vary inversely as the useful labour of mining varies. For present Rand conditions it would be anywhere between 0·85d. per ton and double that figure, varying also as regards the capital charges depending on whether one or two shifts per day are worked. The above is calculated on single shift working.

In conclusion, I think the author deserves great credit for his work in connection with his dust catching appliance, respirator and other apparatus for bettering conditions for workers underground but I do think that so far we are only touching the fringe of the real problem in regard to miners' phthisis.

Mr. E. M. Weston (*Member*): I should like to know if the decision to put down a circular shaft on the New Modderfontein was not influenced largely by the question of ventilation?, and I should like to remind Mr. Laschinger that there are two ways of supplying increased quantities of air up to 200 cub. ft. per minute to underground workers in the deepest mines. One is by using very high air velocities or by largely increasing the size of the shaft which Mr. Laschinger regards as impossible, and the other is by decreasing the number of underground employees per 1,000 tons of ore raised. I am quite certain that we are going to do this in the future, and that with the aid of electrical haulage and mechanical means of breaking and transporting rock the Kafir will be more and more reduced in numbers. The one man drill will tend to eliminate the hammer boy who will become too inefficient a machine to be used in the deepest mines, and with the drills or Mr. Robeson's rock-pounders will need more fresh air to operate them and thus aid ventilation.

Mr. E. J. Laschinger (*Member of Council*): I am very sorry I cannot answer that. You must ask that question of a consulting mining engineer.

Mr. R. Gascoyne (*Member*): There is one point arising out of Mr. Laschinger's remarks. I believe mechanical engineers, judging from a

* From the *Colliery Guardian*, 1910, p. 1297.
† Proceedings A.I.M.E., Nov. 1909, by G. J. Young.

recent after-dinner speech, are very anxious to get below ground and see what they can do to lower costs and improve methods of mining even without employing labour. I suggest that Mr. Laschinger, seeing that machine drilling is a mechanical operation, should take into consideration whether it is not possible for the mechanical engineers to devise some means of, say, using water in connection with drilling machines so as to prevent the production of dust altogether.

Mr. E. J. Laschinger: We might do so and pick up some other kind of disease which is as bad or worse.

Mr. W. Price Griffiths (*Member*): The legislation contemplated by the Union Parliament must make "miners' phthisis" one of the most serious problems in the industrial economics of the Transvaal. Dr. Aymard's paper is, therefore, most timely, and will, I hope, promote such a discussion and an exchange of views amongst those engaged in the industry as will go a long way towards a solution. Two vitiating factors in the mine air are recognised as the cause of this disease, viz.:—

1. The fine particles of dust, and
2. The noxious gases present in the air.

The former, I suppose, is the direct cause of silicosis, while the latter by producing ill-health, and hence a low standard of vitality, increases the susceptibility of the mine worker to the disease, promotes its growth, and in general materially assists in bringing about the fatal end.

Apart from these well-known causes, it is well worth while perhaps to consider the disease from another aspect. Is there a predisposition to the disease in the constitution of some men? If so, can it be discerned on medical examination? The opinion of the medical fraternity on this point would be valuable. If there exists a predisposition to the disease which can be medically ascertained, then I consider it a national duty to exclude all men so affected from entering a mine where phthisis is known to have been produced. Such a course would probably meet with a great outcry, but the success of the industry, apart from social obligations, demands it. To find a remedy for the dust evil of the mines is a duty; to prevent those who are already constitutionally weak, and who are naturally-adapted "toy-things" of the disease, from becoming accessible to the "germs" of silicosis by entering the mines, is another duty equally as important, and incumbent on us from national, moral, and economic considerations.

Turning to the vitiation of the mine air, I consider that the gaseous part of it presents no insurmountable obstacle. This must be readily admitted if one considers for a moment the very large quantities of dangerous gas removed daily from some of the South Wales coal mines and elsewhere. On the Rand it has now become fairly well recognised that natural ventilation is inadequate for maintaining the air in a healthy state. This has naturally led to the consideration of ventilation produced by mechanical means, which has been so very ably and fully discussed before this society that there remains little or nothing to add. It is highly satisfactory to know that, its necessity being recognised, its cheapness and contribution to mining efficiency justifies its introduction. With an improved trunk ventilation, such as mechanical ventilation is bound to produce, the air at the faces of "ends" will become considerably improved.

The necessity of valuing the ore developed should ensure winzes being kept at reasonable and regular distances, and wherever the poorness of the reef driven on is such that economy requires these distances to be increased, recourse may be had to local ventilation by means of small fans driven by compressed air, which is always on hand. The ventilation of these "ends" does not depend, as the author seems to think, on the velocity of the air in the shaft past their entrances. It is well recognised that they depend for their supply of air on difference of temperature, which is the principle of natural ventilation. The ventilation of a drive leading into the side of a hill is produced by the same natural causes. To deny that the velocity assists may be wrong, but to credit it with being the active principle is entirely erroneous.

He also wrongly attributes the stifling atmosphere sometimes experienced at the working face to the slight tendency to vacuum produced by the rapidity of the passage of the air down the shaft. The ventilating pressure which produces the ventilation of one of these levels is, say, about an one-hundredth part of the whole pressure responsible for the ventilation of the whole mine. Let us presume this to be 2 in. w.g. If this be converted into barometer reading, it will be found to be less than ·15 in. Thus the author must readily see that such a small difference of pressure is imperceptible, and in the case of a level it should be one hundred times less.

The author might be interested perhaps in extending his experiment with the two tubes at right angles to one another. By placing at the end of the horizontal tube a U tube half filled with water, he may ascertain the "tendency to vacuum" produced by the rapidity of the passage of air down the vertical tube by measuring the difference in the height of the water in the two limbs of the U tube. This may be

ascertained for various velocities if a Biram's anemometer be suitably introduced. Should the author carry out this experiment, I should very much like his results to be contained in his reply to discussion. He appears unnecessarily alarmed at the stalactitic condition of the drives in places. Will he go so far as to suggest a glazed brick lining or the employment of a gang of dressers to dress down the sides and roofs? I think he will agree with me that the cost of both is prohibitive. In these days, when large crushing plants are the order of the day, haulage considerations should always ensure the sectional areas of airways being more than is actually required for the conveyance of air, and thus an efficient ventilation will be maintained at a sufficiently low water-gauge.

Contrary to the author, I certainly agree with the Commissioners as to the advisability of filling in old stopes and drives. The noxious gases of the Rand mines are chiefly heavier than air, and their tendency, when ventilation is weak, is to congregate in the lower unused workings of the mine. When a fall of the barometer occurs, and especially when this is sudden, they become a source of nuisance, if not of actual danger, by invading the nearest working places. Many miners have experienced this phenomenon. There is yet another reason. These old stopes by being open increase the sectional area, and therefore materially reduce the velocity. The tendency of the heavier than air gaseous impurities diffused in the air for conveyance outside is to separate out. In course of time these stopes would become a reservoir, not of cool refreshing air, but of deadly foul gases.

Coming to the root of the evil— the dust—we are confronted with serious difficulties at the start. Efficient ventilation aggravates it through the increased absorption of water. Deep mining produces the same effect. The dust, therefore, threatens to become more of a "thorn in the flesh" than ever. How may this be remedied?

In discussing this question it is most important to know the habits of the dust in the mines. A large quantity is no doubt carried to the surface, but this is no advantage if it has to go to other working places before it reaches the upcast. Our aim should be to provide air with no dust. I know this is as impossible as the production of a perfect vacuum, but aiming at a high ideal is the surest way to get near it.

The velocity of the air at the working places where a large quantity of dust is being constantly produced is generally slow. On this account the largest proportion of dust settles not far off. Water applied to it here temporarily damps its ardour, but it quickly revives under the influence of a dry atmosphere. Can we not make the effect of water more permanent? It has occurred to me that a soluble viscous substance dissolved in the water would materially assist in this direction. By this means the dust would still remain a captive after the water had evaporated, owing to the adhesive or "sticky" residue remaining. This solution should be applied in the same manner as water is now. All places known to produce dust should be specially attended to daily. Periodical spraying of all the air courses should be adopted, and this more lavishly at those places where the velocity of the air is slow. The effect of this would be to present a very wide adhesive area for the dust to settle on, and it should be the means of withdrawing a big percentage of dust from circulating in the air.

I regret not being able to suggest as well a substance suitable for this purpose, but that there is one which can be successfully applied is, I think, more than probable. Doubtless there are quite a few which are worth experimenting with. Anyway, until such time as we can drill holes and blast them without producing dust, it seems to me the only reasonable direction to seek a solution of the problem. We all agree with the author that the practice of leading dust-laden air through the mines is bad. Air should be portioned out to do a certain amount of work, after which it should be regarded as sufficiently contaminated, and should be conducted to the upcast by the most convenient route. With suitable doors and bratticing, with air-crossings to avoid the downcast, this can easily be effected.

Every one must realize that the solution of this perplexing dust problem will never be attained by any one effective line of defence, but probably by a dozen or more; each one appearing by itself an insignificant detail, but, added together, forming a very efficient whole. They must be accompanied by strict rules, and any infringement must be made punishable. On this account respirators, however successful they may be in supplying the lungs with dustless air, must be regarded of doubtful value. They are admitted to be cumbersome and inconvenient during exertion, and therefore no strict rules can be applied to their use, but to men who value their health they will probably prove invaluable.

It is to be regretted that the author has finally despaired of allaying some of the dust at its origin. One had become so assured of its success from Press comments that the news of failure is unpleasantly disappointing. His idea of allaying the dust, had it been successful, would have effectively trapped a great deal of

dust, and I trust his ingenuity will yet become master of the situation.

Whatever may be said of the justice of the Miners' Phthisis Bill, now before Parliament, it will have one good result, and that is, it will finally solve the great dust problem. Men engaged in the mining industry have been so engrossed in the task of solving problems relative to working costs that social duties were forgotten and disregarded. Phthisis will now become a factor in the working costs, and, like all other problems, it will inevitably suffer the same fate —a solution.

The Act may also have one bad result. It may tend to make the miner (if that be possible) more regardless of his health, and hence inattentive to the means devised for his protection, if they imply any little extra work. Whatever these means, stringent rules must be used to enforce them.

I join hands with our President in thanking the author for his paper.

PRACTICAL NOTES ON COAL.

(*Read at February Meeting, 1911.*)

By Michael Dodd (Member).

DISCUSSION.

Mr. Tom Johnson (*Member of Council*): One must agree with Mr Dodd's remarks in his paper about the waste of coal in working, also, how applicable these remarks are to our gold mining. I have in mind a mine where on a change of management taking place, the new manager started to work out the old workings which he had been told could not be worked. However, he cleared up some of the drives and started to clean out the old workings, the result being that 10% of the tonnage of the mine was recovered from these workings the value being so good as to increase the value of the total tonnage of the mine by 1 dwt. per ton and gradually increase the profits from £1,000 to £8,000 per month. This is a quite modern case. We have other cases not so bad as the one just mentioned, but bad enough, where ore of less value than average cost is left for the future, forgetting that this ore will in the future have to be mined separately at increased cost instead of a reduced cost.

In coal mining at the present time much better methods are in vogue, better care is taken about the working of the upper or lower seam first (a point we people need to look into), the more systematic opening out of the seams and the choice of system of working. Formerly, districts were very conservative about the system of working, but latterly this conservativeness is being broken down so that even in one colliery two systems of working may be seen due to change of conditions.

In the use of coal on the gold mines of the Rand there is room for much improvement. How many horse power are thrown away yearly by our winding engines exhausting to atmosphere, by using too many engines to hoist the rock, by letting compressors run nearly twice the time the drilling machines run, or more correctly running the rock drills little more than half the shift. It is some years ago since I advocated gas engines at the collieries as prime movers for the generation of electricity to be distributed to the gold mines. Then how much coal is wasted by using coal not suitable for the boilers or not altering the grates to suit the coal procurable? Some time ago I mentioned before the Society the successful alteration of grates at the Langlaagte Deep, by Mr. Denby, the engineer.

Much money is wasted on coal by taking the worth on calorific value, forgetting that it is not the heat that is in the coal, but the amount of the heat that can be used that determines the value of the coal to the plant. In the case of the mine mentioned by Mr. Laurie Hamilton working at a cost of 10d. per ton milled, one of the reasons for this low cost was that there was no contract for coal, the manager buying what suited him and the boilers in open market, the resident engineer doing the rest. If this was more general I think the results would be much better.

The President: Before we go I should like to make the complaint that so many of the mining men are not quite playing the game with us, seeing that we arranged to make special meetings for them. I hope the mine managers will come along and give us their views.

