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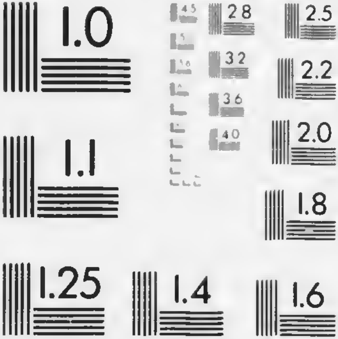

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RECENT CYANIDE PRACTICE

EDITED

BY

T. A. RICKARD

FIRST EDITION

1907

PUBLISHED BY THE

MINING AND SCIENTIFIC PRESS

SAN FRANCISCO

PREFACE.

This book gives, in convenient form, a compilation of the series of articles on cyanidation as they have appeared in the pages of the MINING AND SCIENTIFIC PRESS between January, 1906, and October, 1907. The contributors are the leaders in this branch of metallurgy and they describe the practice in mills all over the world, from Nevada to New Zealand, and from the Transvaal to Korea.

T. A. RICKARD,
Editor.

SAN FRANCISCO, October 12, 1907.

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CYANIDE AS A FACTOR IN GOLD PRODUCTION

(January 6, 1906)

In his presidential address before the chemical section of the British Association, G. T. Beilby said :

From the early beginnings of civilization gold has been highly prized and eagerly sought after. Human life has been freely sacrificed in its acquirement from natural sources, as well as in its forcible seizure from those who already possessed it. The 'Age of Gold' was not necessarily the 'Golden Age,' for the noble metal, in its unique and barbaric splendor, has symbolized much that has been unworthy in national and individual aims and ideals. With the advent of the industrial age gold was destined to take a new place in the world's history as the great medium of exchange, the great promotor of industry and commerce. While individual gain still remained the propelling power toward its discovery and acquisition, every fresh discovery led directly or indirectly to the freer interchange of the products of industry, and thus reacted favorably on the industrial and social conditions of the time. So long as the chief supplies of gold were obtained from alluvial deposits by the simple process of washing, the winning of gold almost necessarily continued to be pursued by individuals, or by small groups of workers, who were mainly attracted by the highly speculative nature of the occupation. These workers endured the greatest hardships and ran the most serious personal risks, drawn on from day to day by the hope that some special stroke of good fortune would be theirs. This condition prevailed also in fields in which the reef gold occurred near the surface, where it was easily accessible without costly mining appliances, and where the precious metal was loosely associated with a weathered matrix. These free-milling ores could be readily handled by crushing and amalgamation with mercury, so that here also no elaborate organization and no great expenditure of capital were necessary. A third stage was reached when the more easily-worked deposits above the water-line had been worked out. Not only were more costly appliances and more elaborately organized efforts required to bring the ore to the surface, but the ore, when obtained, contained less of its gold in the easily recovered, and more in the refractory or combined form. The problem of recovery had now to be attacked by improved mechanical and chemical methods. The sulphides or tellurides with which the gold was associated or combined had to be reduced to a state of minute subdivision by more perfect stamping or grinding, and elaborate precautions were necessary to insure metallic contact between the particles of gold and the solvent mercury. In many cases the amalgamation process failed to extract more than a very moderate proportion of the gold, and the quartz sand or 'tailings', which still contained the remainder found its way into creeks and rivers, or remained in heaps on the ground around the batteries. In neighborhoods where fuel was available a preliminary roasting of the ore was resorted to, to oxidize or volatilize the baser metals and set free the gold; or the sulphides, tellurides, etc., were concentrated by washing, and the concentrates were taken to smelting or chlorinating works in some favorable situation where the most elaborate metallurgical methods could be economically applied. Many efforts were also made to apply the solvent action of chlorine directly to the unconcentrated, unroasted ores; but unfortunately, chlorine is an excellent solvent for other substances besides gold, and in practice it was found that its solvent energy was mainly exercised on the base metals and metalloids, and on the materials of which the apparatus itself was constructed. This practically was the state of matters in 1889, when the use of a dilute solution of cyanide of potassium was first seriously proposed for the extraction of gold from its ores, and the proposal was far from favorably regarded from a chemical point of view, owing to the various difficulties which presented themselves. How each and all of these difficulties have been swept aside, how within little more than a decade this method of gold extraction has spread over the gold-producing countries of the world, now absorbing and now replacing the older processes, but ever carrying all before it all this is already a twice-told tale which he only felt justified in alluding to as they were then meeting on the Rand, where the infant process made its *debut* nearly fourteen years ago. The Rand today is the richest of the world's goldfields, not only in its present capacity, but in its potentialities for the future, twenty years ago its wonderful possibilities were unsuspected, even by experts. In 1889 the world's consumption of cyanide of potassium did not exceed 50 tons per annum. At present the entire consumption of cyanide is not much short of 10,000 tons per year, of which the Transvaal goldfield consumes about one-third. Large cyanide works exist in Great Britain, Germany, France, and America, so that a sure and steady supply of the reagent has been amply provided. In 1894 the price of cyanide in the Transvaal was 2s. per lb.; today it is 8d. Chemistry has so often been called on to play the part of the humble and unrecognized handmaiden to the industrial arts that he might perhaps be pardoned if in this case he called public attention to Cinderella as she shines in her rightful position as the genius of industrial initiation and direction.

THE CYANIDE PROCESS AT GUANAJUATO

By Francis J. Hobson

(January 6, 1906)

In 1899 a number of cyanide tests were made on Cubo ore by chemists in the employ of the Mexican Gold & Silver Recovery Co., the owners of the MacArthur Forrest patents in the Republic of Mexico. The results of these tests were far from satisfactory. The consumption of cyanide was low but the extraction also was low, averaging from 40 to 85% of the silver and gold in the ore. Most of the tests were made on samples weighing 100 grams, crushed to pass screens from 70 to 80-mesh. In addition to solutions of potassium cyanide there were added to the charges under treatment, different percentages of potassium ferri-cyanide, permanganate, and even ferro-cyanide, and to some of them were added various amounts of sodium dioxide. The time of treatment, in nearly every case, was 16 hours. In consequence of these experiments, the process was condemned at Guanajuato. Shortly afterward, on the recommendation of E. A. Wiltsee of the Venture Corporation, the Holmes brothers experimented for several months on Sirena and Cubo ores; the results of these further experiments led them to conclude that the ores of this district could be successfully cyanided by sliming everything. One point in their report was especially prominent, that even with 40 days' leaching the sand resulting from crushing to 40-mesh gave no extraction by treating with cyanide solution. This was on Sirena ore. About three and a half years ago the Butters company took up the question of cyaniding the Sirena ore, and E. M. Hamilton, of that company, ran a series of tests on a large scale at the *hacienda* Pastita, treating sand and slime after ore concentration on Wilfley tables; leaching the sand and agitating the slime. The ore was crushed in a stamp-battery to pass a 30-mesh screen. Fifteen days' leaching of the sand with 0.3% cyanide solution, and slime agitation with 0.05% solution gave excellent extraction. The metals in both cases were electrically precipitated on sheet-lead cathodes. The total extractions recovered by Mr. Hamilton from two series of tests showed 92.5 and 94.5% of the total value of the ore, the yield being in concentrate and bullion. On the basis of Mr. Hamilton's experiments the Guanajuato Consolidated Mining Co. installed the present excellent plant, with a capacity of approximately 200 tons per day.

Two years ago I was engaged by the American Finance & Security Co. to make a metallurgical examination of the Valenciana ores. The material treated first came from the Guadalupe dump, a dump which contains about 1,000,000 tons of ore. The tests were made on part of the samples taken from shafts which were sunk through the dump in order to ascertain its value. The tests were made with the 5-stamp mill and cyanide plant at the *hacienda* Central. First we milled several lots of ore with battery screens ranging from 20 to 30-mesh, concentrating on a Wilfley table, separating sand and slime with spitzkasten and spitzlutten. The spitzlutten were placed immediately under the spitzkasten. An excellent separation was obtained, making approximately 70% of sand and 30% of slime. About 500 lb. of sand and 150 lb. of the slime were shipped to Mexico City and there subjected to experiment in the laboratory of the Mexican Gold & Silver Recovery Co. Lots (each of 5 kilo.) of ore were leached continually, with solutions varying from 0.1 to 0.7% KCy for 15 days. In each case the silver and gold were precipitated on zinc shaving. A tailing sample was taken daily for 15 days, there being 21 sand-charges under treatment. At the end of 12 days the extraction ceased. Cyanide consumption ranged from 0.75 to 1.5 lb. per ton of ore treated. With 0.3 and 0.4% solutions the extraction was equal to that secured with stronger solutions, and the consumption of cyanide was only one pound per ton of ore treated. The experiments on slime were run on a very small scale at first, agitating in bottles with various strengths of solution; a chemical extraction of about 88% of the silver was obtained with 0.05% solution, with a chemical consumption of about one pound of potassium cyanide per ton treated. Then tests were made on a large scale, using compressed air for agitation and a 0.05% solution. Both decantation and the Moore filter-press methods were tried for separating solution from residue. Both were efficient but no apparent advantage could be gained by use of the Moore filter, the only saving being in cyanide, and this was so small that there was no profit in using the method, leaving out the question of royalties. The metals were precipitated on zinc shaving. Exhaustive tests with zinc-dust precipitation were not at all satisfactory. To my personal knowledge this was the second case where zinc-dust precipitation would not work. With these results for a guide, I milled 70 metric tons from the Guadalupe dump at the hacienda Central, concentrating, separating sand from slime, and cyaniding them separately. I used a 0.4% solution for leaching sand and 0.05% for agitating slime. Approximately 35% of the ore value was separated in the form of concentrate. We obtained an extraction on sand of 86% of the silver and 97% of the gold; from slime we extracted 90% of the silver and 97% of the gold. The time of sand-leaching was 12 days, and the total time of slime-treatment was three days, one day of which was active agitation. On sand, the consumption of cyanide was 1.1 lb., and on slime, including mechanical loss, it was a trifle less than 2.5 lb. per ton of dry slime. The metals were precipitated on zinc shaving. A clean-up was made, the coarse zinc was reduced with sulphuric acid, and the product melted into bullion. The two bars of bullion thus obtained contained in gold 2.4c. per ton of ore more than the assays indicated. The silver, on the other hand, was 8 gm., per ton of ore, short. Taking into consideration the fact that the precipitation was in a new wood zinc-box and that, after the clean-up was made, it was shipped to the State of Michoacan to reduce coarse zinc with acid, and then melted four times in order to get bullion 900 fine, it could not be considered otherwise than an absolute check on the work. A lot weighing 100 kg. of concentrate was shipped to Mexico City, one-half being leached without regrinding; the other half was reground to pass 100-mesh, and was then treated with cyanide by the agitation and decantation method. In both cases the metals were precipitated on zinc shaving. The time of treatment in the first case was 90 days with 0.3% solution; in the second eight days, with the same solution. Total extraction was more than 90%. There was no trouble in precipitation. In both cases the clean-up checked the assays of heading and tailing.