Visit to the Roodepoort United Main Reef G. M. Co., Ltd.

By the kind invitation of the Managing Director and the Management, about 150 members of the Society visited the Roodepoort United Main Reef Mine on Saturday afternoon, the 25th March, for the purpose of inspecting the new surface equipment of the property.

The party was rece ved by Mr. P. Q. Treloar, the General Manager, Mr. M. Torrente, the Consulting Metallurgist, Messrs. E. Farrar and J. N. Bulkeley, the Consulting Mechanical and Electrical Engineers, and Messrs. Roger Price, T.

Johnston, A. E. Davis, and E. Musgrave, of the Head Office Staff, under whose guidance the members were shown over the works.

The following is a description of the surface equipment of the mine :—

Kimberley Shaft Plant.—The boiler plant consists of six 250 h.p. Stirling boilers with economisers and stack, and is provided with a Locke damper regulator. The feed pumps are of the well-known Weir type.

The engine house contains :—

One Ingersoll compressor with steam cylinders 18 in. and 34 in. diameter and air cylinders 18¼ in. and 32 in. diameter, all 42 in. stroke.

One Hoerbiger compressor with steam cylinders 31½ in. and 51 in. diameter, and air cylinders 30½ in. and 49 in. diameter, all 47¼ in. stroke.

One Worsley Mesnes direct winder with cylinders 20 in. diameter × 42 in stroke, and drums 8 ft. diameter × 2 ft. 4 in. wide.

One Robey direct winder with cylinders 30 in. diameter × 60 in. stroke, and drums 12 ft. diameter × 3 ft. 11 in. wide.

The condenser for the air compressors is provided with a centrifugal circulating pump and a three-throw Edwards air pump, both driven by a small vertical engine (Allen) at 500 r.p.m.

The condensing water is passed through a natural draught cooling tower standing in a dam on the west side of the boiler house.

The feed water is passed through a vertical feed heater, where it absorbs the heat from the exhaust steam of the winders.

Headgear.—The headgear and bins are of steel, and were made locally in the works of the United Engineering Company, and erected in record time

Electrical Driving.—It may be pointed out that from the time the ore leaves the headgear bin, it is handled and dealt with entirely by electrical power until it reaches the sand dump and slime dams.

The total number of motors employed in the plant is one hundred and fourteen, all supplied by the A.E.G. Electrical Co., the whole of the switch gear being furnished by the S.A. General Electric Co.

Sorting and Breaker Station.—This is to the west of the headgear bins from which the ore is carried by conveyor belts.

Main Fine Belt.—The ore from the fine bin under the grizzly passes to a 20 in. conveyor belt driven by a 7½ h.p. motor, and running the whole length of the sorting house and delivering to the loading bin belt which leads from the breakers to the loading bin.

The main fine belt also receives the fine from the washing trommels and the de-watering trommel, which are taken by two small belts referred to later.

The coarse ore is delivered to a 30 in. belt passing under the main ore bin and also under a small ore bin on the east side, which receives ore from the shaft of the eastern section of the mine about half a mile away.

This belt, which is driven by a 25 h.p. motor, conveys the ore to a Y-shaped chute, which, in turn delivers it to either or both of a pair of washing trommels 14 ft. long × 4 ft. diameter, carried on Lea's patent rollers. Each of these trommels is driven by a 7½ h.p. motor.

The trommels—supplied by S. Sykes & Co.—have two sets of perforations ; the first, of comparatively large size, are intended to take out any fine which may have remained with the coarse ore and to drop it into a hopper beneath, from which it falls on a 16 in. conveyor belt, driven by a 1 h.p. motor, and is carried to the main fine belt already referred to.

The water for washing the ore is applied over the second set of perforations, which are of smaller size, and through which the water and the small fine washed from the ore passes into a second hopper and thence to a de-watering trommel 30 in. diameter × 6 ft. long covered with fine mesh screen. The water passes through this screen to a receiving tank, from which it is pumped up to the main launder in the stamp mill.

The small fine which passes through the trommel is received on a 14 in. belt, driven by a 1 h.p. motor, and delivered to the main fine belt.

The washed ore on passing through the washing trommels is delivered to two 40 in. sorting belts, which convey it to the chutes leading to the breakers. The waste rock sorted out from the belts is dropped through openings in the floor to a bin beneath, from which it is carried by trucks to the waste rock elevator (driven by a 15 h.p. motor), and thence to the dump.

The breakers—five in number, and each driven by a 25 h.p. motor—are of Grusonwerk manufacture, swinging jaw type, size No. 7, and deliver their product to a 30 in. belt leading to the loading bin, and driven by a 25 h.p. motor.

Loading Bin.—For filling the trucks of the mechanical haulage, a bin has been erected, to which the product from the breakers and the fine from the grizzly are conveyed by the loading bin belt already referred to.

The bin has been arranged with a view to further extension, and the head pulley of the belt is carried at a sufficient height to admit of a shuttle belt being placed on the top of the bin so as to distribute the ore to any desired length of bin.

Mechanical Haulage.—For conveying the ore to the mill—a distance of 2,900 ft.—a powerful mechanical haulage gear has been provided, to carry two-ton saddle-back trucks on a 24 in. gauge 30 lb. rail track, laid on steel sleepers. The line passes under the Main Reef Road to a point about 500 ft. west of the end of the mill. To this point also runs a steam locomotive haulage for transporting ore in similar trucks from the main incline shaft, a distance of about one mile.

The trucks are then transferred to a second haulage system, and after passing over an Avery Automatic Weighing and Recording Machine, are taken up to the mill bin on an inclined steel trestle 400 ft. long, the ore is dropped into the bin, and the trucks, after passing over two turntables, are returned down the inclined trestle, passed over a second Avery automatic weigher, and returned to the respective shafts for another load. It will be observed that by this system an exact record is provided of the weight of ore sent to the mill.

The haulage gears were made and supplied by the Austral Ironworks, the trucks by Robert Hudson & Co., and the rails and sleepers by the United States Steel Products Co.

No. 1 haulage is driven by a 75 h.p. motor, and No. 2 by a 25 h.p. motor, both of which can be started and stopped from the weighing machine station, which is connected by an electric bell system to the loading bin and to the mill bin platform.

Stamp Mill.—The mill building and bins are of steel throughout, the only wood used being for floor platforms, ladders, king posts and cam shaft pulleys.

The mill contains 100 stamps arranged back to back. The falling weight of the stamps is 1,901 lb., the cam shafts—each for five stamps—are of nickel steel, and the mortar boxes of cast steel, weighing 14,000 lb. each, the whole of the battery parts being of Grusonwerk manufacture.

The pile blocks are of re-inforced concrete, and the timber king posts are carried on cast iron stools.

Two crawls are provided on each side of the mill over the stems and the cam platform, and a third crawl runs under the cam platform for handling heads, shoes and dies.

A strong rail track runs in front of the mortar-boxes to facilitate their removal and replacement.

Each set of ten stamps has a separate line shaft, and is driven by a motor of 50 h.p. The line shafts are carried on Hyatt roller bearings, and tighteners are provided for the cam shaft belts.

No. 1 Tailing Wheel.—The mill pulp is carried in launders with a 10% grade to the No. 1 tailing wheel, which is of steel, and built on the lines of a bicycle wheel with tension spokes.

The wheel (55 ft. diameter) is carried on steel towers, which also support the steel catch box launders; these are lined with wood and with balata belting to resist the scour of the coarse sand.

The wheel is driven from the centre shaft with two sets of steel spur gear and a belted counter-shaft. A 50 h.p. motor is provided for driving the wheel.

In addition to the wheel, a 10 in. Morris steel-lined sand pump is provided to allow of repairs being made to the tailings wheel buckets without stopping the mill; the motor provided for this pump is 100 h.p.

A spill elevator of the bucket type is installed to lift and return to the wheel any drip from the launders or buckets, and is driven by a 1 h.p. motor.

A small wheel 9 ft. in diameter receives the coarse tube-mill pulp returned by the spitzlutten and delivers it into the main circuit (big wheel).

A drainage tunnel has been driven from the tailings wheel pits so as to prevent their being flooded in the event of any stoppage to the wheels; and slat doors are provided at the entrances to the tunnel to retain the sand, while allowing the water to pass. This tunnel has also been carried to the sites of the wheels for the future extension of the plant. It is connected to the pipe trench under the slime vats, and then runs out into the open on the east side of the extractor house. It may be remarked in passing that this tunnel has proved of the greatest service during the construction work in preventing the flooding of the excavations in the rainy season, thus obviating the necessity of pumping them out and the consequent delay in the execution of the work.

The pulp leaving the wheel is delivered to a pulp distributor of a novel type, replacing the usual coned spitzlutte.

The overflow from this separator is led to a separator of the usual type, which also takes the overflow from the tube mill dewaterer.

The underflow from the primary separator passes to the tube mill dewatering cone, whence the coarse sand is sent to the tube-mill.

The tube-mill discharge passes to the secondary separator already referred to, which returns to the tailings wheel any coarse sand which may have passed the mill, and delivers the overflow to the launder leading to the shaking tables.

Tube Mill Plant.—Five tube mills are provided each 5 ft. 9 in. diameter × 16 ft. 5 in. long

(No. 17 Grusonwerk Mills). Each mill forms part of a separate unit of the plant and is arranged to take one-fifth of the total mill product passing as underflow from the primary separator. Each mill is provided with a de-watering cone, and at the discharge end with two spitzlutten, where a final classification of the product is effected. The product overflowing the spitzlutte (finished product) is delivered on to the amalgamated copper plates. The underflow from the spitzlutten (coarse product) is returned to the main pulp circuit as already described.

Each mill is arranged to deliver to seven shaking tables with amalgamated copper plates, thus providing one tube mill and seven tables for each twenty stamps.

Plate House.—Thirty-five shaking tables are installed here The frames are of steel, and each is driven by a 1 h.p. motor placed beneath it so that no countershafting is required in the building.

At the lower end of the copper plate is provided a short length of wooden table with blankets for retaining any mercury or gold which may escape from the plates.

At one end of the house a strong room is provided, together with small office, clean-up plant and furnace room, so that the amalgam can be retorted and the gold smelted without going outside the building.

No. 2 Tailing Wheel.—After passing over the tables the pulp from each set of seven tables passes an amalgam trap and is delivered to No. 2 wheel, which is practically identical in design and arrangement with No. 1 wheel already described.

Sand Plant.—From No. 2. wheel the pulp passes to spitzkasten, which deliver the overflow to the slime launders, and the underflow to the settling vats.

The spitzkasten are carried on a steel tower, from which project four cantilever platforms carrying the sand launders to the centre of the settling vats.

The settling vats are four in number, each 50 ft diameter × 8 ft. deep, and fitted with Butters' distributors mounted on ball-bearings.

The overflow from the vats, together with the overflow from the spitzkasten, is passed to another separator, which returns any particles of sand to the wheel, and passes the slime on to the slime settlers.

The sand is discharged from the settling vats into trucks, which deliver to a 24 in. conveyor belt, driven by a 25 h.p. motor.

This belt delivers to a Blaisdell distributor over the treatment vats, which are six in number, each 50 ft. diameter × 11 ft. deep.

The sand from the treatment vats is discharged into trucks and taken by a mechanical haulage to the dump. This haulage is driven by a 25 h.p. motor.

Slime Plant—The slime settling vats, three in number, are 70 ft. diameter × 12 ft. deep, with a conical bottom, giving an additional depth of 5 ft. 6 in.

Four similar vats are provided for slime treatment.

The decanted solution from these vats is sent to two clarifying vats, each 40 ft. × 5 ft., and is pumped from these to a receiving vat 40 ft. × 10 ft., from which it gravitates to the extractor house.

The decanted water from the settlers is sent to a return water vat 40 ft. × 10 ft., from which the return water pumps lift it to the three mill supply vats—each 60,000 gallons capacity—standing north of the mill.

These vats are also supplied from the Saxon Dam, about three miles distant, where are installed two 6 in. Rees Roturbo, four-stage pumps, each with a capacity of 30,000 gallons per hour, and each driven by a 100 h.p. motor.

Pump House.—The pump house is arranged to take all the pumps required both for sand and slime plant, together with the return water pumps.

The pumps—each driven by a separate motor—are arranged along the sides of the building, and the motors in two lines near the centre, with a passage between them. The slime pumps are in a pit at the end of the house, sunk to the level of the pipe trench.

The motor switchboards are grouped together on a platform over the slime pumps and communicating with the platform on the top of the slime vats, from which all the valves on the vat pipes are operated.

A five-ton travelling crane—supplied by Hubert Davies & Co.—covers the whole of the pumps and motors.

The following is a list of the pumps installed:

Two 12 in. Morris Slime Pumps, with two 100 h.p. motors.

Two 8 in. Morris Slime Solution Pumps, with two 60 h.p. motors.

Two 6 in. Morris Sand Solution Pumps, with two 50 h.p. motors

Two 4 in. Morris Solution Pumps, with two 10 h.p. motors.

Two 8 in. Rees Roturbo Return Water Pumps, with two 50 h.p. motors.

One 5 in. "Encke" Pressure Pump, with one 30 h.p. motor.

One 3 in. × 4 in. triplex Gould Pump gland supply, with one 7½ h.p. motor.

Extractor House.—The extractor house is a steel building with concrete floor, and under the

same roof is the furnace room, where the zinc slime is calcined and the gold smelted; a strong room and store is also provided. The bisulphate plant is in an annexe on the south side of the house, and the small vertical boiler provided here for heating purposes is the only steam plant on the whole of the reduction works. Twenty-two steel extractor boxes are provided, each 19 ft. × 4 ft. Zinc lathe, ball mill, filter press and a small centrifugal pump for charging are also provided, and are driven by a 7½ h.p. motor through one of the few countershafts to be found on the plant.

Four solution sumps 60 ft. × 10 ft. are placed south of the extractor house.

Two large dams are provided for receiving the treated slime.

Power Distribution.—Power is supplied by the Victoria Falls and Transvaal Power Co., at 10,000 volts. It is led into the transformer house, equipped with three transformers 10,000/500 volts, and two 10,000/3,000 volts, the former being for the power supply to the reduction plant, and the latter for the Saxon Dam and underground pumping plant. The transformer house is arranged so that all the primary switchgear is placed in cells on the transformer side of the house, and the secondary gear on the opposite side, leaving an operating passage down the centre with no live parts accessible. Both primary and secondary bus bars are in duplicate, and arranged in reinforced concrete bus bar cells on the floor immediately above the switches, while the lightning arrestors are above these cells again. The secondary distribution switches are all of oil break type, and arranged in two compartments of a lean-to on the main house. Here also all bus bars are in duplicate. The circuits have all been arranged so that the power supplied to each department can be accurately metered. Recording instruments are also installed to give a graphic record of voltage and total power.

The contract for the excavations for foundations was given out on June 10, 1909, and the plant was started on August 15, 1910.
Engineering Department.

After the tour of the works, the visitors were invited to the dining-room of the mine boarding-house, where light refreshments were hospitably served. Dr. J. Moir (President), seized the opportunity to express the appreciation of the members of the pleasure and information the hospitality of the company had afforded them. The plant, he said, showed what the Rand was coming to. Two features were pre-eminent, the almost complete electrification of the plant, and its compactness, neatness, and efficiency on the metallurgical side.

Mr Treloar said the plant could handle well over 1,500 tons a day if necessary, but part was at present hung up, as the mine was not yet far enough advanced to feed it fully. The present rate was about 1,000 tons a day. In eight months' time they hoped to be doing something like 1,500 tons. Already the reduction costs had been reduced by nearly one-half as compared with those of the old Roodepoort United Mine, and they hoped to lower them still more as the volume dealt with was increased. They knew what it meant to the industry in particular and to the country in general to be able to recover from 21s. to 21s. 6d. from 5·4 dwt. ore, and to make a profit on it. He hoped, even on that basis, to raise the profits of the mine to £10,000 a month. That could only be accomplished by working on a good scale. The feature of his work at the Roodepoort United, however, of which he was most proud was that since he took charge four months ago he had only made one change in the staff. That was to say, he only brought one man with him. He had every confidence in the staff as it stood. It had done good work, and worked well together, and he was confident they would make the mine a success.

Mr. Torrenti eulogised the work of the Society as one of the greatest factors in the advancement of metallurgy and of the gold-mining industry. They enjoyed an *esprit de corps* which made for co-operation, and they did their best, first because they were metallurgists with high ideals of their profession, and secondly because they had a sense of duty to their employers.

Mr. Edward Farrar invited criticisms of the plant, and emphasised the fact that practically all the work on the plant had been done departmentally. Only some minor and special features had been done on contract.

Mr. E. J. Laschinger also spoke, and reminded the members of the Society that the old Roodepoort Mine was one of the first to adopt the cyanide process. He thought the Society should discuss the merits of the plant at its next meeting. Such visits would thereby be enhanced in value.

Mr. A. F. Crosse said he had been 22 years on these fields, and had never before seen a plant which pleased him so much. Everything was so compact and neat, and so proportioned to the service required of it. The Romans built things to last, it was true, but they overdid it. The plant they had just seen would last as long as the mine, and not too much longer.

Mr. W. R. Dowling doubted whether the saying that a plant was out-of-date before it was

completed was correct in the case of the plant they had just visited. The high stamp duty attained and the use of electrical power were all thoroughly modern. Even the shaking tables in the amalgamating house had impressed him so much that he felt he should have seen them before reading his paper on the respective merits of shaking and stationary plates.

The proceedings closed with cordial expressions of thanks to the hosts and pilots of the afternoon.

Obituary.

The news has just been received by cable of the death of Mr. JOHN HENRY JOHNSON, a former associate of the Society. The deceased gentleman, who was the father of our Past-President, Mr. Edward H. Johnson, and Mr. J. Hayward Johnson, paid a lengthy visit to his sons on the Rand some years ago, and attended many of the meetings of the Society, in the work of which he was greatly interested. The sympathy of the members of the Society is extended to Messrs. E. H. and J. H. Johnson in their bereavement.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

DUST, SOOT AND SMOKE.—The increasing attention now being given to the allied questions of smoke-prevention and the improvement of the purity of the atmosphere in all large towns renders the introduction of correct and scientific methods of recording the amount of dust and soot suspended in the air and the blackness or density of the smoke emitted from factory chimneys of the highest importance.

In the past there has been no attempt made to obtain comparative and permanent records of this kind on the part of the health or sanitary authorities of any town, and 'smoke prosecutions' at the present date are still too often based upon incorrect or unscientific evidence. The first step in any reform of the existing system is to show that more satisfactory and scientific methods of observation have been developed and are now available for general use. In the first portion of this article several methods of estimating the amount of soot and dust held in suspension in the atmosphere will be described, while in the second section of the article the newer methods of observing and recording the density of the smoke emitted from individual chimneys will be dealt with. Until the health or sanitary authorities of all large towns and cities take up the study of smoke-emission in this manner their statements regarding the degree of purity or otherwise of their towns' atmosphere must be accepted with some reserve, for there is no subject of common observation upon which independent observers are more disposed to differ than upon the comparative density of the smoke emitted from their own and their neighbours' chimneys, or the relative purity of their own town's atmosphere as compared with that of their nearest rival. The personal factor must, in fact, be removed before accuracy can be hoped for or arrived at.

I.—Scientific Methods of Estimating the Amount of Soot and Dust in the Atmosphere of Towns and Cities.—A—Rübner's Method.—The Rübner method of soot and dust determination is based upon the aspiration of a large volume of the air which is to be tested (200 to 300 cub. ft.) through a filter paper placed in a special holder. Upon the surface of this paper the soot and dust are collected. (Fig. I*a*.)

The aspiration is effected by a water-jet pump, and the volume of air is measured by an ordinary dry meter. The determination of the amount of soot and dust collected on the paper in twelve or twenty-four hours is carried out by comparison with a standard scale, since the amount is too small, even with the large volume of air used, to be determined by direct weighing. Renk has employed for the comparison a mixture of a known amount of soot and oil which, when used in a glass vessel of wedge-shaped design, gives a colour scale of gradually increasing density. This method of estimating the soot and dust in the atmosphere is now being employed by the officials of the Hygienic Institute in the city of Hamburg, the results being carefully recorded. It is stated that a series of these tests will shortly be published.

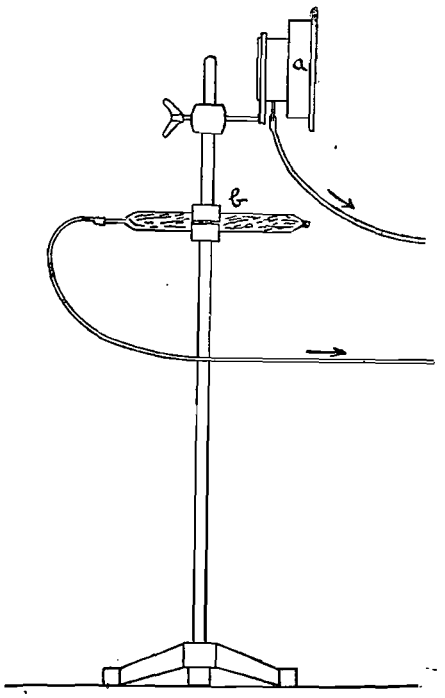


FIG. I.—The Rübner and Hahn Apparatus.

B.—Hahn's Method.—The apparatus devised by Hahn consists of a double-cylinder pump, worked by an electric motor with current from accumulators. The volume of air in this case is ascertained by multiplying the cubical capacity of the two cylinders of the pump by the number of revolutions of the crankshaft, as recorded by a counter attached to the same. The soot and

dust are retained by drawing the air through a tube containing collodion wool. (Fig. I*b*.) The estimation of the soot and dust is made by dissolving this collodion wool with its deposit of soot and dust in a mixture of two parts of ether and one part of alcohol. The final step is to compare the turbid fluid thus obtained with various standard solutions containing collodion and known amounts of soot and dust. The apparatus for carrying out this method is more costly than that of Rübner, and the use of ether adds considerably to the expense of the final comparison.

C. — Liefmann's Method. — Liefmann's method differs essentially in principle from the two methods already described, since it is based upon deposition or 'sedimentation' of the soot and dust particles upon prepared plates and not upon filtration.

The method is described in considerable detail in the Habilationsschrift' (Degree Thesis), published by Liefmann at Halle University, in 1907. Considerable space is devoted in his pamphlet to the theory of 'sedimentation' as a means of measuring the soot and dust particles suspended in the air and to the reasons why *two* surfaces are necessary (a vertical one and a horizontal one) in order to obtain correct results. The practical operation of the method only can be described in this article. The apparatus required is simpler than in the case of the Rübner and Hahn methods, and it is therefore, more easily moved from place to place or erected at any desired point of observation. A firm, three-footed base of cast iron is provided with a vertical centre rod or spindle. This carries a horizontal plate on the edges of which are fixed four thin uprights carrying a light roof or cover. Under this cover is fixed the vertical disc of glass covered with a thin film of oil for catching and retaining the particles of soot and dust blown by the wind in a horizontal or inclined direction. These particles would not be caught by the horizontal disc fixed on top of the apparatus, since only the particles falling in a vertical direction impinge on this disc. The vertical plate must, of course, be kept facing the wind or air current, and for the purpose of causing it to turn with any change in the direction of the wind a large vane is fastened to the back of the spindle which carries it. (Fig. II.) The glass discs used by Liefmann in his trials of the method were the usual

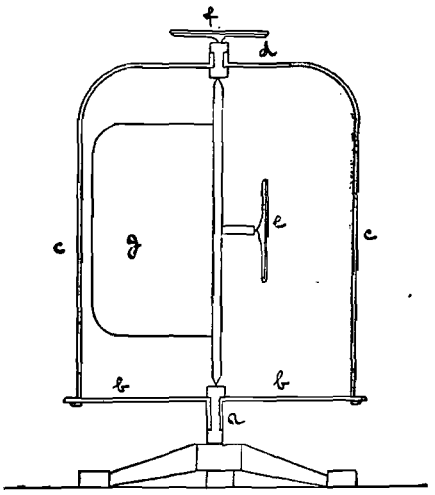


FIG. II.—Liefmann's Apparatus.

chemical clock glasses 100 sq. cm. in superficial area, and the oil film was obtained by painting their concave surfaces with a perfectly white-bleached oil by means of a small brush.

After twelve or twenty-four hours', or longer, exposure to the atmosphere the two glass discs are removed from the apparatus and newly-prepared ones are inserted, if the observations are to be continued. The exposed discs, with their films of soot and dust-covered oil, are then held over a small glass mortar, and the film is removed by aid of ether, which dissolves the oil and holds the soot and dust in suspension.