In consequence of these experiments, the 80-stamp mill at the Bustos and the cyanide plant at the *hacienda* Flores are being installed. I entertain no doubt of obtaining in practice results equal to those achieved in the laboratory in Mexico City and in the experimental mill at the *hacienda* Central. The laboratory results and the extractions obtained in actual milling varied less than one per cent. I believe Mr. Hamilton's tests are being confirmed in practice at Pastita.

Two years ago the owners of the Cubo mine installed a cyanide plant at their *hacienda.* I am not familiar with the results obtained, but they appear to be good from the fact that the plant is still running. About one year ago the Peregrina Mining & Milling Co. employed me to make a metallurgical examination. Results were highly satisfactory, 94% of the precious metals being extracted; and the company is operating a 20-stamp mill and cyanide plant; they are now erecting an additional 100 stamps with cyanide annex. It is stated that the Peregrina mill has obtained as much as 96% extraction of silver. George W. Bryant has a cyanide plant in operation at the *hacienda* Central, and another nearly ready at Nayal; the extraction is satisfactory and this plant also is being enlarged. Frank G. Peek, of the Portland Mining Co. at Cripple Creek, has purchased, and will soon have erected, a 40-stamp mill and cyanide annex at the *hacienda* San Matias. Mr. Ibarguengoitia has changed his *patio* process to cyanidation, and is treating daily about 30 tons of custom ore. The Adams company at La Luz expects to erect 100 stamps with cyanide plant. The Guanajuato district is now cyaniding 350 tons of ore daily; with the plants now purchased and under erection, this amount will be increased shortly to 1,100 tons per day.

FINE GRINDING

(January 27, 1906)

The Editor:

Sir—I have read with interest the communication on 'Fine Grinding in Metallurgy,' by Mr. E. E. Wann, dated November 29, in your issue of December 16

Mr. Wann says, in part: "Fine grinding having proven a pre-requisite for the highest extraction by amalgamation and cyanide leaching of gold ores." By fine grinding, I take it that Mr. Wann means crushing the material to such an extent that the largest particle will be fine enough to pass 150-mesh screen. My experience, gained in this country, the Australian colonies, and Mexico, leads me to believe that in the great majority of cases such fine grinding is by no means essential to securing the best commercial results. Of the number of varying classes of ore I have had occasion to examine during my six or seven years' residence in the West, I have met but two or three instances where fine grinding gave an increased commercial extraction of the gold and silver content. In most cases, where the larger portions of the crushed ore were fine enough to pass a 50-mesh screen, or, in other words, were approximately one one-hundredth inch diameter, a greater profit was obtained than when the material was pulverized to slime.

It may be generally stated that, wherever careful classification is carried out, and absolutely free sharp sand is dealt with by percolation, results equally as commercially good will be obtained from the sand as from the slime, though, of course, the treatment of the sand will require probably four to ten times as long as the slime.

In the somewhat rare cases in America in which fine grinding will show greater profit than average crushing, sand and slime treatment, I am strongly in favor of the grinding pan as a comminutor. The pans in pairs are about equal to one 14-ft. tube-mill in capacity, are slightly cheaper in operating and maintenance expense, and are about one-half the cost erected. If used in two series—stage grinding, with intermediate classification—it is probable that their capacity would be still further increased.

For slime treatment, either for true slime or finely pulverized ore, agitation in large, deep, flat-bottomed vats, fitted with revolving arms, to keep the solids from settling out of the thin pulp generally necessary, and with large centrifugal pumps, drawing from the bottom and throwing to the top for agitation and aeration proper, is universally the most economical system.

Removal of the dissolved value can be most economically and perfectly performed by the use of vacuum filters of the Moore-Butters-Cassel type. These filters are rapidly coming into use. They are about one-half the cost, by tonnage treated, of the standard-type filter-press, and their operation is attended with about one-third the cost. In but very few cases can an adequate extraction be obtained from the slime by conducting the treatment in filter-presses; it is almost universally imperative to dissolve the metals by agitation prior to the separation of the gold-bearing liquors from the solids.

In the treatment of gold and silver ore, considerable time is saved and a slightly increased extraction obtained if crushing in the solvent be resorted to. For the treatment of clean sand by percolation, more particularly silver-bearing material, two, and even in some cases five, treatments have been found profitable. The handling of this material can be economically performed by the use of the Blaisdell excavating, distributing, and conveying apparatus. In a skillfully designed plant, sand may be moved from one vat to another, using the above machinery, for approximately one cent per ton, and the extraction and profits enormously increased.

As an instance of the fallacy of attempting to fit the methods of one country to the ores of another, may be mentioned the case of Westren Australian ores and those of Cripple Creek, Colorado. In the former case, fine grinding is distinctly necessary to fit the raw ore for the highest commercial extraction, while at Cripple Creek an almost equally good extraction may be obtained at 30-mesh as at 200-mesh, either by bromo-cyanide treatment of raw ore, or by simple cyanidation of the roasted ore. As a matter of fact, a considerably higher extraction may be obtained on the roasted ore at 30-mesh than upon the raw ore, using bromo-cyanide at 200-mesh, and finer. The ore in both the above-mentioned districts has points in common, that is, that much of the gold in the material from the lower levels of the mines is combined with tellurium.

GODFREY DOVETON.

Denver, Colo., January 8, 1906.

FILTER PRESS PRACTICE IN WESTERN AUSTRALIA

By A. B. Wallace

February 3, 1906

When the mine operators in Western Australia turned their attention to the treatment of the gold-bearing slime, they had a choice of one of two methods, namely, decantation and filter-pressing. The serious difficulty was, and is, the scarcity and cost of water. At most mines at that time the water used for milling was the salt water which was pumped out of the mines or from wells. For boiler purposes, this water had to be distilled and cost from \$1.25 to \$2.50 per hundred gallons. At the present time, however, Kalgoorlie is amply supplied with fresh water by the 'Coolgardie Water Scheme,' as it is called, which supplies about 5,000,000 gal. per day. This water was pumped through a 30-in. main from the Swan river, a distance of 300 miles, and is sold for about 50c. per thousand gallons, meter measure. But in other districts beyond Kalgoorlie, as, for instance, at Menzies, many mines have to pay as high as \$5 per thousand gallons of salt water. For this reason, decantation methods, which involved the loss of, on an average, from 300 to 500 gal., or one or two tons of water per ton of dry slime treated, proved too costly, and, in those localities, filter-pressing was almost universally adopted as the only possible method of slime treatment.

The Dehne press or one of similar type is almost universally used. The presses are supplied with feed and wash channels through lugs attached to, but outside of, the frame. At first it was the practice to collect the water from the mill after separating the sand and allow the slime to settle, drawing off the clear water to be returned to the mill. The slimy pulp, in the proportion of about three of water to one of slime was then forced into the press by means of compressed air. As soon as the press was full the slime was treated for about four hours by pumping cyanide solution through it. In some cases compressed air also was used to dry the pulp; this method, however, took too much press capacity and was finally abandoned.

The method now used is to collect the slime from the mill and agitate it in a vat with mechanical stirrers, with the requisite cyanide solution, in the proportion of two solution to one slime, until all the gold which can be got into solution is dissolved and then filter-pressed, using the filter-press as a separating machine only.

The presses usually have had a capacity of from three to five tons of dry slime per charge; the thickness of the cakes varies, according to the material, from one inch to as much as three inches. The thicker the cake, of course, the more economical the treatment. The size of the cake is usually 40 in. square, there being from 30 to 50 cakes in each press. The filter-cloth is material specially woven for that purpose and costs about 50c. per yard at wholesale, so that each cloth costs about $1.25, and it has a life of from 30 to 90 days. If the material pressed contains much sand, the cloth is liable to get cut much quicker. One of the advantages of filter-pressing is the large proportion of dissolved gold and silver which can be recovered. When the pulp is pumped or forced into the press, practically 70% of the solution is recovered at once, then a weak wash or water is forced through the cakes. The washing is very thorough, as the water passes horizontally through the slime from one filter-plate to the other, only about two tons of water being necessary to wash one charge of three and one-half tons of slime. Compressed air is then introduced under about 90-lb. pressure, which forces out more of the remaining water so that the slime, when the treatment is completed, carries only a comparatively small proportion of moisture. That percentage of moisture varies. At the Lady Stenton mine, at Menzies, we often had a residue containing as little as from 11 to 15% moisture, but I consider that the average all over the country would be about 21 per cent.

Regarding the cost of treatment, this varies so largely in different districts that it is practically impossible to give an accurate average. In Western Australia it may be said to vary from $1.50 to nearly $4 per ton of slime, depending on the nature of the material on local conditions and on the time occupied by one complete cycle of operations. The last is an important factor, and it may be of interest to give a few figures. First, with reference to the labor necessary. It requires two men to discharge a press independent of the 'truckers,' who convey the residue to the dump. These two men can usually discharge a press, clean the frames, and close the press ready for another charge in from 30 to 45 minutes. As a rule two men are employed for each pair of presses, discharging and cleaning one while the other is being filled and dried. In some mills, where it takes longer to perform a cycle of operations, two men can handle three presses. On large plants, the shift-boss attends to the filling and washing, but on smaller ones this is also done by the pressmen, who receive about $2.80 per shift of eight hours. The time occupied in filling a press depends on the material and on the pressure used. As a rule, the pulp is forced under a pressure of from 60 to 100 lb. per sq. in. and the press is filled in from 20 to 30 minutes. Wash water is then used for from 15 to 25 minutes, and generally compressed air to dry the slime is admitted for about 10 minutes; the full cycle of operations, including the discharging and cleaning, being completed in about two hours. Thus, at Menzies, two men, using two presses, handle seven charges in an eight-hour shift, or about 27 short tons of dry slime.

The heavy labor cost is due chiefly to the difficulty of getting rid of the residues; these are taken away in cars and frequently have to be hauled some distance over the flat site on which the mill is necessarily erected in a flat desert country. Power is also an expensive item when fuel costs from $6 to $7.50 per cord, and water for boilers has to be distilled or bought. On the other hand, the method could be used economically in a district where there was a plentiful supply of water, as the residues could be sluiced away and water could be used to supply power both for agitation and for compressing air.

Of late it has been the custom to use pumps for filling the presses instead of compressed air, as being cheaper, but when this is done, a large air-chamber must be added to produce as steady a pressure as possible.

The Diehl process, which involves the sliming of all the ore, has been employed with great success at some of the large properties. This described briefly is as follows: The ore, after crushing and amalgamating, is sized, the fine slime going direct to the agitation vats, and the coarse sand going to flint or tube-mills for re-grinding. The product from these is again sized and the coarse material returned, the fine going, as before, to the agitation vats, which only receive the ore after it is ground to slime. It is then agitated with cyanide solution for a period varying from 16 to

24 hours, then a solution of bromo-cyanide is added and the agitation continued for a further period, after which the whole is filter-pressed. This method, on the whole, has been highly successful in Western Australia and owing to the peculiar local conditions, was the only one possible. In many cases, slime has been treated which had accumulated in the settling dams. This is taken in cars to a 'mixer,' in which the slime is thoroughly broken up in cyanide solution, after which the treatment, of course, is the same as previously described.