This ether solution is then evaporated by placing the mortar containing it in a warm place. It is most important to note that no light must be near this place, and that, as ether-vapour is heavier than air, it flows *downwards*. The residue is then rubbed up with 5 cub. cm. of the originally used oil, in order to yield a mixture for comparison with the standards. These standards are prepared by taking a weighed quantity of dry soot prepared from naphthalene and by mixing this carefully with 5 cub. cm. of white bleached oil in a glass mortar. These mixtures are then placed in a series of glass-stoppered tubes having flat feet, the quantities of soot used giving a scale reading from 1/10 mg. up to 5/10 mg. While the first tube of the scale is only slightly tinged a dark colour, the last tube will be perfectly black, and the difference in tint between successive tubes of the series is clear and distinct. The oil-and-soot mixture obtained from each of the exposed discs is now placed in a similar glass-stoppered tube and is compared with the standard tubes, in order to determine the amount of soot in suspension. The results must be calculated upon 5 cub. cm. of oil and multiplied by 100, in order to give the quantity of soot falling upon an area of 1 sq. metre (10·75 sq. ft.) during the time of exposure.

According to Liefmann, a test by this method can be completed in ten minutes, exclusive of the time taken for the evaporation of the ether; the accuracy is within 0·05 mg.

The following are some of the figures obtained in Hamburg in the winter of 1903-4 by this method:—

Date 1903.	Weather.	Results calculated upon one square metre and 24 hours mg.	Remarks.
Dec. 18	Bad	23·59	On horizontal disc.
Dec. 18	Bad	18·85	On horizontal disc.
Dec. 19	Bad	18·94	On horizontal disc.
Dec. 19	Bad	16·14	On horizontal disc.
1904.			
Jan. 15	Windy	188·45	On vertical disc, wind blowing from harbour.
Jan. 22-23	Good	35·29	On horizontal disc.
Jan. 23-27	Moderate	298·70	On horizontal disc.
Jan. 24-25	Moderate	76·40	On vertical disc.
Jan. 24-25	Moderate	19·11	On horizontal disc.
Jan. 27-28	Moderate	69·60	On vertical disc.
Jan. 27-28	Moderate	13·90	On horizontal disc.

The above tests show that, as a rule, more deposit is collected on the vertical surface than on the horizontal one, which is in accordance with the theory of the sedimentation process, since on most days there is sufficient motion of the air to cause a drift of the dust and soot particles in a horizontal direction. On perfectly calm days the horizontally placed surface will collect the larger portion of soot and dust.

II.—New Methods of Observing and Recording the Density of Chimney Smoke. — The need for more scientific and accurate methods of observing and

recording the density of the smoke emitted from factory and works chimneys was well brought out in the address delivered by Mr. James Swinburne, F.R.S., before the Association of 'Engineers in Charge' in October of last year. In the course of some comments upon the evidence given in 'black-smoke' prosecutions under the existing acts, Mr. Swinburne made the following pertinent remarks:—

"Smoke records are, naturally, chiefly of use in giving evidence about smoke, as, without them, the magistrate has to decide whether there has been 'black smoke,' or smoke amounting to a nuisance, from the evidence of a number of witnesses called an each side. This is rather apt to be unfair to the defendant, because the prosecution can bring evidence that, at such a time, on such a day, a witness saw 'black smoke.'

"The defendants had not their attention called to the smoke at that moment, and all they can do is to give evidence that at other times they did not emit 'black smoke,' or that their furnaces are so good that they do not believe they ever emitted 'black smoke.'

"The great difficulty is that, up to now, there has been no way of making any sort of estimate as to the blackness of the smoke. One witness may say, quite fairly, that he considered the smoke dense black, while another witness, quite as fair, looking at the same smoke, may say it is only grey or, at the most, dark grey smoke. There must always be the difficulty that the witness for the prosecution sees the smoke of which there is complaint without the defendant being able to produce any evidence about it."

Within recent years several instruments have been devised for over-coming this difficulty and for eliminating the unsatisfactory personal factor from the observation of the smoke emitted from the chimneys of factory or works. These will now be described seriatim.

(*a*)—*The Swinburne Photographic Smoke-Recording Apparatus*—A full and detailed description of this method and of the apparatus required for carrying it out was given by the inventor in the issue of *The Times* (Engineering Supplement), of March 20, 1907. The following description is drawn from that article, the 'smoke-scale' referred to being that issued by the committee of the Institution of Civil Engineers in their 'Report on Steam Engine and Boiler Trials,' published by Messrs. Clowes & Co. The principle of the method is that of photographing the smoke-scale and the smoke upon the one plate, so that any variation in the depth of tone of the negative, due to the method or time of exposure, or to the methods and chemicals used in developing will affect both the chart and the smoke equally, and a fair comparison will be obtained between the two. As Mr. Swinburne says:—

"To photograph the smoke in the ordinary way would be quite useless as a record, because the appearance of the smoke in the resulting print would depend on the kind of plate used, whether it is slow or fast; whether it is isochromatic, and used with or without a screen; how far it is developed, and whether the negative is intensified. Then the print may be on a print-out paper giving great contrast, or on a platinum one, or a bromide paper may be used, properly exposed and fully developed, or over-exposed and incompletely developed. Most of these sources of error can be completely avoided by photographing the smoke-scale with the smoke. They then both receive the same treatment, and the smoke on one print is compared, not with the smoke on another print, but with its own smoke scale. This smoke on one print may be very light. The scale is then very light, too, so that the smoke corresponds with the scale at, say, 0·4, which means that the smoke stops four-tenths, or 40%, of the light. On another print taken by someone else at the same time the smoke may look very dark; but the scale will look very dark, too, so that the smoke will still correspond with the part of the scale marked 0·4.

"The scale is made by putting a process plate in a printing frame with a paper screen in front of it. This screen has a rectangular hole to correspond with the smoke scale. The hole is covered with a paper shutter, which is withdrawn in steps. A division into eight steps is probably as many as will ever be wanted. For a process plate a candle, about 2 ft. off, will do. The exposures for eight grades will then be sufficient if the first tint is exposed 64 seconds, the second and first 32, and so on. This gives exposures of 1, 2—128 seconds in geometrical progression. There is no special reason for exposing in geometrical progression like this, but photographers always do it. The finished plate is then clear glass, except that there is a scale along one side, so as to be in the sky in a photograph. The scale plate is mounted in the camera so as to be in front of the sensitive plate. It may be loose, so that when the camera is held lens upwards it falls right on to the sensitive plate. On turning the camera with the lens down, it falls back on to stops in the camera frame. A film camera would most likely be more easily fitted.

"A cheap pocket quarter-plate camera does quite as well. Unfortunately, the shutters have generally only one speed, chosen for beginners' home portraiture, and this may be far too fast for practical smoke and sky work in this climate. In taking the chimney the camera is held so that the smoke comes close to the scale. To get good results, the smoke should be taken against a background of uniform sky. Fortunately, it is easy to get a uniform dull grey sky in this country."

(*b*) *Lowden's Smoke Tintometer.*—This instrument is comparatively simple in construction and use, and will be found of value in making the preliminary observations of the blackness of the smoke issuing from a chimney before attempting to obtain a photographic record of the same by the Swinburne method and apparatus.

For it may be noted here that the emission of dense black smoke from most factory and works chimneys is *periodic in character*, and that if a chimney is observed closely it will be found that the periods (of longer or shorter duration) when dense smoke is emitted agree very closely with the periods during which fresh fuel is being thrown upon the fires. If black smoke is then observed issuing from a chimney at 11 o'clock a.m., say, for a period of one, two or three minutes, it is probable that smoke of the same density will again be emitted at 11.15 or 11.30, according to the interval between the times of firing. The smoke inspector or observer having once decided that the chimney is an offending one under the local acts, by observation with such an instrument as the Lowden tintometer, can then proceed to obtain a photographic record of the smoke by the Swinburne method and apparatus.

Lowden's apparatus consists of a metal tube having at one end an eye-piece and at the other end two apertures. One of these is fitted with a revolving diaphragm, by means of which it is possible to bring different pieces of tinted glass in front of the aperture. The diaphragm has six holes, corresponding in size to the object aperture, and five of these are provided

with tinted glasses of varying density, marked No. 0 to No. 5, according to the scale of the well-known Ringelmann smoke chart.

'In estimating the density of smoke with this instrument the observer approaches as close to the chimney as will enable him to see through the observing aperture of the instrument a circular disc of smoke without any of the surrounding sky. The instrument is held so that the other aperture is to the windward side of the smoke column, and by revolving the diaphragm the observer can match the varying densities of the smoke as it issues from the chimney top.'

(*c*)—*Bonham and Weber's Method.* — The two methods of observing and reading the density of the smoke described above are not continuous in character, and are further handicapped by the fact that an *observer* is required to operate them.

Messrs. Bonham and Weber have devised a continuous and automatic method of recording the density of smoke issuing from a chimney which is well worth installation in every large works, since the records obtained with it can be of great assistance and value in the general over-sight of the firing of all boilers and furnaces.

A perforated ring pipe is inserted in the main flue or breeching at the base of the chimney, and from this point a small proportion of the gases passing up the chimney are abstracted by an electrically-operated fan or pump, and are forced under pressure to the point where the recording apparatus is installed. This should be near to the chimney, to prevent deposition of soot in the pipe, and the flow of gas should be rapid.

The apparatus consists of two large drums revolved by clockwork and of a long piece of white tape, chemically treated to retain the soot and dust. The tape is wound upon one drum and the free end is attached to the second drum in such a manner that the chemically-prepared surface of the tape stretched between the two drums is upwards and is exposed to the chimney gases as they issue, *under pressure*, from a narrow opening in the mouth of the pipe which leads them to the apparatus.

The clockwork and the electric pump are then set in motion, and the apparatus is left working for a period of twelve or twenty-four hours, when the tape wound upon the second drum is cut off and is removed and examined for its record of soot and dust. It is obvious that this apparatus must be enclosed in a dust-proof cupboard and be kept under lock and key, in order to obtain accurate records. A still simpler modification of this method consists in the use of the tape without any chemical treatment, the record being obtained by drawing the tape *tightly stretched* over the orifice in the smoke pipe and by employing sufficient pressure in pumping the gases to force them *through* the tape.

(*d*)—*Methods for Guidance of the Firemen and Boiler Engineers.*—A difficulty which handicaps many firemen in the discharge of their duties is that the chimney top is quite out of sight from the firing plate of the boilers, and that they have no means of knowing, unless they leave their boilers and go outside the boiler house, whether black smoke is being emitted or not. The simplest method of overcoming this difficulty in those works where it exists is to cut openings in the boiler shed and to fix silvered glass mirrors at one or two points on walls, so that the fireman can see the chimney top, either directly or by reflection, *without leaving the firing plate of the boiler house.* The writer first saw this arrangement in Gallagher's tobacco factory, in Belfast, in 1902, and it has now, on his suggestion, been adopted in many works in the United Kingdom and in America. When this plan is impracticable, owing to the large size of the works and to the number and height of the intervening buildings, that proposed by Messrs. Bonham and Weber should be adopted. The mechanism already described in sub-section C is utilised not only to supply a sample of the smoke-laden gases to the recording apparatus, but a portion of the gas is forced through a small chimney stack erected in a convenient spot on the floor of the boiler or furnace house, and the top of this miniature chimney is provided with a white background, so that the colour of the smoke and issuing gases can be instantly seen.

It is necessary, however, when this method is adopted to arrange that the volume of exit gases travelling to the smaller smokestack shall be proportioned to its smaller diameter in relation to the larger one; otherwise, the indications will be wrong. With this device, both the firemen and the engineer in charge of the boiler or furnace house will have constantly under their eyes an indication of the density of the smoke emitted from the chimney of the works. Allowance will, of course, have to be made for the difference in the diameter of the two chimneys. A column of smoke which appears brown when seen through a depth of 6 in. will seem to be much darker, and may even appear quite black when viewed through a depth of 6 ft. or more, and the firemen will have to regard the slightest sign of smoke at the top of the chimney stack as indicative of a much worse effect at the top of the works chimney. However, there is not the slightest doubt in the writer's mind that, if every boiler house possessed this simple addition to its equipment, there would be a great diminution in the amount of black smoke emitted from the chimneys of works and factories on both sides of the Atlantic, to the material benefit of the surroundings.—JOHN B. C. KERSHAW.—*Cassier's Magazine*, January, 1911, p. 251. (R. A.)

METALLURGY.

THE DEVELOPMENT OF GOLD EXTRACTION METHODS ON THE WITWATERSRAND.—"It may be assumed that members of this Association are familiar with the main outlines of the methods employed in extracting the precious metals from the Witwatersrand banket reefs.