CYANIDATION OF CONCENTRATE

By Francis J. Hobson

(February 3, 1906)

It is generally held that gold as it exists in concentrates, is soluble in potassium cyanide, and the adaptability of the cyanide process depends entirely upon the physical characteristics of the concentrate to be treated, the relative association of the crystalline particles; provided, there are no base metals present to cause an excessive consumption of cyanide and consequent fouling of solutions.

In the case of silver, the question arises as to the solubility of the various natural occurrences of the metal as well as the physical and chemical characteristics of the concentrate.

The chief minerals of silver are: The native metal and the sulphide, argentite; four species among the sulph-arsenites and sulph-antimonites, namely, proustite, or ruby silver; pyrargyrite, dark red silver; freieslebenite, antimonial silver-lead sulphide; stephanite or brittle silver; the bromide and chlorobromide, bromyrite and embolite. Argentiferous tetrahedrite contains sometimes as high as 30%, silver as argentite; there is also argentiferous galenite and blende containing silver as sulphide.

The first mentioned, native silver, is so slowly soluble in cyanide that from it practically no extraction can be obtained. Argentite is readily soluble, likewise the chloride, bromide, and chlorobromide. Ruby silver, stephanite, and freieslebenite, are sparingly soluble in cyanide solutions but they are readily soluble in a solution of mercurous potassic cyanide. The extraction of silver by cyanide from argentiferous galenite, is a problem yet to be solved.

Taking these facts as a basis, in the absence of chemical interference, silver can be extracted from concentrates where it exists in soluble form, if the physical structure permits of solution-contact with the contained argentiferous compound. I have proved in the case of concentrates from Veta Madre ore, that the physical structure will permit contact sufficient for the extraction of both silver and gold without the fouling of solutions. I leached 100 lb. of concentrate obtained from ore crushed by stamps to pass a 30-mesh screen. This material was treated continuously for 90 days with 0.3% KCy solution. An extraction of 89% of the silver and 97% of the gold was obtained, an extraction of 91.5% of the total value. The metals were precipitated on zinc shaving in a porcelain zinc-box; the clean-up checked with the extractions called for by assay; all of the zinc was reduced with acid

One hundred pounds of concentrate, re-ground to pass a 100-mesh screen, were treated by agitation in a revolving barrel with a 0.5% cyanide solution, for eight days, decanting and renewing solution daily. The extraction obtained was 93.5% of the silver and 96 of the gold. I believe that 50 days' leaching of the re-ground concentrate would have given an equally good result. The cyanide consumption in both tests was seven kilograms per ton. The concentrate was low-grade, assaying about four kilograms silver and 30 grams gold per metric ton.

I am now running a test in my laboratory on a 100-kilo. charge of concentrate, assaying 26.5 kilo. silver and 130 grams gold. Silver exists in the concentrate chiefly as argentite. It is being leached with 0.7% solution. So far it does not show an excessive consumption of cyanide, and after seven days' leaching the extraction of silver was about 20%. After leaching for six weeks, the extraction is 53% of the silver and 80% of the gold. Leaching is still going on. Comparing this result with that obtained in the same time in the 90 days' leach, I expect a like extraction with three months' leaching. At first this may seem to be an exceedingly long time of treatment, but when you consider that from ores in this camp, less than one per cent of concentrate by weight of ore is obtained, a small addition to our cyanide mills will take care of our concentrate output. In the case of the Guanajuato Mines & Reduction Company's plant, we are erecting sand-vats each of 350-tons capacity. Two additional vats will handle concentrate, although it will doubtless be better to put in three vats one half the size of our sand-vats. This will allow for six weeks' collecting of concentrate and three months' leaching. The sand-mill solution can be used for this purpose without interference with sand treatment. The precipitation can take place in the sand-solution zinc-boxes.

I am of the opinion that concentrate from other ores of the Guanajuato district will be less amenable to cyanide treatment than those from the Veta Madre, as considerable value is carried in arsenopyrite whose molecular structure is more compact than that of the pyrite; consequently, if the extraction can be obtained, it will be by longer contact, as a more compact structure will necessarily cause slower diffusion of solution.

The best method of treating our concentrates is likely to be by leaching with cyanide solution for periods of from 30 to 180 days, followed by chloridizing roasts and further leaching with cyanide. The objection will be made that returns of bullion from the concentrate-leaching will be greatly delayed, but it will not be as long as the length of treatment suggests, because the larger part of the extraction will be obtained in the earlier period of leaching, and this bullion will be recovered as it is extracted, in the regular clean-up.

FINE GRINDING

(February 3 1906)

The Editor :

Sir—In the discussion on this subject in your issues of December 16 and January 27, several interesting questions arise. Is fine grinding necessary to the highest extraction by cyanide? Does fine grinding improve amalgamation ? Are the best commercial results obtained by fine grinding ? Mr. Doveton's definition of the term will bear amplifying. In this country 'fine grinding' is generally understood to mean a secondary comminution of the mill-tailing through 150-mesh or finer, after the usual mill treatment by amalgamation and concentration, or both. All the parties to this discussion seem to be of one opinion as to the effectiveness of this secondary grinding. Even Mr. Doveton, who is apparently not a very warm advocate of the system, concedes, by implication, that it aids extraction, but is only in a few instances commercially profitable. It seems to be a settled fact then that cyanide solutions will yield the highest extraction from finely crushed material. The exceptions to this are so rare that the fact may be safely accepted as one of the axioms of cyaniding.

The effect of fine grinding on amalgamation is a subject which, curiously enough, has been thrust into prominence by recent developments in cyaniding. I do not know of any instance where amalgamation is used after the fine grinding or sliming operation; although it is said to have been tried with some success in South Africa and in Korea. I think it has been the experience of mill-men that crushing finer than the approved and traditional 30 or 40-mesh rather hinders than aids amalgamation. In the Goldfield district, Nevada, the practice is to re-crush the coarsest of the 16 or 20-mesh product from the stamps in Huntington and Bryan mills. This has the effect of brightening the characteristically reddish-colored gold of the district, as well as to release the mineral imbedded in the coarse particles of quartz, and the use of large amalgamating plates after the second crushing has been of distinct advantage. My belief is that amalgamation, in cases where the gold does not amalgamate readily and where re-crushing is necessary for the highest extraction by cyanide, may be best carried on in two stages, especially where—as in the case of the Combination mill at Goldfield—the second crushing does not reduce the ore to finer than 40-mesh. When Mr. Wann speaks of "fine grinding having proven a pre-requisite for the highest extraction by amalgamation and cyanide leaching of gold ores," he implies that it is now considered an essential to good amalgamation. It would be interesting to know of the special instances wherein this practice has improved amalgamation.

Mr. Doveton's views on fine grinding, discrediting this aid to high extraction as commercially unprofitable, will be received with some surprise by companies now using tube-mills in this country and in Mexico. Several such mills are in operation at El Oro. Mr. Butters is installing them on his Mexican properties, the Liberty Bell company at Telluride, Colorado, and the Standard company at Bodie, California, have tube-mills in operation, apparently at a cost per ton which justifies their use. If fine grinding is a commercial success in Australia and at the various properties above named, all working on dissimilar ores, it must have decided merits. Obviously, the profitableness of this additional grinding will depend in each case on the margin of saving. If sliming a $2 sand will raise the extraction by cyanide from 80 to 90%, and the operation costs 50c. per ton, manifestly it will not pay. But if such an increase of saving is made on $10 sand, it *will* pay. It seems impossible to generalize briefly in estimating the value of a process, the success of which must depend upon so many inherent conditions. My own experience in fine grinding has been somewhat happier than Mr. Doveton's; though I must admit that, while an advocate of this method, I have not had occasion until recently to install a tube-mill or grinding-pan. What knowledge I have of the subject has been gathered from experimental work and from available data of plants in operation. I assume that Mr. Doveton has drawn his conclusions from reliable results, that is, from fair tests on a working scale over a considerable period. Obviously a mere experimental test to determine cost of treatment would not be conclusive. Those interested in this important subject would be glad to know specifically from Mr. Doveton the cases in which he met with failure in plants resorting to fine grinding on a scale large enough to afford reliable data. Such information would be of special value at the present time.

In the treatment of silver ores by cyanide I am disposed to take a middle ground between the extreme views held by Mr. Wann and Mr. Doveton. Agitation of finely crushed silver ores for the long period usually required for the best extraction is in most cases neither practicable nor economical. In the treatment of such ores I have obtained the best commercial results by regrinding to a fine sand (60 or 80-mesh) and leaching this product from 15 to 20 days, after a careful separation of the slime. The pure slime, by reason of the fine state of division of the mineral, does not require more than three or four days' agitation for the best results.

Mr. Doveton calls timely attention to the Blaisdell apparatus for emptying and filling tanks, and for carrying sand from one tank to another. The merits of this system from all points of view were tried and proved at the Butters plant at Virginia City. I am informed that two large cyanide plants now in course of erection in Mexico are to be equipped with this apparatus, as well as the new 500-ton cyanide plant of the Tonopah Mining Co., at Tonopah, Nevada. In districts where the price of labor is high, or where cheap labor is uncertain and unreliable, as in parts of Mexico, the value of mechanism of this sort is incalculable.

The evolution of slime filtering is one of the most interesting of recent developments. Filter-presses were never popular or much used in America. The Moore process, at first installed at the Golden Gate mill at Mercur, Utah, was soon discarded, but was later introduced at the Lundberg & Dorr mill at Terry, South Dakota, at the Liberty Bell mill in Colorado, and at the Standard Co.'s mill, at Bodie, California. At all of these places it is said to be operating successfully. Cassel introduced an important modification of the Moore filter by practically reversing the operation, that is, conveying the pulp and the various solutions to a box carrying stationary filters, instead of shifting the filters, as in the Moore scheme. The merits of these two types of filters were thoroughly investigated at Virginia City, where, after several months of quiet and earnest work, the Butters-Cassel filter, so called, was finally evolved. This is now in use at the Butters plant, where 150 tons of slime per day are being filtered at a cost of 11¼c. per ton. The system is very simple and works with a convincing smoothness and precision which appeal at once to the practical eye. At the mil of the Combination Mines Co. at Goldfield, this process is being installed to treat 40 tons of high-grade slime per day. The filter-press which has been in service for several months will be discarded as too expensive to operate.

FRANCIS L. BOSQUI.

San Francisco, January 27, 1906.

FINE GRINDING

(Editorial, February 10, 1906)

The contributions of Messrs. Bosqui and Doveton on this interesting subject ought to elicit further useful discussion. A few definitions, or at least, an agreement as to the precise meaning of the terms employed, may prevent confusion. Mr. Henry Louis, professor of mining at the Durham College of Science, has suggested that we use specific terms to describe the various stages of grinding: 'breaking' to be confined to coarse breaking of crude ore; 'crushing' for smaller division by pressure or impact; 'grinding' for comminution by attrition. The idea of technical terms is not to invent new or high-sounding words, but to secure precision by giving individual words a particular duty. 'Re-grinding' and 'sliming' are often used interchangeably, but they do not refer to an identical process; re-grinding is an intermediate operation which often precedes the final reduction to slime. Two separate products are made sometimes, as in the cyanidation of silver ores, in the two operations referred to; in other cases only one product is obtained by a two-stage process. As to 'sand' and 'slime', we have had a good deal of clever dissection of these terms; the last word of precision was said when slime was defined, by Mr. A. W. Warwick, as a colloid hydrate; but this is the chemist's not the millman's point of view. To the millman, 'sand' and slime' express a condition and not a theory. Formerly a product of extreme pulverization, so finely comminuted as to be unleachable, was called 'slime'. But we have changed all that; filtering has been so developed that solutions can be forced through materi l once regarded as impermeable, and slime is willfully made because the extreme subdivision of particles which it re, 'esents is favorable to quick solution in the presence of certain chemicals, such as potassium cyanide.

Slime is no longer an unleachable product. We have returned to the old definition which depended on sizing, and for the moment the arbitrary limit is 150-mesh, simply because that degree of comminution is found by practice to give the best results in the presence of two economic factors, the cost of reducing the size of the particles and the time required for solution of the precious metals. Unless made sufficiently small, the cyanide cannot get at the gold encased in the quartz. Meanwhile 'sand' continues to be the granular product, with sharp edges, readily percolable by solutions. The tendency is to decrease the proportion of this product by re-grinding it to the consistence of slime. Such 'sand' when re-ground to 150-mesh is not 'slime' in the eyes of a microscopist, because the particles are sharp, they are not colloid; but the cyanide expert who knows all these things, will call this product 'slime' just as much as the clay or mud produced in the first instance from the soft aluminous or feldspathic constituents of the ore. It is 'slime' to him because he gives it the treatment alloted to slime. In other words, 'sand and 'slime" are two mill products; they are not based on nice chemical or physical distinctions and they serve their present purpose. Anything finer than 150-mesh is slime, anything coarser than 100-mesh is sand, anything between is an accidental by-product. Being based on no true scientific distinction, we expect to see these terms discarded as cyanide practice becomes further developed.

FINE GRINDING

February 17, 1906.

The Editor:

Sir—As one whose experience in ore-grinding began 36 years ago and who subsequently has been almost uninterruptedly associated with it, I am pleased that interest seems to be taken in the public discussion of the subject. It is be hoped that a generous spirit will prevail and no bitterness engendered.

With some ores, values are pretty well released without very fine comminution, while I believe that we of the United States have generally erred in not making our pulp fine enough. Our *patio* neighbors to the south of us have always in the past excelled our work in that respect. They have been content with their slow-going *arrastres* and unmodernized Chilean mills, in which the coarser particles settle to the effective-crushing zone while the finer particles float above.

We early evolved the pan, with a faster motion, to grind as well as to amalgamate, but in this there is no systematic selection of the particles most needed to pass the grinding surfaces. The pan served the purpose when we knew no better, and strange to say it still, by some, seems to be considered an economical grinder. More than 20 years ago I departed from this practice by introducing special grinding pans in which the pulp passed once under a muller having a solid ring disc and thence out of the pan to another of the same kind, if finer pulp was desired. These were decidedly better and more economical than the amalgamating pan, used as a grinder, yet they were not as efficient and desirable as a grinder adopted some 16 years ago which was of the old buhr-stone flour-mill type, made entirely of metal, and run at a high speed. Where the bottom disc revolves instead of the upper, the sand particles rotate better between the two discs at the period of crushing, and so cut the metal to a less extent, and also the crushed particles pass more quickly away. Undoubtedly no method of grinding between two parallel surfaces of metal can surpass this simple and almost primitive device, and the cost of it is probably less than that of any other machine of like capacity. Yet all grinding between parallel surfaces unavoidably involves considerable loss of shoe and die metal.

Power is consumed by that as well as by any other means of making a hard ore very fine. When one considers the square surface area of cleavage involved in breaking a ton of ore to pass a 200-mesh screen, the power involved, without allowing for loss in its mechanical attainment, appears formidable in ratio with the hardness and toughness of the ore in hand. If a solid cube of rock weighing one ton is cut into cubes measuring $^1/_{400}$ of one inch, it involves a cleavage of approximately 15,600 square feet; yet if a sample could be had solely of such small cubes, it would appear as fine sand and not as slime. It is quite probable that a pulp that has passed a 200-mesh screen would more nearly represent three or four times the aforementioned area of cleavage.

It is not possible, at least commercially, to obtain as fine a product from grinding between two parallel surfaces of metal as that produced by a tube-mill, still it is not yet safe to say that there are no situations wherein the grinder is more advantageous than the tube-mill, and this without reflection upon the success of the latter.

The commercial profit line is affected by characteristics of environment as well as characteristics of an ore, and in one locality it might be profitable to grind an ore finer than the same ore in another locality, so that every individual case must be determined apart. Data of costs and conditions bearing upon other ores when treated by different means, are an assistance in forming a decision as to what course to pursue in any given case; but decision must be made by individual judgment and not by fixed rule.

M. P. Boss.

San Francisco, February 12, 1906.

MILLING V. SMELTING IN THE TREATMENT OF TONOPAH-GOLDFIELD ORES

By Francis L. Bosqui

(March 31 1906)

The report that Mr. Charles Schwab has entered into a smelter contract for the treatment of his Nevada ore raises the question as to how far smelters can compete with local mills in the treatment of the silicious ores of the southern Nevada district. I have been for some time of the opinion that this interesting field does not offer an encouraging prospect to the smelter. Recent experiments have shown conclusively that the representative ore of the Tonopah-Goldfield district—not including, of course, the higher grade silver-lead ores of the outlying country—can be handled by modern mill-processes to far better advantage than by smelting, at the current freight and smelter rates. At Goldfield, at the present time, it is unlikely that the smelter, even with much reduced rates, can hope to compete with the phenomenally high extraction obtained from oxidized ore at the Combination Mines Co.'s mill, the only milling plant in the district in continuous operation. From other properties at Goldfield shipments are being made to the smelters, but only because of the lack of efficient reduction plants. A large and picturesque variety of small custom mills have been in desultory operation at Goldfield for a year or more; but these have been hampered by scarcity of water, in some instances by lack of capital for proper equipment, and possibly, by unskilled management. The lively competition between these mills has resulted in the adoption of ridiculously low rates for custom-milling, and preposterous guarantees of extraction, involving a hard and unprofitable struggle. This has had the logical effect of discrediting milling in general, and has reacted somewhat in favor of the smelters. And until other mines follow the lead of the Combination company and erect their own plants, this will probably continue to be the status of milling in the Goldfield district. It is true that the present scarcity of water in the immediate vicinity is a serious obstacle to milling on a large scale; but this is not by any means insurmountable if mine owners will only be convinced of the advantage of owning and operating their own plants.

At the Combination mill the expediency of treating ore under $50 per ton, where an extraction of 90%, and better is obtained, is so obvious that no attempt has been made to ship material of this grade. In fact it has not been customary to send anything to the smelter under 20 oz. gold per ton.

The following interesting calculation of comparative net profits from shipping and milling high-grade ore from the Combination mine was made to determine to what point milling could be profitably carried. This table is based upon the Selby Smelting & Lead Co.'s schedule for September, 1905, as follows:

Value of ore per ton.	Smelter discount, freight and all expense of shipping	Net returns from smelter	Milling costs and losses in tailing, assuming $6 per ton for milling, and 90% extraction	Net returns from milling	Net profit in favor of.		
$200	$48 20	$151 80	$26	$174	Milling,	$22 20	per ton
300	50 30	249 70	36	264	"	14 30	"
400	57 .90	342 10	46	354	"	11 90	"
500	65 50	434 50	56	444	"	9 50	"
600	65 60	534 40	66	534	Shipping,	0 40	"
700	71 95	628 05	76	624	"	4 05	"
800	78 30	721 70	86	714	"	7 20	"

Freight rate from Goldfield to smelter, $22 per ton for ores assaying above 40 oz. gold per ton, with an increase of 3% for valuation above $300 per ton.

Smelter discount, $1 17 per oz. of gold for ore assaying under 15 oz., smelter discount, $0 92 per oz. of gold for ore assaying between 15 and 30 oz., smelter discount, $0 67 per oz. of gold for ore assaying 30 oz. and above. $1 per ton haulage from mine to railroad depot, $3 per ton cost of sacks and sacking.

It will be seen from the above that at the Combination mine it is more profitable to mill than to ship ore running even as high as $600 per ton, assuming an extraction of 90%. And, in practice, ore of this grade is being used to 'sweeten' the lower grade material to make a milling product of about 5 oz. gold

It should be explained in this connection, and with reference to the above table, that milling costs and losses would be still further increased by the amount of discount, freight, and treatment of concentrate, cyanide products, and bullion, reducing correspondingly the net returns and profit by milling. On the other hand, an increase of two or three per cent in mill-saving over 90% would very considerably raise the net returns from milling.

The following record of a working test made at the Combination mill, on comparatively high-grade ore, has a bearing on this point:

Tons treated, according to actual weight, 63.212; tons treated, by estimate of various products, 63.18, determined as follows: Slime, 14.4 tons; concentrate, 0.78 ton; sand, 48 tons.

Bullion recovery: 8.13.1 oz. amalgam, or 244.3 oz. bullion 0.912 fine, or 230 oz. fine gold.

Concentrate recovery: 0.779 ton, assaying 24.57 oz. gold, or 19.11 oz. fine gold.

Heading assay, sand, 2.4 oz. gold; tailing assay, 0.27 oz. gold.

Heading assay, slime, 2.065 oz. gold; tailing assay, 0.1425 oz. gold.

SUMMARY.

Recovered from bullion	230.0 oz.
Recovered from concentrate	19.1 "
Indicated recovery from sand	102.2 "
Indicated recovery from slime	27.7 "
Total gold	379.0 oz.
Loss in sand residue	13.0 oz.
Loss in slime residue	2.05"
Assay of ore, per ton	6.23"
Total gold in ore	394.05 oz.
Extraction	96.2%

In the above statement it should be noted that the recovery from sand and slime is indicated only on the basis of careful sampling and assaying, as it was not practicable to make a cyanide clean-up from so small a lot.

In reference to the oxidized Goldfield ores, it may be stated that they are peculiarly adapted to milling. The gold is free, but most of it is so extremely fine that it does not readily amalgamate, and a complete cyaniding equipment for handling both sand and slime is absolutely essential.

The sulphide ore at the Combination mine yields to a somewhat different treatment, in which elaborate preliminary concentration is the chief feature, followed by cyaniding. The small quantity of concentrate produced from the oxidized ore will be treated by cyanide. The sulphide concentrate, however, does not yield to cyanide, and will no doubt eventually be chlorinated.