A reference to the last paper on this subject will make clear to anyone the underlying principles of the practice then adopted, and these have undergone no considerable change. I refer to the paper read in April, 1903, before this Association by Dr. Caldecott, who, as is well known, has taken a large share in the developments of the last seven years, with which this paper chiefly deals. It will perhaps be useful to take this statement of the facts of 1903 as a base on which to develop the principal features of present-day practice, and to follow the same order of treatment of the various processes to be considered.

As it is usually necessary in mining to stope out some quantity of inter-bedded, but valueless, rock, the first operation to be performed is to remove, as far as practicable, all such 'waste' from the valuable ore. The use of sorting tables and belts for this operation is still maintained, and in 1909 the average amount of material so discarded was 14·45% of the total mined, against 14·24% in 1903. This left nearly twenty-one million tons of ore to be further dealt with. Somewhat more than half of this passed over the screening arrangements to be broken down into 1½-in. cubes in the rock breakers.

There has not been much change in the devices employed for sorting and breaking; and improvement in detail and the use of larger units have been chiefly responsible for the saving effected in cost of operation. In the transport of the crushed ore from the breaking plant to the mill bins, belts have been introduced where the distance was short, and full-sized railway trucks and engines are employed where the mills are further away, and may be seen in operation at the Simmer and Jack or the Simmer Deep.

It is proposed to use electric haulage on a similar large scale for the new mill being erected at the City Deep, and this is in keeping with the development of electric power utilisation, consequent on the formation of large power companies whose sole business is the production and distribution of electric energy.

The considerable saving effected by generating electricity in large quantity, by huge machines centred in suitable localities, will ensure that this means of reducing capital expenditure and economy in cost of operation will be rapidly extended in the near future.

Many mills are now driven by electric motors, either in large units driving 100 or more stamps, as at the Angelo, or by means of smaller motors for each 10-stamp battery.

It is now becoming usual to arrange 10 stamps to be driven from one cam-shaft, but the mortar-box for each 5-stamp unit is still retained. Breakage of these cam-shafts is still a constant source of expense and of stoppage, though this has been considerably reduced as a result of careful investigation of the quality of the steel used by means of a microscopic examination and the utilisation of the latest knowledge in production and heat treatment of this metal. It is proposed at the City Deep mill to put an inch hole right through the cam-shaft centre, and to supply extra bearings between the cams, in accordance with Mr. Laschinger's designs, in order to obviate breakage as far as possible.

The greatest changes that have taken place in the battery are in the weight of the stamp and in the size of the screen used. The average weight of the stamps running in 1903 was 1,100 lb., and no stamp heavier than 1,500 lb. was in operation. The duty, 4·91 tons crushed per stamp per 24 hours, was thus small compared with the average for 1909, which amounted to 6·79 tons. This increase of duty is not only due to the heavier weight of the stamp employed or to the fact that the weight is rendered more effective by concentration near the crushing point by increasing the weight of the head and using shorter stems; it is also due to the fact that since the introduction of tube mills it is possible to use a much coarser screen in the mortar-box. The average weight of the stamps now pounding rock within 20 miles of Johannesburg is probably not more than 1,350 lb., but stamps up to 1,800 lb. are now at work, and in the near future a weight of 2,000 lb. will be employed at the City Deep mill, with a duty as high as 20 tons a day, or possibly more, the limit being set by the coarseness of the screen employed.

In 1903 a screen mesh of 600 to 1,000 holes to the square inch was usual, and anything coarser than 500 mesh was rare. To-day screens with nine holes to the square inch are actually in use, and the highest efficiency has been determined by the Mines Trials Committee to be correlated with this aperture. The same stamps which give a duty of 5·88 tons with a screen having an aperture of 0·015 in. show the high figure of 20·03 tons with the 9-mesh screen, having an aperture of 0·27 in. Even this result may be improved by separating from the ore fed to the battery that portion already fine enough to pass the screen employed (about 30%), so that in future it is possible that a duty of 30 tons per stamp per diem may be fairly common.

It may be pointed out that the present economic limit of nine holes to the square inch is to some extent governed by the diameter of the tube mills employed. The standard size at present is 5 ft. 6 in. in diameter and 22 ft. long, but it is possible that in the future tubes of a greater diameter may be successfully employed. This will tend to increase the 'size of unit,' which is a favourable means of decreasing both capital required and working cost and may render economic the use of even coarser screens. Proceeding on these lines, a time may come when the stamp mill will be improved entirely out of existence, and the heavy rock-breakers employed will be set to crush finely enough to pass directly to the larger tube mills, or rolls might be used for the slight reduction still required.

No signs of such a progress are yet visible, though it appears merely a logical deduction from the success which has attended the efforts of our metallurgists in the direction of improvement in the operation of the tube mill plant, which is growing to be a more significant part of every new mill erected.

Coincident with the increase of coarseness of the pulp leaving the mortar-box is the difficulty of directly amalgamating the product without further crushing, and it is becoming clear that the advantage of removing the apron plates entirely away from the boxes will emphasize its necessity in the near future. This removing of the amalgamating tables from proximity to the mortar-boxes and the relegation of their duty to shaking plates, placed after the tube mills, is contemplated in many mines, though I believe it was first carried into effect at the West Rand Central mill, where it has proved entirely successful. The only possible drawback to this arrangement—that the lighter portion of the crushed ore, called slime, largely evades the tube mill shaking plates—can be obviated by special arrangement, as proposed for the 600-stamp mill being erected at Randfontein. At the City Deep new plant the shaking plates are placed under the same roof as the extractor-boxes of the cyanide plant, an arrangement which enables all gold recovered to be superintended and handled, till it is in smelted bars for the banks, in one building.

The tube mill is of course the principal novelty in the history of the reduction plant since Dr. Caldecott's paper was read in 1903, though he foreshadowed it when he wrote advocating the finer crushing of the coarse and pyritic portions of the mill pulp.

Mr. Denny first advocated the use of the tube mill for this purpose in this country, and referred to its successful use in Australia by Dr. Diehl.

At first introduced cautiously, after careful observation of its performance and of the improvement in extraction shown after the finer crushing obtained by its means, it is now used almost universally, not only on account of this increased extraction of the gold, but also because it has been proved here to be, in conjunction with the heavier stamps and coarser screens, a much cheaper method of reducing ore to the fineness demanded by economic considerations.

An endeavour has been made, on purely theoretic grounds, to show that the crushing efficiency of the tube mill is only one-fourth of that of the stamp

battery, but it is certainly true that its extended use is always accompanied by lower cost for reducing the ore to the utmost fineness now considered economically sound. Thus one or two tube mills were at first employed for every 100 stamps running, but at the end of 1909 158 tubes were in use, against 9,545 stamps erected, and it must be observed that there are many mines, whose 'lives' are now too short for the repayment of the necessary capital expenditure, where no tube mills are yet put up, and that the present tendency is in the direction of one tube to every 15 or 20 stamps; it is possible that the future will see even this proportion of tube mills increased.

A change of some importance has been made in the method employed for elevating the pulp leaving the mill for classification and subsequent treatment. Formerly tailing-wheels were almost universally employed, though even seven years ago plunger pumps were successfully used in one or two cases. The great disadvantage of the tailing-wheel, beside its high capital cost, was total lack of elasticity when the introduction of improved appliances demanded an increase in the total elevation necessary.

After considerable experimenting, a type of centrifugal pump, with cheap and easily-renewable wearing parts, was found suitable for elevating to any required height the coarsest material yet produced, and many are now in operation. Though the wearing parts may only last 20 or 30 days, the total cost, including capital redemption, of elevating pulp is less than by any other method, and any increase in elevation subsequently required is easily arranged by adding a length of pipe to the delivery column and suitably increasing the speed of the motor. One disadvantage which can, however, be overcome without very great difficulty is the lack of regular flow from the pump if the amount of the pulp to be lifted is varied by hanging up stamps or from other causes. This lack of regularity interferes considerably with the effectiveness of any arrangement for classifying the pulp and separating the coarser and heavier portions as is required for their different treatments.

The use of compressed air in the 'Pohle lift' is quite effective for elevating pulp, as at the East Rand Mines, and, though the efficiency is low, a similar method is largely used where the aerating effect is also of value in the solution of gold, as in 'Brown' agitators or similar apparatus.

The method of separation by means of cones has in the last seven years largely displaced the use of spitzlutte and spitzkasten, formerly chiefly employed for this purpose, and the introduction by Dr. Caldecott of a diaphragm partition near the bottom of the cone enables a much steadier and more uniform produce to be returned to the tube-mill plant for re-grinding, and increases the efficiency of the apparatus for other separations required. The underflow may be thus arranged to retain even less water than is suitable for the best tube-mill work, so that a second set of cones for removing the excess water may now be dispensed with, and any required adjustment may be accurately made by adding the amount of water experiment has shown to be requisite. A further advantage is found in the increased size of discharging nozzle that may be used when diaphragms are suitably employed. This avoids the frequent choking up by small pebbles of the smaller nozzles necessarily employed with less efficient arrangements.

The increased efficiency of the cone used in place of an inverted pyramid is largely connected with the fact that it is always arranged to overflow all round the circumference, thus ensuring greater tranquility for settlement and a smaller height of overflow. The spitzkasten has been invariably used with an overflow over one edge only, though of course it would be quite simple to arrange it to be used, similarly to the cone, with central inflow and baffle, and overflow over all four sides, where only wood is available for construction material, and in some other cases such an arrangement would present considerable advantages.

It has always been understood that successful treatment of crushed ore by cyanide solutions depended not only upon the ultimate fineness attained, but also upon the completeness of the separation of the leachable sand from the finer slime which tends to clog up the material to be percolated and to render the access of the air required more difficult.

By the older methods, still largely in use, it is found impracticable to separate sand containing less than 3% to 5% of slime, and even this efficiency is rarely attained. This difficulty is incidentally largely obviated in the method of collection by means of the Caldecott table. This appliance consists of a slowly-moving table with a peripheral band from 1 ft. to 3 ft. wide, upon which the thick pulp, containing less than 30% of moisture as separated at the underflow of a deep cone provided with a diaphragm, is allowed to fall. This band is provided with a filter-bottom, under which a vacuum is maintained by suitable air and water pumps. By these means the excess moisture is sucked out, and the dried material, with about 12% or less water, is continuously removed by a plough-scraper just before the point where the new material is being deposited. A few inches of sand are allowed to remain to assist the filtering medium, and this has to be removed at intervals when cleaning is necessary. The overflow of the separating cone employed carries away the greater portion of the slime, but in some cases a second cone is employed in which the underflow of the first is washed in clean water, and by this means less than 1% of slime is allowed to remain with the sand. The clean sand removed from the table is washed by means of a stream of weak cyanide solution to a centrifugal pump, and thus delivered to the collecting vats through Butters distributors, a large portion of the gold getting dissolved in transit. The treatment of the charge may be completed in this collecting vat, as is done at the East Rand Mines, but some little advantage is found in transferring it to one of a series of second treatment vats, as at the Simmer and Jack and elsewhere.

The advantages gained in the cleanness of the sand, and the fact that it is in contact with cyanide solution a few minutes after being crushed enable a considerable saving in capital expenditure to be made in the vat capacity required. Incidentally also an increase in the tonnage of slime separated this way, which may amount to 8% to 12%, permits a saving in working cost due to the fact that slime can be more cheaply treated than sand.

It is probable, as the greater part of the advantages of this method of collection are really due to the increased efficiency of separation, that its principles will be more largely availed of in the future, either by increased use of the whole apparatus or by other means of dealing with the same problem.

A radically different method of solving this problem of most economic treatment of the crushed ore

is that being tried at the East Rand Mines under the name of the Arbuckle Process. This involves the treatment of the pulp as a whole without separation of slime from sand, and thus solves the difficulty by the method of evasion. The pulp, mixed with about three times its weight of weak cyanide solution, is agitated by means of compressed air in deep vats with conical bottoms, circulation being maintained by means of the 'Pohle' air lift principle. About 20 hours is required for a sufficient solution of the gold, when the whole pulp and solution are transferred to an ingenious separating apparatus, the solid portion with as little as 25% moisture is sent to a further apparatus in order to wash out all the dissolved gold, and the clean solutions are sent to the precipitation plant.

The possible drawbacks are extra cost for power for air compression, and for the mechanical arrangements involved, and the inherent defect that the light slime, which really needs only about one hour's treatment, must remain, undergoing needless circulation for the full time required by the most refractory portions of the ore. The general decrease in working costs, owing to larger units, greater skill in handling material, and other important reasons, render it probable that in future this method, perhaps modified in the direction of greater simplicity in the mechanical details, which promises lower capital cost for plant, which is very suitable for the finer grinding, gradually coming into the region of practical economy, and which evades the difficulty of successful percolation with extremely fine sand, will in this country become of as great importance as very similar methods in other parts of the world. It is possible that, as in Australia and elsewhere, some method of mechanical filtration may be found suitable for extracting the dissolved gold from the material treated, and already on these fields the vacuum filter is undergoing successful trial at the Crown Mines, where, however, it is in use for handling slime only. It is yet too early to form an opinion as to whether the more compact apparatus will affect the abolition from future plants of the enormous vats now used for the slime decantation process. It is certainly generally felt among metallurgists that the large and expensive plants used for treatment of slime by the standard process offer tempting opportunities in the matter of reduction of capital cost, but so far it is the general experience that a more than countervailing increase of working cost is involved. When it is considered that a shilling will cover all cost by the standard methods on the large scale now usual, with an extraction of 85% to 89% of the total gold, it is clear that the field for economy, either by increasing extraction, where residues are now worth 1s. per ton or less, or in decreasing working cost, is not a wide one. The fact must also be borne in mind that in all feasible methods so far suggested the large vats employed for collection of the slime are still necessary, and even those for solution of the gold are either still required or a more expensive treatment in smaller vessels must be substituted.

With reference to the precipitation of the gold from the working solutions, it may at once be remarked that little change has been made during the last seven years. As, however, this work is performed at a cost of less than one penny per ton, and as the value is brought so low as 0·01 dwt., here again the field of future economy is restricted. Experiments have been made with zinc dust, in place of zinc shavings, and the results obtained so far promise economy of space, less gold locked up, and greater security of the bullion produced. Certain difficulties in the clean-up may possibly be evaded, and it is this weak point in our present methods that leads to the pious hope that in the near future the plan of using zinc dust and filter presses may have a brighter prospect.

A glance at the whole series of improvements effected in the past few years will render evident the fact that progress has been almost entirely in the domain of mechanics, and that nearly all proposals for the future emphasise the reduction in capital cost, and in some cases this is pressed so far as to partly ignore the fact that the increased working cost involved is more than counterbalancing. The origin of this state of things can no doubt be referred to the fact of the demonstrated economy in providing a much greater ratio of reduction capacity to mine area, and to the fact that much larger areas are now worked under one company. These facts combine to render necessary very huge capital sums, and all possible means of reducing the amounts necessary are so much the more welcome.

The most interesting question at the present time refers to the disposal of the residue after extraction of the gold has been pushed to its economic limits. Up to the present time nothing has been proposed or done with the slime but store it in dams, where it is useless, unbeautiful, and a source of expense. The coarser portion of the pulp, that is the sand, is now being sent back direct to the mine at the Simmer and Jack, which pioneered this improvement, and at several mines in the Central Rand sand is being removed from the old dumps and sluiced into the worked-out stopes below. This method of sand-filling was inaugurated this year at the Village Deep, and the success obtained is causing the adoption of similar methods on many of our leading mines. The disadvantages of using old residue for this purpose are the double handling of material involved and the expense of supplying lime to neutralise the large amount of acid and acid salts generated after long exposure to the atmosphere. On the other hand, when using sand directly from the treatment vats, the cyanide present must be effectually destroyed in order to avoid generation of prussic acid gas in the underground workings. It is fortunate that several methods of performing this work are now available. Dr. Moir and Mr. Gray have suggested the use of sulphate of iron and an alkali, and if this method is found sufficiently safe in practice it will cost an insignificant amount for the chemicals necessary. At the Simmer and Jack permanganate of potash has been found to completely destroy the cyanide, so far as even delicate indicators can determine, at a cost of about one halfpenny per ton, and experiments with an alkaline solution of bleaching powder, which promises even lower cost, are now being made.

It is certain that by some of the means suggested above it is perfectly feasible to solve the question of direct return of sand to the mine, but at the present moment there is little hope of rendering the slime available for underground support, and though the future may see the disappearance of the dismal, dusty sand dump, the permanent presence of the unsightly slime-dam seems an assured feature of the landscape of the Witwatersrand.

Since so little progress, except in details, has been made in the chemical side of our work, it is much to be deplored that we have no University in the State where the research work, so necessary for imparting life to the dull routine of teaching, and so urgent for a correct understanding and development

of the practice of gold solution and recovery, can be carried on; and it is to be hoped that this defect may in a short time be remedied.

In conclusion, the interesting observation may be made that in no period of its history in the past has so long a vista been possible as now of the future life of these, the most important goldfields yet discovered."—HENRY ARTHUR WHITE.—*The South African Journal of Science* (S.A. Association for the Advancement of Science), January, 1911, Vol. vii., p. 96. (W. R. D.)

LEAD ACETATE IN THE CYANIDATION OF SILVER ORES.—"I should like to suggest for the consideration of fellow-workers in the cyanide field that the advantages of the addition of lead acetate to cyanide solutions for the treatment of silver ores is considerably over-rated. It seems to be taken as an axiom at some plants that the use of a lead salt is almost as essential as the use of cyanide, and it is accordingly added as a matter of course; in some instances without any trial to ascertain its exact usefulness. Again, if on starting a new plant, working experiments are made with lead acetate and the advantage of its use demonstrated, the question is usually considered permanently settled, and is not subsequently reopened. In the course of a wide experience in the cyaniding of silver ores, I have found many cases where lead acetate appeared to have no effect whatever on the solution of the silver and sometimes even a deleterious effect on the extraction of the gold. In some instances the slight gain in silver extracted would be counterbalanced by the gold loss, leaving a net loss on the treatment amounting to the cost of the acetate used. I need hardly state that the above conclusions were arrived at by averaging the results of many parallel tests. Another phenomenon I have had occasion to notice is this: that sometimes on starting a new plant and making careful experiments with parallel charges the advantage of lead acetate seems abundantly proved, and yet if six or eight months later the acetate be discontinued for several weeks to give time for the elimination of lead salts from the stock solutions, and then the foregoing tests be repeated, no advantage will be indicated. I remember that two or three years ago a certain cyanide superintendent who had just started a new plant wrote me enthusiastically about the splendid profit he was making by using lead acetate; but when eight or nine months later I found the opportunity to put the question 'Are you absolutely sure you are gaining anything by the use of acetate?' his reply was "No, I am not sure, but I am trying to find out." I ascertained a few months later that he had discontinued it.

The use of lead acetate is based on an assumed presence of soluble sulphide in the solution due to the reaction, $Ag_2S + 4KCy = K_2S + 2KAgCy_2$

This reaction is no doubt correct as far as it goes, but is probably little more than momentary, and represents merely a stage in the process. The K_2S present, as it is in such extreme dilution, is very sensitive to oxidation, and except in the case of ores having a marked reducing action there will probably be sufficient dissolved oxygen in the solution to oxidize it all at once. When making some experiments on a large scale with Crosse's regeneration method, part of which consists in adding Na_2S to a sump solution to precipitate the zinc as sulphide, I noticed that solution which after treatment would yield an unmistakable brown coloration, on addition of a lead salt, was, by the mere process of decanting into another vat, entirely purified from any such visible reaction. I have never yet succeeded in finding soluble sulphide in any solution used for cyaniding a silver ore, though KCNS is almost invariably present. In view of these facts, therefore, I am led to think that with silver ores of average cyaniding value which have not a reducing action on the solution other than that due to Ag_2S there will usually be sufficient oxygen present to satisfy all requirements for the solution of the silver without excessive aeration or addition of foreign chemical salts.

In the case of ores that tend to impoverish the solution in regard to oxygen, the addition of lead acetate may be more or less beneficial. I say 'more or less' because, as stated above, I have found it 'more' beneficial in laboratory tests and and when starting a new plant, and 'less' beneficial, or not at all so, when the same plant has been running for a number of months. The explanation of the fact, I believe, lies in the presence or absence in the treatment solution of zinc absorbed during precipitation which would react with soluble sulphides in the same manner as lead acetate. Thus, when the solutions are freshly made up the acetate added will perform the function ascribed to it, but after they have been in use long enough to accumulate a considerable amount of zinc, the latter acts as a protective, and the effect of the lead salt ceases to be apparent. I was led to this explanation by a curious experience related to me by James S. Colbath, manager of El Rayo Mining and Developing Co. When working a certain stope in the mine he noticed a serious falling off in his extraction, coupled with a disorganization of his zinc precipitation. At the same time the zinc in the solutions gradually diminished to the vanishing point, as evidenced by the cyanide titrations. The readings for 'free' and 'total' coming closer and closer till they coincided. He at once suspected soluble sulphides caused by a peculiar susceptibility of the sulphides in the ore to rapid decomposition, and of course applied the stock remedy, lead acetate. The improvement, however, was very slight, and his gold extraction suffered somewhat from its use, so he began to argue from the unwonted absence of zinc in solution that it would probably be beneficial to add the latter from outside sources, which he did in the form of zinc cyanide manufactured on the spot from zinc sulphate. The effect was immediate and remarkable. As soon as zinc was present in excess the extraction and precipitation became normal and no further trouble was experienced. The outcome of these observations seems to be that lead acetate is not a panacea for all the ills of silver cyanidation. In the majority of instances where I have made careful comparative tests I have not been able to find that it was of the slightest aid in the extraction of the silver. I do not deny that there are cases where its use is indicated, but I maintain that such advantage cannot be taken for granted and must be verified from time to time. I am not prepared to offer any explanation of the harmful action of lead acetate on the extraction of the gold, but I have noticed it sufficiently often to accept it as a fact, at least in some cases. As stated above, Mr. Colbath also noticed the same thing, so it is a point worth looking into."—E. M. HAMILTON.—*The Pacific Miner*, Sept., 1910, p. 340. (A. R.)

CYANIDE EXPERIENCES IN NORTHERN MEXICO.—"We have experimented with various types of

grinding mills, and are convinced that, for usual conditions, the best combination is single-unit stamps, Chilian mills, and small-diameter tube mills. Fuller Leigh mills have low capacity when grinding dry. The results on one installation showed 2½ h.p. required per ton per day, whereas the standard arrangement mentioned above will produce the same results with 1½ h.p. per day. Grinding wet, the mill has a prohibitive consumption of iron in wearing parts.

After careful observation of the 8-ft. conical mill under working conditions, we have been able to find no sorting of the pebbles to produce the claimed 'sledgehammer-tackhammer' effect, namely, an arrangement of the large pebbles at the large end and small ones at the discharge end. For producing slime it has about the same efficiency as the tube mill. A test conducted on the same lines as those described by V. B. Sherrod in the *Trans. del Instituto Mexicano de Mines y Metalurgia* gave the following results on a hard siliciouis ore. The feed contained 30% moisture in each case and the discharge 55%, 200 mesh :

	Kgs. per min. through 200-mesh per h.p.
Allis-Chalmers—	
4½ × 16-ft. Tube Mill	·30
Hardinge Conical	
Mill 8 ft. diameter ...	·306

As a competitor of the tube mill we believe that there is no advantage in the conical mill, but it is probable that with coarse ore, say with 4-mesh, there is a profitable field for the conical mills. This is as it should be, because the forces of impact and sliding increase with the increased diameter of the mill, and for efficiency should expend themselves in breaking larger particles of ore.

My experience is that in nearly all cyanide mills the best opportunity for increasing the efficiency of the work will be found in the concentrating department. Usually little care is taken to effect a clean separation, still there is no ore which yields the values under the same conditions for concentrates as for concentrate-free ore. I cannot find that many companies are systematically determining the portion of the value in the tailing which is still locked up in the sulphides after treatment. The results are usually surprising, and justify the making of a daily assay of concentrates in the tailing.

The use of weak sulphuric wash for removing cyanicides from concentrates before treatment, as worked out by Hutchinson at the Goldfield Consolidated mill, ought to increase the field for cyaniding concentrates in Mexico, where in some cases the process is prohibited by a cyanide consumption of 60 pounds per ton.

For filter-pressing slimes the Butters and the Moore are the only machines doing good work on high-grade ore in the northern part of the Republic, but the Oliver continuous rotary filter has a useful field on ores up to 15 pesos per ton in value. It is limited to this figure, because with this machine it is hard to get a satisfactory washing of the cake to remove the dissolved values. This is especially true when the filter is used in connection with pneumatic agitation, as rapid deposition of lime takes place, which rapidly decreases the efficiency of the washing. Acid treating must be conducted with care in order to secure good results, and the tailing must be washed each day to determine the amount of dissolved value contained. Daily washing of the tailing to determine the work of all classes of filters is essential for consistent work, and this is a point that is neglected in many plants.