The Tonopah ores, which carry much silver, as well as manganese, a small quantity of copper and other bases, were for a long time considered unsuitable for milling. This view is no longer held by the leading mine managers. Tests now being made on a representative Tonopah ore at an ore testing mill in San Francisco show that an extraction of 90% by a combination of concentration and cyaniding may reasonably be expected in a properly equipped plant. The ore in question carries about 25 oz. silver per ton. If we assume $15 to be the average value of the milling ore in the district, an extraction of 90% and a milling cost of $3 per ton on a 40-stamp basis, we have a total milling expense and loss of only $4.50. This would be slightly increased by the cost of shipping the concentrate; but even then the cost would be sufficiently low to discourage shipping and smelting. And if the high-grade ore of Goldfield can be more profitably milled than smelted, then the same rule should apply to the Tonopah ores. Indeed, every present indication points to the distinct advantage of milling over smelting as applied to the silicious gold and silver ores of this whole new district; and with the development of a better water-supply for milling purposes there is no reason why every property of any consequence in southern Nevada should not have its own independent reduction plant.

RE-GRINDING

(Editorial, April 7, 1906)

It is likely that the tube-mill and the Chilean mill will be close rivals as devices for re-grinding the product of the stamp-mill. At Johannesburg, for instance, among other installations of tube-mills, two 18-ft. tubes have just been erected at the Ferreira mill; one of these was started running on January 11. This mill is re-grinding the spitzlutten discharge from a 120-stamp mill that is crushing through a 22-mesh light screen and giving a duty of about six tons per stamp per 24 hours. About 250 tons of product per day is passing through the tube-mill. After leaving the tube-mill, the pulp passes over four shaking amalgamated plates, where about 60 ounces of amalgam per day are caught from the daily scrapes and from black sand caught on the plates. Before starting the tube-mill, the custom was to collect from the spitzkasten about 10 per cent of concentrate averaging 14 dwt. per ton. This was given a three weeks' treatment with cyanide, which brought the gold in the residue down to ab t 2.5 dwt. Now only two products are being made: Sand, representing 70 per cent of the mill-tonnage, and slime representing 30 per cent. The total residue from the company's works before starting tube-mills averaged 1.06 dwt., now they have come down to 0.6 dwt., and the manager expects to do better. The following figures in the use of shaking amalgamated plates in this mill during 1905 will be of interest: There are 24 of these plates running below the ordinary fixed plates at the batteries; during the year there was collected from these plates 16,028 ounces of amalgam; 78 per cent of this total was recovered from dressing the plates and 22 per cent from steaming them; the total fine gold obtained was 4,808 ounces, valued at $97,941, less cost of running and maintenance of plates, namely, $5,683; that is, $92,258 represents additional profit to the mill.

In Mexico, a Chilean mill fed with a product that had passed a 2½-inch grizzly, reduced 18 tons per day at one operation, so that 20 per cent was between 100 and 200-mesh fineness, while over 77 per cent passed 200-mesh. With a 10 to 1 dilution and a consumption of 12 horsepower, this mill ran for 18 months without stopping for repairs. It cost $4,000 at Pachuca and did the work described for 23 cents per ton. Chilean mills have long been in use at Pachuca, for they are legitimate descendants of the *arrastre;* nevertheless, even the Spanish engineers at that ancient mining centre are using Abbé tube-mills, so that it is evident that we should soon be in possession of valuable data, secured by making trustworthy tests between the various machines employed for re-grinding.

IRON V. WOOD FOR CYANIDE LEACHING VATS

By Francis L. Bosqui

April 14, 1906

The following facts in regard to iron and redwood leaching vats for the cyanide process will determine which type is best to use in a given locality.

In warm countries, where the winters are not severe enough to require housing the plant, the iron vat is to be preferred. Iron vats are more expensive in first cost and erection than redwood; they are also more expensive to maintain, owing to the necessity of frequent coating to protect both the iron and the cyanide solutions. They must be set up with special care and subjected to water-tight test, as leaks are difficult to stop, especially in the bottom, after the filter has once been laid. A serious objection, not generally recognized, is the lack of rigidity of iron vats when under pressure of a charge of ore, and the tendency of the periphery to shift from the line of the true circle. This shifting or bulging will take place during the filling or after solution is applied, and is due to the charge not being uniform in density and of varying pressure at different points. Such a variation, with the consequent yielding of the iron, will oftentimes produce cracks throughout the charge of sand. This prevents uniform percolation and is a serious matter. It can be prevented, however, in great measure by using sufficiently heavy material and reinforcing the vats with bands of angle iron.

The redwood vat is perfectly rigid, and is much cheaper in first cost and maintenance than the iron vat. It does not require as frequent coating with protective paint. There is a popular prejudice against wooden vats owing to their alleged greater liability to leakage, and their absorption of values. The leakage may be prevented by good workmanship in setting up the vat. The redwood leaching vats at the Smuggler-Union cyanide plant in Colorado are 40 feet in diameter, and are absolutely tight; and I am quite positive that the same may be said of the large 54-ft. vats at the Homestake. Rare cases have been reported of losses from absorption. I investigated this point several years ago. The results of the tests made on that occasion are quoted herewith from my article in *The Engineering and Mining Journal* of Febuary 26, 1898.

A piece of redwood 5 ft. 2 in. long and 2½ in. thick was submerged for three weeks in a gold solution at Bodie. Weighing before and after submersion showed an absorption of 2¾ lb. in 80 hr., equal to 22% of the original weight. Assuming that a vat weighing six tons would absorb as much as 50% of a $5 cyanide solution, the absorption would amount to three tons, or $15—an almost imperceptible loss in a plant treating several thousand tons per month.

Another way to account for loss of gold by absorption is to assume that the absorption is cumulative—that is, in a leaching vat submitted to intermittent contact with solution, the gold is deposited in the wood by a series of partial evaporations. To determine which was accompanied by the greater loss by absorption—continuous or intermittent contact—and whether either was responsible for any considerable loss, the following tests were made:

First series (continuous contact).

(a) A piece of dressed redwood, thoroughly seasoned, weighing 2 lb. 2 oz. and exposing 180 sq. in. of surface, was submerged for three weeks in a solution carrying an average of $5 per ton in gold. The wood was then taken out and dried, and reduced to charcoal in a clean solder stove and further reduced to ash in a new assay-muffle. The ash was carefully fluxed, cupeled, and the button weighed, with the following results, silver being valued at 50c. per ounce:

		Gold.	Silver.	Total
Actual absorption	mg.	1.65	2.75	4.40
Absorption per ton of wood	grams	1.552	2.588	4.14
Value absorbed per ton of wood		$1.03	$0.04	$1.07

(b) A piece of seasoned dressed pine weighing 3 lb. and exposing 180 sq. in. of surface was submerged at the same time as the redwood and subjected to the same conditions:

		Gold.	Silver.	Total.
Actual absorption	mg.	1.7	2.6	4.3
Absorption per ton of wood	grams	1.133	1.733	2.866
Value absorbed per ton of wood		$0.75	$0.02	$0.77

(c) A piece of rough pine weighing 2 lb. 7 oz. and exposing 144 sq. in. of surface was taken from the bottom of a discarded screen-frame, which had been constantly and directly exposed to a strong solution (averaging $5 per ton in gold) for a period of eight months, and again for a period of four months:

		Gold.	Silver.	Total.
Actual absorption.	mg.	9.5	10.8	20.3
Absorption per ton of wood	grams	7.791	8.86	16.65
Value absorbed per ton of wood		$5.18	$0.14	$5.32

Second series of tests (intermittent contact).

(a) A piece of dressed redwood weighing 2 lb. 7 oz. and exposing 180 sq. in. of surface was subjected to intermittent submersion in a strong gold solution for three weeks. The piece was left in the liquid for 16 hr. in 24, and then removed and dried in the sun for the other eight hours:

		Gold.	Silver.	Total.
Actual absorption.	mg.	4.8	2.1	6.9
Absorption per ton of wood	grams	3.657	1.6	5.257
Value absorbed per ton of wood		$2.43	$0.02	$2.45

(b) Two pieces of rough pine, weighing together 3 lb. 4 oz. and exposing each 160 sq. in. of surface, were taken from near the top of the disused screen-frame from the gold tank. This portion of the frame had been alternately exposed and dried for 12 months, and was probably about one half of the time under solution:

		Gold.	Silver.	Total.
Actual absorption.	mg.	6.7	13.7	20.4
Absorption per ton of wood	grams	4.122	8.429	12.551
Value absorbed per ton of wood		$2.74	$0.13	$2.87

It may be noted that the portion of the screen-frame subjected to long continuous submersion showed the highest absorption—$5.32 per ton of wood—and that the piece of redwood exposed intermittently to solution for three weeks indicated almost the same degree of absorption as the top portion of the screen-frame exposed for several months.

The indications are that in no case would the absorption be much of a factor in gold losses, even in a vat of unprotected surface.

THE ASSAY OF CYANIDE SOLUTIONS

BY WILLIAM MAGENAU

(April 14, 1906)

About a year ago I had occasion to search for an extra rapid method for determining gold in cyanide mill solutions, to be used in controlling the operations of an agitation plant which handled charges of 100 tons each. The process involved special chemical treatments, and washing by decantation; and a quick solution-assay was imperative right through. As a basis, I gathered from text-books and files of the technical press all suggestions along this line, and tried but a few before finding one which, with some modifications, answered well.

It may interest cyaniders to know this method, especially as they may have had the same discouraging experience as I had with the method as published. It was described in *The Engineering and Mining Journal* of March 28, 1903, by Alfred Chiddey; in substance it was as follows: Add to four assay-tons (or more) of the solution 10 c.c. of 10% solution of lead acetate; then four grams zinc shavings; boil a minute; add 20 c.c. HCl (strong). When action has ceased, boil again and decant from the ball of spongy lead which has been compacted a little with a glass rod, and wash same by decantation. Transfer to a piece of filter-paper, roll into a compact ball, and place (without drying) on a hot cupel in the muffle.