At one plant where the Oliver filters are installed we found it necessary to maintain the protective alkali at 0·1% CaO to prevent zinc cyanide from accumulating in the solution. The pulp is agitated in Pachuca tanks, and under these conditions it is necessary to acid-treat the filter canvas every third day in order to keep the dissolved values in the tailing below 3 oz. silver per ton.

Acid treating usually takes about two hours' time. It is first necessary to scrub the canvas to remove all the slime adhering, and then apply the acid through the wash-pipes, scrubbing the canvas constantly in order to bring the acids in contact with all parts. It usually pays to save the spent acid as it is discharged from the pump. When all effervescence ceases we wash the canvas with water.

Difficulty has been experienced in keeping the pulp in agitation beneath the revolving drum. Small air jets have been used, but have the bad feature of introducing a large quantity of carbon dioxide, which continuously deposits calcium carbonate in the band around the drum of the jet. Agitators of the Trent design are now being used, and give good results. We are at present experimenting with a pulp box of concrete with small clearances between the sides and the drum, and the feed of pulp will be at the bottom, so that the current will serve to keep the sand in agitation. It is necessary to have the sides steep, so that the sand will not accumulate at any point and stop the drum.

Using either zinc dust or shavings for precipitation, there is a great economy that is not appreciated in most cyanide plants. It is usual to hear cyanide men speak with pride of their zinc-box work in getting tailings that contain only a trace of gold and silver. But I claim that this implies an unnecessary consumption of zinc, because it is really only necessary to have a small portion of the total solution barren, namely, that portion which is necessary to wash the filter-press cake. This should never exceed one ton per ton of ore treated. Still some of the largest plants in Mexico are precipitating several tons of solution for each ton of ore, to traces. The following table will show comparative results on a month's run made at the Dolores mill :—

Class of solution.	Tons.	Value in pesos per ton.		Lb. zinc.	Lb. zinc per ton solution
		Before ppt.	After ppt.		
Strong agitated solution ...	3,757	28·00	1·50	1,368	0·37
Mill solution					
(1) Press barren solution ..	3,359	8·00		1,813	0·51
(2) Battery ...	5,174	8·00	1·50	880	0·17

Tons of ore treated, 3,174.
Total zinc per ton, 1·28.

Zinc dust is particularly efficient when used in this way, so as to leave about an ounce of silver in the solution. It is then possible to cut the zinc so closely that only 5 per cent. of the metal will be left with the precipitate of gold and silver. The product is then easily melted, and affords a high-grade bullion.

We have a method for using the short zinc that is produced in the zinc boxes which may be of interest. The short zinc as produced is washed in a trommel and stored under solution. It is then fed in small portions to a 4-ft. agitator, into which a small stream of ·2% free cyanide solution enters at the

bottom and overflows at the top of the tank. This solution contains about 10 oz. of silver and gold per ton, which is partly precipitated by the short zinc, and then flows to the zinc boxes to be completely precipitated. It is thus possible to do useful work with the short zinc rather than by using acid to dissolve it or even the roasting and melting process which is so well carried out at the Dos Estrellas mill, but which requires a large amount of fluxes.

For ordinary melting of precipitates we find that a flux calculated as half of Na_2O, ZnO, SiO_2 and half Na_2O, ZnO, B_4O_6 is the least expensive. Here ZnO represents all the impurities The average portion of fluxes to form the slag are $6\frac{1}{2}$% borax glass and 4% dehydrated sodium carbonate. The silica costs nothing, and is usually about the same amount as the borax glass. Percentages here are based on the wet weight of the precipitate. At times the slag contains 50 oz. of silver, but all of the slag is milled and any shot metal recovered.

A by-product of matte from the bullion smelting is being treated at the Dolores mill by an interesting process. It pays to give a rough treatment, because the matte assays 150 oz. gold and 6,000 oz. silver per ton, and in the quicker realization of money value is a considerable advantage. The treatment consists of melting the separate portions of matte together in a graphite crucible and oxidizing the sulphur by a stream of compressed air introduced into the charge in the crucible in a graphite tube. This requires about two hours' time, although all of the matte is not oxidized, and the remaining part assays about 1·5 oz. gold and 1,000 oz. silver per ton. This is granulated in water and shipped to the smelter. The bullion separates by gravity and is easily detached from the low-grade matte."—L. M. KNIFFIN. —*The Pacific Miner*, Sept., 1910, p. 313. (A. R.)

MINING.

COAL MINING IN NOVA SCOTIA.—"A new era in the history of coal mining is rapidly developing in Nova Scotia, and the hand-pick, wherever the conditions are favourable, is being supplanted by machines for under-cutting coal. Where coal-cutting machines cannot be used, improved methods of mining have been adopted which tend to greater economy and better results.

The larger coal companies, such as the Dominion and the Nova Scotia Steel & Coal Companies, having coal-fields lying at easy angles and, therefore, favourable to machine mining, have equipped their collieries with the most modern and improved coal-cutting machines, and in common with many of the smaller coal companies, have adopted the latest methods of handling, cleaning and assorting coal on the surface in preparation for the market.

The principal reasons for the installation of mining machinery are—to reduce the cost of production, to increase the output, and to obtain the largest percentage of lump coal.

The introduction of machinery into the coal mine has not reduced the number of men employed, but as in other trades, the stimulus of labour-saving devices has greatly helped the industry, and a scarcity of miners rather than a surplus now exists.

If we make a comparison of, say, one machine colliery with a hand-pick, or a number of hand-pick collieries, the advantages of coal-cutting machines will be fully demonstrated. Dominion No. 1 colliery, Cape Breton, during the year 1909, employed underground 463 men. The output for the year was 544,499 tons. The hand-pick collieries of Cumberland combined, only produced 508,202 tons in the year 1908, yet they employed underground 1,806 men.

While mining machines are of great value in coal getting, they are a most important factor in the rapid development of the collieries.

The introduction and extended use of coal-cutting machines in thin 'seams,' or beds have made the working of these profitable, and placed many of them on a competitive basis with the thicker seams. The British Royal Commission on Coal Supplies in their report stated that in the year 1900 about 17·7 per cent. of the total output of the United Kingdom of Great Britain was obtained from the seams of less than three feet thickness. If seams less than three feet can be worked successfully in any other country, there is no reason why they should not be worked in Nova Scotia, conditions being equal.

Coal mining being the chief industry of Nova Scotia, the province's greatest asset is her mines and minerals. These should therefore be protected, as they are the heritage of the people. Along the eastern seaboard of the County of Cape Breton there are valuable coal-fields. Some of these fields contain five or six workable seams overlying each other with a variable thickness of strata intervening. The question arises, which of these seams should be first attacked? In my opinion, if the market affords it and other conditions permit, the descending order of working is the most practical. Working the top seam first is beneficial to the seams below, forming a cushion over them, especially over the thick seams. By thus relieving the pressure above, the proportion of slack in the underlying seams is reduced. The liability of a crush is averted, and the risk of men's lives is lessened by the lessening of the vertical pressure of the strata. Fatal accidents have often resulted in times of 'creeps' or 'crushes' in coal mines when sudden pressure has been exerted, and large lumps of coal suddenly forced out upon the miner without any warning whatever.

By working the lower seams first, all the strata right up to the surface is disturbed and broken; and while this settling process is taking place, the upper seams are damaged and sometimes wholly destroyed.

The possibility, therefore, of danger to other seams must be carefully considered in opening up a coal-field containing seams overlying each other, and the method least calculated to seriously affect the commercial value of the coal-field should be adopted. The tendency of past years seems to have been to work the most profitable seams first, but it has had very baneful effects, and has not always been the most profitable system of mining. Of course, I am well aware that competition plays a large part in determining what seams must be opened up first. But if keen competition forces the opening up of the thicker seams at the beginning, might it not be well to consider the workings of thin seams at the same time? This would be a decided improvement on the methods of mining coal, and wherever tried in the province has met with good results.

The method of mining is generally determined after a careful study of the thickness of the seams, the nature of the strata above and below, the angle of the seam and the texture of the coal.

The method of mining coal in Nova Scotia is generally either bord-and-pillar, or long-wall.

The conditions favourable to long-wall working are thin seams which freely part from the roof, and tender strata overhead. A band of 'dirt' which is easily separated from the coal, makes material for stowage with which to fill up the goaf when the coal

has been extracted, so that the roadways, which sometimes must be naturally very long, can be supported. Particular pains are, however, taken to support the roadways and packs, or pack-walls are built on each side of the roadway from the floor to the roof. These are usually of stone blasted from the roof or the pavement. In the long-wall method, ventilation is simple, as the full current sweeps along the working-face without the aid of brattice, and free from obstructions met with in the bord-and-pillar system. In very deep workings the vertical pressure is such that long-wall is the only system that can be adopted.

When conditions are not favourable to long-wall the most practical system is the panel system. When passages are being driven to win the coal, these should be continued back to the boundaries. At the same time headways or balances should also be driven from the main levels to the rise, a distance of four or five hundred feet, and rooms turned off to suit the cleavage of the coal. As soon as the rooms have been driven through to the adjoining balances, pillars should be drawn at once and all the coal extracted. This would save the expense of ribbing, would relieve the pressure and prevent a 'crush' or 'creep' in that district, and, as usual, where work is concentrated, the cost would be reduced to a minimum and larger profits obt ined.

The size of pillars varies in different collieries and is determined by the nature of the coal, the thickness of the seam and of the cover over it. Small pillars often induceed 'crush' and 'creep' and brought great loss to the owners; but this lesson has been so well learned in Nova Scotia that the present tendency is now all the other way. Especially must large pillars be left where the roof and floor are very strong and the nature of the coal is soft and tender. Where these conditions are the opposite, a smaller pillar may be left; but at no time should any risk be taken, and the margin of strength in the pillar be less than that stated by leading mining authorities.

As passage-ways are driven to win the coal, and districts opened up, all coal should he extracted, except where it is necessary to leave pillars for the support of surface plants, residences, main roads and railways. Under sea working (and we have at the present very much of it in the Island of Cape Breton) requires special attention and has to be dealt with according to the nature and thickness of the cover. The fact that much of the coal that is now being extracted is taken from collieries opened up fifty years ago, is proof that present methods are superior to those of our predecessors, and it is only natural that they should be."—NEIL A. NICHOLSON, *Mining Society of Nova Scotia*, October 15, 1910, p. 627. (A. R.)

MODERN AMERICAN COLLIERY EQUIPMENT.—"A typical example of the most modern and efficient American colliery equipment is furnished by the Marianna mines, owned by the Pittsburg-Buffalo Company, Pennsylvania. They are situated within the boundaries of a coalfield, which it is estimated has an area of some 800 square miles, and which is expected to provide the future fuel supply of Pittsburg, the great iron and steel centre. For economical operation with a large area such as that available at Marianna, it was necessary to install the most modern machinery, as 1½d. per ton saved in operation more than covers the entire cost of this equipment. The installation of absolutely fireproof buildings for boiler, engine and power houses, and other buildings, prevents the possibility of delays by fire and is an insurance against stoppage of work from such cause. In equipping the plant at Marianna the above factors have been taken into consideration, and notwithstanding that the most modern machinery and fireproof buildings have been provided, the cost in proportion to the annual output has been less than the average mine in the district, or in the United States, being under 1½d. per ton of coal production. The force of the above statement will be better appreciated when it is considered that many mines having an annual capacity of only 50,000 tons to 100,000 tons per annum, and having a field of only 100 acres of coal, have equipment costing £20,000, or from 5d. to 7½d., per ton of coal produced. When it is realised that the production of the Marianna pits will be 1,500,000 tons per annum, or 5,000 tons per day, as against an average production of all mines in the United States of less than 300 tons per day of 280 working days per year, the magnitude of the operations will be apparent. Eighteen tons have been wound in one minute, or at the rate of 1,000 tons per hour, or more in one hour than the average American mine produces in three days.

Shafts, etc.—For the development of the seams two main rectangular winding shafts, each 22 ft. × 33 ft. outside the lagging and about 450 ft. deep, were sunk 4,800 ft. apart. These shafts are so situated that the coal from the entire field can be delivered to them with the gradients almost entirely in favour of the loads. The shafts have cageways 7 ft. × 20 ft., so that two trams can be raised on each cage in tandem; they are also provided with an airway 11 ft. × 20 ft., and with stairways from the surface to the bottom of the coal. A small shaft 12½ ft. × 24 ft. was sunk far the purpose of taking men in and out of the mine, as well as for handling slate and supplies; so that the larger shafts could be given over entirely to the winding of coal; while at the same time all supplies could he lowered in daylight. A circular brick-lined shaft, 16 ft. in diameter, ventilates the entire mine, avoiding all partitions and brattices in the shaft, the air leakage inevitable with such partitions, and at the same time making the air current independent of the motion of the cages. The tipple, washery and coke ovens have been located with regard to the duty of each in relation to the others.