Mr. Chiddey states that a result can be obtained in 25 minutes, which I failed to confirm. It is certainly the experience of most of us that it requires several times that length of time to bring four assay-tons (120 c.c.) of solution twice to a boil, to *dissolve four grams Zn in HCl*, to conduct a cupellation, then part and weigh, to say nothing of manipulation. But far from making any speed records, I could not get the method to work at all, following the scheme to the letter. Instead of the lead coming down spongy, it was finely divided and showed no tendency to agglomerate, and had to be filtered out to collect it at all, which it was the very merit of the proposed method to avoid. But the precious metals were perfectly precipitated, and realizing the advantages of the scheme, I experimented with other proportions of reagents than those given and got good results from the following:

Take a large beaker (about 600 c.c. capacity) five or ten assay-tons of solution, and if it is very weak in KCy add enough of strong pure solution to bring it up to about 0.5% strength. Add 10 c.c. of a 10% solution of lead acetate, slightly acidified with acetic acid (12 c.c. if 10 assay-tons were taken); then add by rough measure on the point of a spatula between 0.3 and 0.4 gram (not more) of zinc dust, this being sufficient to precipitate all lead present and leave an excess of zinc to act on Au and Ag. Stir and bring to a boil; then add 15 c.c. strong HCl and leave on the hot plate until excess Zn is dissolved and lead has gathered into a sponge. This will take but a few minutes, and the assay should then be taken from the heat, else the lead will begin to dissolve. Now gather the lead a little with a glass rod whose end has been flattened while hot, or with the fingers encased in a rubber cot, decant rapidly, wash by decantation two or three times with large quantities of water, pick up the lead (which should be soft and strongly inclined to stick together when pressed) in the fingers and compress it into a ball which place upon a piece of lead foil 1¼ in. square. (Using filter-paper instead of lead foil, the spongy lead is apt to separate in the cupel.) Fold up in such a way as to leave an egress for steam, and drop into a hot cupel in the muffle, having first added a bit of silver, if the assay is for gold only.

The solution on which this method gave excellent results, and checked perfectly with the old evaporation methods in minimum time, carried about 0.075% KCy and from nothing to one ounce Au per ton with almost no Ag. It is possible that Mr. Chiddey's method, as published, works better on other solutions, though it is not clear why so much as four grams zinc need be used, taking a long time to dissolve when one-tenth as much, in the shape of 'dust,' is ample. Perhaps different proportions of reagents will be necessary to suit each case, but operators will find it worth a little experimenting to secure a method which involves no filtration, no fusion nor scorification, is accurate and rapid, easily performed and equally suited to large batches and single assays.

The method may be used without application of heat, as the precipitation of precious metals is perfect in the cold, but the lead does not agglomerate well and the excess zinc takes longer to dissolve. When done this way, the precipitated lead (plus Au and Ag) must be filtered out and the paper, unwashed and still wet, rolled up in a piece of lead foil about two inches square and dropped into a hot cupel.

It may be of further interest and a convenience to have the list of methods which I gathered, chiefly from the American technical journals of the past eight years; cyanide chemists may be encouraged thereby to give them a trial, and let the profession know the results. Gold determination has been the chief concern, but these methods, except where noted to the contrary, may be used equally well for silver.

I. METHODS INVOLVING CRUCIBLE FUSION OR SCORIFICATION BEFORE CUPELLATION.

1. *Evaporation* in a lead-foil tray, either direct, or after concentration to small bulk in a porcelain dish, the finish to be conducted at a very low heat. The tray is rolled up so as to enclose the residue, and is scorified with test lead before cupellation, parting, and weighing. With small charges (say one assay-ton) of solution carrying only small amounts of dissolved matter, the scorification can be omitted. A variation of the method which avoids the lead tray, is to sprinkle into the charge of solution (in a porcelain dish) 20 grams litharge, in such a way as to cover the bottom; then evaporate to dryness, and scrape the residue with a spatula into a crucible containing 10 grams sodium bicarbonate, five grams borax glass, and one gram flour or argol. The dish can be thoroughly cleaned by moistening with dilute nitric acid and mopping out with small pieces of filter-paper which are added to the crucible. Fusion results in a lead button, which is cupelled, etc. The evaporation method is the oldest in use, and when carefully done is accurate. It involves much time, but little and simple manipulation.

2. *Precipitation as Sulphide in Acidified Solution.*—This is given in *The Engineering and Mining Journal*, November, 1898, by Henry Watson, and is stated to be quick, accurate, and economical.

Acidify five or ten assay-tons solution with HCl and heat to boiling; while boiling, add a solution of two grams lead acetate and pass a current of sulphuretted hydrogen until all Pb is precipitated. Allow to cool a little while, still passing H_2S, then filter, precipitate, dry, and reduce to a lead button by scorification with lead test or fusion with fluxes in a crucible; cupel, etc.

3. *Precipitation by Cement Copper in Acid Solution.*—Albert Arents in *Trans.* A. I. M. E., Vol. XXXIV, p. 182: Add a few cubic centimetres H_2SO_4 to six or eight assay-tons of solution, stir well, and add one gram (not much more nor less) of commercial cement copper. Boil hard for 10 min. Filter through a large gray paper without washing. Place in an assay crucible one-third of a fluxing mixture consisting of 30 grams litharge with usual amounts of soda, borax, and reducer, and on the drained filter, another third. Then pick up the filter, tuck it into the crucible, and cover with the remaining third of flux. Now fuse, cupel, etc. Instead of cement copper, a solution of $CuSO_4$ may be added and then some aluminum foil. Boiling the assay for some time brings down the copper. The aluminum foil should be added to the fusion of filter.

4. *Precipitation by Silver Nitrate.*—Credited to A. Crosse on p. 161 of Gaze's book on the 'Cyanide Process.' Add an excess of silver nitrate solution; filter and wash; dry filter and either scorify it with test-lead after incineration, or fuse with fluxes in a crucible, cupelling the resultant lead button. This method, of course, can be used for Au only. Gaze also gives a modification of this which employs a bichromate of potash indicator to determine when there is an excess of $AgNO_3$, and directs that a few grams of zinc-dust be added besides. After stirring well, an excess of H_2SO_4 is added. Then filter and wash, dry, fuse, and cupel; or the incinerated filter may be cupelled direct when wrapped in lead-foil. Mercuric chloride may be used instead of silver nitrate.

5. *Precipitation by Copper Salt.*—Maurice Lindemann in *The Engineering and Mining Journal*, July 1, 1904. Heat 10 assay-tons until quite hot; add ammoniacal copper nitrate solution to a permanent blue; carefully acidify with H_2SO_4; stir and filter immediately. Dry and incinerate the filter, fuse or scorify, cupel, etc. Some chemists claim this method will not work on foul solutions.

Grant D. Miller in *The Engineering and Mining Journal*, June 23, 1904, treats 34.3 assay-tons (one litre) of solution in a two-litre flask with one to two grams copper sulphate. After shaking well, add 10 to 15 c.c. strong HCl and shake well again. Filter, dry, and incinerate, fuse, cupel, etc., as before.

6. *Precipitation by Zinc Shaving or Dust in Acid Solution.*—H. L. Durant in *Jour. Chem. & Met. Soc.* of S. A., December, 1902. Place a large measured bulk of solution in a flask, acidify strongly with H_2SO_4 and bring to a boil. Add 30 grams zinc shaving, about five grams at a time, waiting with each addition until all that previously added is dissolved. Keep solution strongly acid throughout. When all has been added and dissolved, put in one gram precipitated silicia to collect floating matter, filter through double papers, dry and char the papers on a scorifier, and then add test-lead, scorify and cupel. This method can be used for Au only, since Ag dissolves as sulphate.

A much more practical method is used in the Black Hills and elsewhere. To eight assay-tons of solution are added enough silver nitrate solution to furnish inquartation for gold; then 0.33 to 0.5 gram zinc-dust and sufficient H_2SO_4 to react completely, say 5 to 10 c.c. Stir a little and filter through an ordinary paper, dry, incinerate, and fuse in a crucible with litharge and reducer for a lead button, which is cupelled, etc.

7. *Precipitation by Emulsion of Zinc-Dust in Lead Acetate.*—This method was used in some large mills in Colorado a few years ago, but I am unable to give more than a bare outline, and do not know how it will work out. Make a thin paste of zinc-dust in a strong solution of lead acetate and add about 8 c.c. of this to 20 or 30 assay-tons of solution. Agitate and let stand for a time. Filter and add dilute H_2SO_4 to the filter and wash. Dry, filter, incinerate, fuse with litharge (or scorify with test-lead), and cupel, etc. This is virtually the same as suggested for cold treatment of the modified Chiddey scheme, already described, though differing in manipulation.

II. METHODS WHICH GATHER PRECIOUS METALS IN LEAD THAT CAN BE CUPELLED DIRECT.

1. *Evaporation* either conducted entirely in, or finished in, a lead-foil tray, when solution is comparatively small in amount (one to two assay tons) and contains only small amounts of dissolved salts [see I (1)].

2. *Chiddey Method*, or modification described. This seems to be really the only method properly and exclusively in this class.

III. COLORIMETRIC METHODS (for gold only).

1. *By Stannous Chloride Following Precipitation and Re-solution of Gold.*—Prister in *Jour. Chem. Met. & Min. Soc.* of S. A., December, 1903, gave a method which would seem too long for practical use. He adds to the acidified and boiled solution under assay a few cubic centimetres of a 10% solution of $CuSO_4$ containing 20% NaCl, and acidified with acetic acid after having been boiled 10 min. with copper shaving and cooled. The precipitate resulting from addition of this reagent is filtered out and re-dissolved in a KCy solution, and to this is added a half gram of zinc-dust. After agitation for a few minutes, this is filtered off and treated on the filter with dilute HCl. Treatment of the same filter with aqua regia (minimum quantity) dissolves the gold, and in the solution an excess of strong stannous chloride solution gives the 'purple of Cassius' test, which may be compared with standards.

Later Prof. Prister published a simplification of this scheme which reads better. The solution under test is brought up to one per cent KCy. To 234 c.c. (eight assay-tons) add one gram zinc-dust and heat to boiling. Filter; pour on filter 20 c.c. hot 10% H_2SO_4 repeating until all the zinc is dissolved. Treat residue on the filter with 10 c.c. hot aqua regia (6HCl, $2HNO_3$, 6 water), passing the 10 c.c. repeatedly through same filter at a boil. Collect in a test tube, cool and add a "few drops" stannous chloride as before. Prof. Prister does not say how he avoids disintergration of the filter by action of hot aqua regia. From work I have done with 'purple of Cassius' tests on aqua regia solutions of gold, good and constant colors cannot be obtained unless the acids are present in small amounts and a large excess of stannous chloride is used.

2. *With Stannous Chloride Without Previous Precipitation of Gold.*—Henry R. Cassel has given a promising quick test as follows: To 10 c.c. of solution in a test-tube is added 0.5 gram potassium bromate, then pure strong H_2SO_4 slowly until action starts. When action has ceased, add drop by drop a strong solution of stannous chloride, until the solution is just colorless. In about half a minute the purple color will form. Mr. Cassel says the test

can be performed in three minutes, and is delicate to a value of a few cents per ton of solution. By concentrating by evaporation very low-grade solutions or diluting those too strong in gold, this method may apply to a wide range.

He also used other agents than potassium bromate as a destroyer of the interference of cyanogen, namely, potassium chlorate and HCl, or H_2SO_4, with long boiling; potassium bromide followed by sodium peroxide and neutralization with H_2SO_4, after which HCl, and stannous chloride are added to give the color. This last acts well. Another scheme is to add to the solution one-third its bulk of strong ammonia and then neutralize with strong H_2SO_4. The solution will then react with stannous chloride. For quantitative work, standards must be prepared, and to make these durable is a great difficulty of the method. Considerable skill is also needful.

IV. ELECTROLYTIC METHODS.

1. *Deposition on Lead.*—In *Trans. Inst. of Min. & Met.*, May 16, 1905, F. B. Stevens gives a method which does not appear to possess any advantages over the chemical processes in points of time, convenience, or accuracy. Fourteen to twenty assays of 10 assay-tons each (292 c.c.) are electrolyzed by current from 110-volt lighting main for four hours, using a lead-foil cathode of cylindrical shape 2½ in. high and 1 in. diam., with three V notches around the bottom, and a large iron nail for anode. To each assay 12 to 15 c.c. of strong ammonia is added. The cathode should be covered with one-half inch of solution. Twenty assays give just right resistance, and with less than 14, electric lamps must be put in circuit. With a current varying from 0.06 to 1.2 ampere, according to foulness of the solution, the gold comes down bright. The cathode is washed, dried, rolled up, and scorified with test-lead before cupelling.

The only figures given for completeness of precipitation state that a solution carrying \$10 to \$15 in gold will be cleaned to about 10c. in four hours. If this is the best that can be done, the method is not applicable to low-grade solutions. It would seem that cupellation of the cathode might be made without preliminary scorification

It will be noticed that for most of the above methods charges of 10 assay-tons, or less, are recommended, which will perhaps seem inadequate to some, particularly South African operators. Most English authorities (as Julian and Smart) insist that only very large charges be used. The authors named say not less than 30 assay-tons and up to 200. In America balances can readily be purchased which are sensitive to 0.01 mg., or even one-fourth of this. Taking the former figure, 200 assay-tons would give gold results accurate so far as the weighing is concerned, to 0.001 per ton. What conceivable use such a degree of precision can be is not stated. Even 30 assay-tons would weigh down to 0.007, which is far finer than available methods of sampling and measuring ore and solution tonnages justify. Ten assay-tons give a bead weighable to 0.02, and nothing besides investigation work need be concerned with less than this.

THE ASSAY OF CYANIDE SOLUTIONS

(April 28, 1906)

The Editor:

Sir—Mr. William Magenau, in your issue of April 14, mentions some modifications in the method of Mr. Alfred Chiddey described in *The Engineering and Mining Journal*, of March 28, 1903. I have used the same method with somewhat different modifications, getting accurate results but consuming a much longer time. The method, as modified, is as follows:

Take 100 c.c. of the solution to be assayed, add 7 c.c. of a 10% lead acetate solution, then add one gram of zinc shavings and place on the hot plate. Heat, but not to boiling, until the lead has gathered around the pieces of zinc. This usually takes about 25 minutes. This precipitation being complete, 20 c.c. concentrated HCl is added and the heating continued until all effervescence has stopped. The lead is then in such a spongy condition, that by the aid of a flattened glass rod, it can be pressed into a cake and the clear solution poured off. Wash the lead twice and then with the fingers press it into a compact mass, drop in a lead-foil funnel, leaving a vent for escape of the steam, place in a hot cupel and proceed as in an ordinary cupellation. This latter operation consumes from 25 to 35 minutes; the lead button is ready for cupellation in about one hour.

I have used this on solutions varying in strength from 0.01% KCN to 0.4% KCN and in value from 0.04 to 3 oz. gold per ton. I have also used it for the assay of silver solutions, but when they are rich the results are lower than those obtained by the evaporation in lead boats.

The secret of keeping the lead from breaking up, is not to allow the solution to come to a boil at any state of the procedure.

NORMAN C. STINES.

Berkeley, Cal., April 23, 1906.

THE TREATMENT OF DESERT ORES

(May 26, 1906) *

The Editor.

Sir—One of the first requisites of good criticism is that the critic shall at least give his victim a fair hearing. I refer to certain strictures in your issue of April 28, on a brief article of mine touching upon ore treatment in the Tonopah-Goldfield district. If unwittingly I have trodden rough-shod on any of Mr. Bertram Hunt's metallurgical hobbies, I am sorry; but before proffering a formal apology I must ask him to do me the justice to re-read my article, which he so bluntly dismisses and so obviously misunderstands.

In reference to the necessity of amalgamation at the Combination mill at Goldfield, I was not undertaking, as Mr. Hunt infers, a general discussion of the treatment "of oxidized ores of the desert regions of Nevada and California." The classification of 'desert ores' is Mr. Hunt's own. It seems to me a poor one, because all sorts of ores are found in the desert, requiring various modes of treatment; and variations quite as pronounced are found in mountainous regions and high altitudes. Hard ores are found in the desert, as well as soft ores; oxidized as well as sulphide; and any technical man familiar with desert conditions, especially those observable at Tonopah, Goldfield, and Bullfrog, would hardly fall into the blunder of making so meaningless a classification.

I was not generalizing when speaking of ore treatment at the Combination mine. I was plainly referring to a specific case of an unusual type of oxidized ore, peculiar to a limited district. Mr. Hunt advocates dry crushing as applied to the soft oxidized porous ores of the desert, under the impression, evidently, that this is the sort of ore found at Goldfield, and that it is characteristic of the deserts of Nevada and California. This is an error into which others have fallen who are not acquainted with the southern Nevada fields. The oxidized ores of Goldfield are not porous, and they are not soft; in fact they are extremely hard. The ore of the Combination mine has been described as a silicified andesite. It is so hard and tough and fine-grained that selected pieces of it have been considered suitable for replacing the flint pebbles in the tube-mill used for re-grinding the softer sulphide ore. Some idea of its hardness may be gathered from the fact that 1,300-lb. stamps, with 6-in. drop, falling 100 times per min. will, under the best conditions, put only 3 to 3½ tons per stamp per day through a 12-mesh wire screen.

Mr. Hunt bewails "the present fashion of putting in stamp-mills and amalgamating arrangements followed by a cyanide annex, as due solely to the compelling power of custom." If such is the prevailing practice, it is much to be deplored; and I regret it no less than he. But the allusion so papably insinuates that I have offended in the case of the Combination mill that I hasten to relieve Mr. Hunt's anxiety on this score by offering a few facts.

The Combination mill at Goldfield was not installed until the treatment of the ore had been throughly investigated This required about six months. Laboratory tests were confirmed by milling tests. Dry crushing was given a fair trial, but the extraction in each instance was poor; and after fifteen days' leaching with the ordinary strength of solution used on gold ores, it was still found possible to pan from the tailing considerable quantities of gold undissolved by cyanide. Several strengths of solution were tried; the ore was reduced to various degrees of fineness; and still dry crushing was not applicable. Finally wet crushing under stamps, followed by amalgamation, concentration, and cyanidation, was found to be the proper treatment, and was adopted. I do not propose to enter here into the details of treatment, as I am reserving this for a special article. But suffice it to say that contrary to Mr. Hunt's belief, the matter of ore treatment was given a thorough study; and we were not, at least in this instance, the slaves of custom.

In regard to the recovery of gold on plates; I would certainly advocate the use of amalgamation where as much as 50% of the gold can be so recovered. Mr. Hunt is possibly not familiar with the character of the gold found in the Goldfield district. It is fine, to be sure, as distinguished from the coarse gold of the California Mother Lode, which yields such high results to amalgamation; but it is not by any means invisible, and most of it can be 'panned out' in an ordinary miner's pan. In other words, it is too coarse for direct cyaniding and too fine for the best results by amalgamation. Moreover, Mr. Hunt has evidently not considered the great advantage of being able to secure daily from plates 50% of the extraction obtained from $50 ore, as compared with waiting for the tedious monthly or bi-monthly clean-up in a cyanide plant, and his suggestion that amalgamation be tried after cyanide treatment by passing the tailing "over some amalgamating device," while no doubt ingenious, will hardly appeal seriously to practical mill-men. And what, let me ask, does Mr. Hunt propose to do with the very high-grade concentrate obtained from this ore, which it pays so well to extract and ship? Must this also be removed after cyanide treatment?

My critic's views on this whole subject are summed up in his query, "As the majority of these oxidized ores give a practically complete extraction by direct cyanide treatment, what is the advantage of using both amalgamation and cyanide to recover the same value?"

My answer is that I do not admit the correctness of Mr. Hunt's premises. He is evidently an ardent advocate of dry crushing; and while I did not intend to prolong this discussion into a consideration of the respective merits of the two processes, I cannot refrain from mentioning some of the more serious limitations of dry crushing. A few years ago I was happily instrumental in converting one of the largest dry-crushing plants in the West (that of the Gold Road mine in Arizona) into a wet-crushing mill. It was an expensive and well-equipped plant and had been in operation several years; and so far as I could see, it had been worked to its highest efficiency. The extraction obtained was about 80%. The change to wet crushing, using Huntington mills in conjunction with the coarse rolls already installed, and Dehne filter-presses to handle the slime, raised the extraction 10%, and considerably decreased the cost of treatment. Amalgamation, in this case, was dispensed with altogether, as only about 15% of the value could be so recovered. Two other representative dry-crushing mills, one in Utah and the other in southern California, are now considering the remodeling of their plants for dry crushing, after obtaining promising experimental results.

These projects and changes are all based upon the following facts, which I believe will sooner or later reduce the applicability of dry crushing to a few rare and isolated cases: (1) The impossibility (from an economic standpoint) of reducing ore to a sufficient fineness to ensure the best recovery, without making percolation prohibitively slow. (2) The now generally accepted view that the product of any ore-crushing mill must be handled either as a slime, or as classified sand and slime. The slime requires short treatment with weak solutions; the sand, long contact with stronger solutions. Every modern mill, operating on a logical basis, must handle its final cyaniding material as two separate products.

Aside from the great discomfort to workmen in a dry-crushing mill, and the resulting well-grounded prejudice which they have against these mills, the great fundamental objection to this type is that it is not adapted to the requirements of the latest and best practice in cyaniding. An exception to this would be in the treatment of a porous ore containing little or no amalgamable gold, and requiring hardly more than breaking, like the ore formerly treated in the old Mercur plant, or an ore which requires roasting before cyaniding. Conditions so ideal for dry crushing are rare; at least they are not common enough to affect the general rule prescribed above.

Mr. Hunt mentions, among the objections to wet crushing, that the moisture in the ore must be displaced, entailing a loss of cyanide. This would be a very serious matter indeed if there were no conceivable objections to dry crushing. Each method, of course, has its objectionable features. The discussion of so important a subject should be undertaken in a fair and liberal spirit; the merits and demerits of each process carefully weighed; and obviously the final choice accorded to the method which has the preponderance of commercial advantages as applied to a particular ore. After a careful study of the two methods, and some experience in both, I, for one, am persuaded that the present recognized need of fine grinding in cyanide work, and the treatment of slime as a separate product, will ultimately reduce the field of dry crushing to a very limited scope.

FRANCIS L. BOSQUI.

San Francisco, May 10, 1906.

NOTES ON TUBE-MILLS AT EL ORO, MEXICO

By Charles Butters

May 26, 1906

Foundations.—The management of El Oro Mining & Railway Co. had a good deal of trouble with the foundation of one tube-mill, due to improper setting of the cement. There is a terrible vibration when the mill is running. It would be best to make the foundations 10% heavier than the plans usually called for, using the very best cement, and allowing sufficient time to set clear through before the tube starts. Underneath there should be a good cement floor sloping to a gutter leading to the cone, spitzkasten, pump, or wheel, constituting the return circuit, so that the floor can be washed with a hose, the washing being returned with all leakage of mills for re-grinding.