The principal factors which determined the design of the surface plant were—that it should load a large tonnage of coal with minimum breakage in handling; that all impurities should be removed from the coal, and that there should be the least possible delay on account of breakdowns. The cages carry two loaded trams, each with a capacity of three tons of through-and-through coal, so that a large tonnage can be delivered with the least amount of handling. The coal is dumped over four cross-over dumps of heavy pattern, on to 1¼ in. bar screens. The lump coal, after going over the 1¼ in. screens is dumped on to the picking tables, where all impurities are removed and the coal delivered into railway wagons. The impurities which are taken out are thrown on to a lateral conveyor which delivers them to a crushing plant, where they are made serviceable for use in the boilers. The picking tables are 6 ft. in width, about 80 ft. long, and are run at a speed of 40 ft. per minute. The small coal falls through the 1¼ in. bar screen on to conveyors, whence it is elevated to revolving screens. These separate the nuts from the slack. From the revolving screens the nuts and slack are distributed by separate conveyors to the loading bins, the washer, or to the small bins over each picking table. The surface equipment includes

a Luhrig coal washer, with a capacity of 100 tons per hour. It has 16 primary jigs, and four secondary jigs which rewash the refuse from the primary jigs, thus recovering the "bone" coal.

Amusement Hall and Bath House, etc.—The houses in the town of Marianna are built almost entirely of brick and vary in size from four to fourteen rooms. Each house is provided with hot and cold water and bath. The houses are also equipped with electric light, and, if the tenants desire, with natural gas.

The amusement hall is a brick building of three storeys, 106 ft. 8 in. long by 62 ft. 8 in. wide. The basement contains eight bowling alleys, with lavatories, a barber's shop and restaurant. The floor above contains a billiard room and theatre, while the greater portion of the third floor is given over to a roller skating rink and dancing hall. On this floor there is also a reading room, and a lecture room in which it is proposed to give instruction in mining.

The bath house is modelled after the most advanced German shower or rain baths. It is a brick building 42 ft. × 125 ft. in size, and of four storeys. It is so arranged that men coming out of the mine are landed on a covered concrete bridge leading to the fourth floor of the building, which constitutes the assembly room. In the centre of the assembly room is the rack containing the safety lamps. The men going into the mine here receive lamps which are prepared for them by the attendants, and the men working coal-cutters receive their bits and oil for operating the machines. The men coming out of the mines deposit their lamps, empty oil cans and worn machine bits with the attendants, who clean and fill the lamps and exchange worn bits and empty oil cans for sharp bits and full cans of oil for the following day. The men coming out are checked by the time-keeper and pass down stairs to the bath rooms, where they bathe and leave the building with clean, dry clothing, leaving their soiled mine clothing suspended from the ceiling and drying out for use the next day. The building contains 99 shower baths for the men and two separate bath rooms for boys employed as slate pickers, etc., on the surface. There are also baths for the officials and visitors in separate rooms. An emergency hospital on the ground floor is a valuable adjunct.

The machine shop is really a collection of shops all under one roof, as the building contains a machine shop, carpenters' shop, blacksmiths' shop, tram repair shop, brass foundry and four large store rooms. The building is equipped with one 25-ton electric travelling crane, three small travelling cranes and a jib crane. The blacksmiths' shop and forge equipment is large enough to make all kinds of forgings for railway wagons, mine trams, and mining machinery.

Underground Workings.—The underground workings are on the same scale as the surface equipment. The main shaft bottom has a capacity of 250 loaded, and as many empty mine trams, and is arched for 1,100 ft. on the rise and 600 ft. on the dip side of the shaft. The gradient is ideal for handling coal, being 1·1 per cent. in favour of the loads for the entire length, so that no grading was required. The loaded trams are fed on to the cages by a chain haulage and the empties will be returned by a rope haulage which is now being installed. All the coal is gathered by means of 6-ton compound gathering locomotives. Steel track is laid into all workings, and there is not a mule in the mine. The main line hauling is done by compound compressed-air locomotives 15 tons and 19 tons in weight. These machines deliver the coal to the shaft bottom."—*Iron and Coal Trades Review*, Nov. 25, 1910, p. 860. (A. R.)

MISCELLANEOUS.

THE RUBY.—"Mineralogically the ruby belongs to the corundums, of which the sapphire, topaz, Oriental emerald, and Oriental amethyst are also members. Corundum is a sesquioxide of aluminium, and the various tints are due to the presence of different colouring matters. In the ruby oxide of chromium in greater or less degree is responsible for the ruddy colour so sought after.

A ruby of perfect colour is a very rare and beautiful gem, and its value is commensurate with its rarity. Indeed, a flawless 'pigeon's-blood' ruby weighing 5 carats would be worth ten times as much as a first-water diamond of the same weight, while perfect rubies of 6, 8, or more carats are of such unusual occurrence as to bring very high prices. By the term 'pigeon's blood' when applied to the colour of the ruby is meant that shade which is by international taste accorded the post of honour, though many of the gems have a darker tint which is quite as beautiful.

Burma, in India, is conceded to be the principal source of the true pigeon's-blood ruby. There the mining is in the hands of the Burma Ruby Mines, Ltd., an English corporation which secured the right of working the district soon after Burma became ceded to Great Britain in 1886. Previous to that date the location of the famous mines was unknown, and all attempts to learn about them resulted in failure, for the rulers exercised a most strict surveillance over foreigners, and saw to it that any such were kept in ignorance of the exact location, etc. It is known, however, that the methods used were very crude and tedious; the natives merely dug out the 'byon,' or ruby-bearing earth, hoisted it by crude derricks to the top of the pit (these 'pits' were from 50 ft. to 60 ft. deep), and there left it to dry in the sun before searching for gems. To-day essentially the same methods are used by the native miners, in whose hands such labour is still left, and who work under terms much more advantageous to themselves than were permitted under the old regime. In defence of the antiquated methods used it may be said that they bring results, and do away with the necessity of costly machinery.

Besides the true gems, large quantities of spinel and balas rubies are found. These spinels, though often (and quite excusably) mistaken for the genuine stone, are really minerals of very inferior qualities and entirely different composition, being in this latter respect aluminate of magnesium. The spinel is much less rare than the ruby and not so hard. Besides red, they exhibit a great variety of colours, some being bright cherry; a rare variety is a deep violet, others have a cinnamon shade; and a white spinel comes from Brazil mixed with diamonds. Many of the large historic rubies in royal regalias (including that in the Maltese cross of the English crown) have been pronounced spinels by modern mineralogists, and are consequently of little value. The balas ruby is merely a spinel of proper quality, having a rose or deep pink colour. The best balas are found principally in Ceylon, though both they and spinels are generally found wherever the ruby occurs.

From Ceylon also comes a true ruby of a rich rose colour, but not many of the desired pigeon's-blood shade are found, though in all other respects they are beautiful and attractive stones, suffering, of

course from the flaws which occur in all rubies. The Ceylon stones often belie the name, for in colour they run from very light red almost to a pinkish, being otherwise very brilliant.

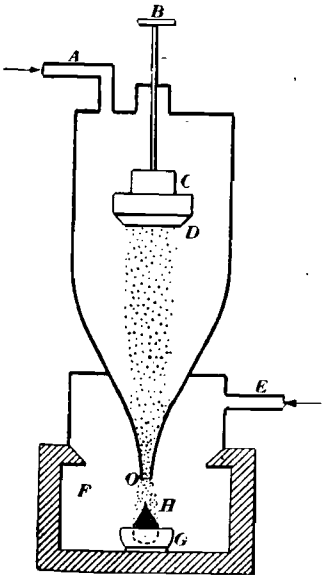
Similar to the Ceylon stone in many respects is the ruby of Siam. This country, though principally noted for its beautiful sapphires, often produces rubies that rival the best Burma stones, though their colour is more frequently a deep red (often called 'ox blood'). It is believed that any really good stones which may be discovered in Siam are sent overland to Burma and sold as Burma rubies.

The United States has brought forth rubies as well as sapphires. The principal districts are in North Carolina and Montana. In the former State gems of 3 and 4 carats were found, free from inclusions and of good colour; indeed, the quality of these stones often equals the pigeon's-blood rubies of Burma, though their occurrence is rare.

The detection of artificial rubies, as they are made to-day is an exceedingly difficult task, for the reason that the best artificial ruby is made of precisely the same materials of which the natural ruby is composed—pure alum and oxide of chromium or manganese, which latter form the colouring matter.

The furnace used in making these 'synthetic rubies' is illustrated in the accompanying diagram. It is a very ingenious application of the principles of the oxyhydrogen blow-pipe, and by its means a temperature as high as 2,000° C. (over 3,000° F.) may be obtained. The process of manufacture is as follows:—

A certain amount of powdered alum is placed within the box *C*, along with a proportionate measure of oxide of chromium to produce the red colour of the ruby. By means of a small hammer operated by an electromagnet, the tapper *B* is touched at various times. This causes the box to vibrate, and the powdered mixture is thus sifted through a fine gauze sieve *D*. Thence it passes down to the orifice *O* within the firebox *F*. At that



Furnace for making Rubies.

point the combining of the oxygen gas (admitted through the pipe *A*) with the hydrogen, which enters at *E*, on ignition causes a heat so intense that the powdered alum and oxide of chromium are instantly melted and fused into crystal globules, which fall into the platinum crucible *G*. In this manner a pear-shaped gem *H* (called a 'brut') is built up, the size depending upon the amount of material used. Some of these synthetic rubies are 80 carats in weight

After this 'brut' has been allowed to cool (if touched when warm it would fly to pieces owing to the great strain of its molecules), it is sent to the jeweller's, by whom it is cut into one or more gems which in colour, refraction, hardness, durability, and general beauty are identical with the natural ruby. Indeed, so absolute is this identity that the pawnbrokers of Paris and other great cities refuse to take rubies on pawn, for they cannot distinguish the artificial from the genuine.

Rubies can be made in this way at an expense of about 40 cents a carat. In the little laboratory of M. Pasquier in Paris (who introduced this method) 100 carats a day may be produced, and 8 to 10 carats an hour is a fair average in all such factories.

Such a discovery as this will have a serious and lasting effect upon the legitimate mining of stone. According to Mr. R. K. Duncan, in his 'Chemistry of Commerce': 'The ruby mines on their present basis of profitable working are absolutely doomed.' We can only hope that those who control this process will exercise good judgment in regulating the manufacture of their beautiful product, so that the market may not become unstable or overstocked.

Already the use of these synthetic gems is very widespread, and the natural ruby bids fair to become the exception rather than the rule. This fact, however, is not to be greatly deplored, for the artificial product is not an imitation; it is an equal of the genuine, even excelling it at times in beauty of colour and crystallization, and in freedom from flaws. If this were not so, the synthetic ruby could never have become the success which it is to-day."—MORRIS R. WARD.—*Mines and Minerals*, Dec. 1910, p. 319. (A. R.)

POTASSIUM CYANIDE FROM BEETS.—"The only factories where potassium cyanide is made from a by-product of sugar beets are two in Germany and one at Kolin, 40 miles east of Prague, Bohemia, according to a report made by Consul Joseph I. Brittain of Prague. The molasses by-product is sold to factories manufacturing alcohol, which by fermentation and distillation produce a first quality 97·7% pure alcohol for medicinal and chemical purposes, and a second quality denatured for fuel and light. The refined wholesales at $12.27 and the denatured at $0.13 to $9.54 for 26·41 gallons. The former pays an internal revenue tax of $18.27 per 26.41 gallons. After making the alcohol there remains a still thicker and darker coloured molasses, resembling pine tar in appearance. This residue is sent from the various alcohol factories in tank cars and barrels to Kolin where, after subjected to steam heat, it flows into long metal troughs and to retorts where it is burned for several hours, until the gas escapes into pipes. The material then passes through an extended system of pipes, undergoing various processes until it reaches the place where it is mixed with lye, when the potassium cyanide is formed, after which the moisture is extracted by centrifugal force. The powder is then conveyed to another room, where it is hydraulically pressed into cakes and packed

into boxes for shipment. The output of the Kolto factory is about 240,000 lb. a month, and is sold in gold and silver-mining companies in South Africa." —*Mining Science*, Sept. 25, 1910. (K. L. G.)

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