Rotation.—It is important that the tubes rotate in a direction such as to cause the thrust of the driving pinion to be downward into the pillow-block, and not up against the cap; the rotation should be effected by an open belt, if possible. If in accomplishing this the mill runs backward, it is best to reverse the spiral in the feed-throat, as the spiral is the only thing that makes any difference to the direction of rotation. The spiral at the discharge end can be removed altogether, as it seems to be of no practical value, mills working just as well without it, as others with it on the same work.

Leaks.—A leak will often start around a bolt when it is impracticable to shut the mill down. If there is a cement floor it will make no difference, or a bolt may cut out entirely. In this case, feel in the hole with the finger and if the liner is good, put a wooden plug in the hole until such time as is necessary to shut down for more extensive repairs. If the leak occurs at the head of the mill, the sand will get into gears and cut them badly. This can be obviated by putting a shield on the gear-wheel rim, to cover outside ends of teeth, by fastening (with cap-screws) a ring cut from sheet-iron No. 14 or 16 gauge. This should, by all means, be done before the mill is set up.

Feed.—In case the feed-hopper and the pipe that enters the mill come in one piece (as they did at El Oro), cut the pipe off as close to the hopper as possible, and put in a flange or sleeve, threaded in direction, so that the rotation of the mill will not screw it off. The pipe will have to be made a little longer than the piece cut off, as the connection will necessitate setting the hopper further back, in order to get at the packing gland. There is a great deal of wear on the pipe. By this means it can be quickly and easily replaced. The best packing to use is ⅜-in. common hose cut in individual rings. There should be a de-watering cone, or pulp-thickener, directly over the feed-hopper, to get the pulp as thick as possible for the mill—one to one, or less. The overflow from this can be carried over the mill into the discharge-box to dilute the pulp again so as to make it flow through launders.

Platform.—There should be a platform over each mill, with a hopper over each man-hole for putting in pebbles, and the pebbles should be brought in at this level if possible, to avoid elevating them. The lifting of pebbles into the side of the mill is a slow and laborious job. This platform would give access to the pulp-thickening cone also. With everything arranged in the most convenient manner, the mills should run a maximum of 29½ days per month. If there are only a few pebbles to put in, two or three men will distribute them easily, but if there is a great quantity to add, it is quicker to fill the mill full up to manhole, put on the door temporarily, and give the mill a few turns to distribute them, repeating the operation until sufficient pebbles have been introduced. At El Oro they keep the mills one to three inches more than half full, by measurement from top tube to pebbles. When the mill is opened for repairs the foreman measures the pebbles and can tell the workmen just how many sacks to add. They never allow them to wear down more than four inches before refilling.

Liners.—The Krupp liners at El Oro are 1½ in. thick, of chilled cast iron; the chill is only about ¼ in. deep, and when it wears off, the balance of the plate wears much faster. The wear on the plates is chiefly at the head of the tube, and diminishes with each row of plates toward the tail. The average life of Krupp plates being 40 days, with mill No. 3, running 32 rev. per min., containing 7½ tons of pebbles and grinding 125 tons of sand per day, of a very hard quartz, producing about 50% of material that will pass 200-mesh, 40% of the remainder being between 100 and 150 mesh. They are now making their own plates, 1¼-in. thick, of white cast iron, not chilled, but it is run in very thin sand molds. The founder has a formula for this casting, which produces a very hard metal clear through, and wears evenly and equally down to a quarter inch or less. It was not possible to get the exact composition of the iron, but any intelligent foundry-foreman could soon test it out, as a hard white casting is made by using re-melted iron until it is very low in silicon, and adding a little sulphur. No. 3 mill has four rows of plates, with three holes each. The length is to be changed, making three plates with two holes each to take the place of two plates, and make them 1½ in. thick. At El Oro they never empty the mill to re-line it, but put in one or more plates at a time as needed, with the pebbles all in. Thus the 1½-in. short plate will be lighter to handle than the 1¼-in. long plate, it will last longer, and if one breaks before it is worn out (as they sometimes do), there will be less iron to throw away. These plates wear much longer than Krupp plates, and cost just half as much. The plates must be cast true to curve, the holes very carefully cored, and the bolts forged from common iron to exact taper of hole. When the plates are put in, see that the core sand is thoroughly cleaned out of hole; place two men inside and two outside of mill; take two rods 2 ft. 6 in. long to pass through the holes, so that the men outside can help to lift the plate, the rods guiding it to exact position. The men inside can easily hold it in place, while the rods are removed, one at a time, and the bolts put in. Drive the bolts to a firm and close seat, with a sledge, while the nut is being tightened with a 2-ft. wrench, using a washer and gasket of ¼-in. sheet-packing under the nut. If the plates are well put in so there is no spring and the bolts fit perfectly, they will wear evenly down to ¼ in., without breaking or causing any trouble. Four men, after they become familiar with the work, will take out old plates and put in new at the rate of about three per hour.

At El Oro they have found that the end-discharge plate or grating discharged pebbles too large, causing undue waste, and have thus replaced these with others containing oblong holes ⅜ in. by 1½ in., there being 48 holes in each half plate An experimental run has been made with mills in series, that is, running all the coarse sand into one mill, and the fine return sand into a second mill. The consensus of opinion is that this is the best and most economical method and they are installing two new mills, to be put in series with the old ones. The pebbles discharging from the first mill are put into the second, where a fine discharge grate is used, the pebbles finally issuing being not more than ⅜ to ½ in. The discharge must be into an open box containing a screen with about ¼-in. holes to catch the pebbles. They have in use a No. 5 mill, 5 ft. diam., 27 ft. long, inside measure, running 28 rev. per min., containing 16 to 17 tons pebbles and 8,640 kilo. liners, requiring 107 h.p.; a No. 4 mill 5 ft. by 24 ft., 28 rev. per min., containing 11 to 12 tons pebbles, 7,520 kilo. liners, and requiring 85 h.p.; and a No. 3 mill, 4 ft. 1 in. by 19 ft. 9 in. running at 32 rev. per min., containing 7½ tons pebbles, 4,520 kilo. liners, requiring 60 h.p. They have determined, after several months' careful observation, that the No. 3 mill is the most economical and efficient to use, and, in future, they will install no other size.

The consumption of pebbles and liners for the months of October and November, 1905 (mills running 95% of the time) was:

Mill No.	October Pebbles Kilo.	October Liners Kilo.	November Pebbles Kilo.	November Liners Kilo.
3	5,165	1,365	5,370	1,360
4	8,625	729	10,625	2,589
5	7,987	1,345	16,310	2,589
Total	21,777	3,439	32,305	6,538

The mills were grinding about 8,700 tons per month of battery sand, besides the return sand.

In my opinion it would be a great advantage to have the liner plates short, as mentioned above, for there is little room in a No. 3 mill; no more than two men can work to advantage, and the plates should be light enough so they can be handled quickly and easily. But it would be a great mistake to make them with only two holes, because if a bolt cut out, the leverage of the plate on the other bolt will soon break it, causing an endless amount of trouble, necessitating a shut-down to replace a bolt at once. With three bolts in a plate, if one comes out, the other two will hold the plate firmly in position until such time as is necessary to shut down for more extensive repairs.

METALLURGICAL DEVELOPMENT ON THE RAND

By G. A. Denny and H. S. Denny

(June 2 1906)

As far back as July, 1903, one of the writers had the temerity to traverse current practice and make many alterative suggestions representing departures of a radical nature. These recommendations were substantially: (1) To substitute coarse for fine screen on batteries. (2) To discard all mechanical concentration. (3) To separate the coarse product by hydraulic devices. (4) To re-grind coarse products in tube-mills. (5) To provide secondary amalgamation tables after the tubes. (6) To improve the methods of inter-handling in slime-plants. (7) To make use of the filter-press in the treatment of slime. (8) To investigate the economical possibilities of reducing the whole of the mill-pulp to a fitness suitable for filter-pressing, and thus treat one product in one operation.

The point upon which the greatest emphasis was laid was the necessity for fine grinding, and overwhelming evidence was produced to prove conclusively that the pyritic constituent of the Witwatersrand ore, after being subjected to fine grinding, yielded its gold to an ordinary cyanide solution as readily as the silicious portion of the ore. It was further stated that by the use of the tube-mill the duty of the stamps would be increased and the extraction improved.

As the result of these suggestions, the first tube-mill on the Witwatersrand was introduced at the New Goch mine, and this one was followed a little later by several others at other mines. Since that time there has been a good deal of experience gained in the use of the tube-mill, but it is a significant fact that the recommendations above specified in regard to the tube-mill, secondary amalgamation, coarse screening to get increased stamp duty and higher extraction, have been followed and generally accepted in this mining centre. The further suggestion to filter-press has not been followed outside of the group of mines technically controlled by the writers, and many other departures have since been jointly advised and put into practice at the Meyer &

Charlton and the New Goch mines. It is on the results of the work done at these two mines that the writers propose to base the whole of the statements to be embodied in the present contribution. The real metallurgical work began in January, 1906, and we have therefore had three months in which to go into detailed investigation.

The consumption of water at the Meyer & Charlton is now 170 gal. as against an average of 500 gal. at other mines. The actual saving is illustrated by the fact that whereas before the new plant was installed the amount of water available was only just sufficient to meet the daily requirements, there is an excess today of over 100,000 gal., equal to some 300 gal. per ton of ore crushed. The consumption in the metallurgical plant is 36.8 gal. per ton of ore crushed

At the Meyer & Charlton plant the average cost of cyanide in March was 4.19d. per ton treated, the actual consumption of cyanide being half a pound, equal to a total of 5,376 lb. The cost of lime was 0.157d. per ton treated, the actual consumption being 0.8 lb. per ton treated, the total consumption being 8,460 lb. The total amount of solution in circulation amounted to 2,133.3 tons, which had an average strength of 0.038% KCy, and an average value of 2 dwt gold per ton, the total cyanide contained being 1,638.05 lb., and the total gold content 213 oz., the alkalinity averaging 0.008 per cent.

The average amount of gold dissolved in the mortar boxes is 1.37 dwt. per ton of ore crushed. The effect on the plates is marked, and at the Meyer & Charlton it was such that the lower portion of the plates has been cut off, as it was found that they were doing no good as far as arresting gold was concerned, while on the other hand they were showing signs of pitting and gradual disappearance in the circulating solution. These plates, however, are 12 years old, and were thin when the circulation of solution was begun. It is believed that an electrolytic action is set up as between the mortar-boxes and the plates, the cyanide solution being the medium; hence the solution of the copper. A number of investigations on the point have been carefully carried out. It is estimated that the life of copper plates with circulating cyanide solution will be something like three years, and it may be possible by the short-circuiting of the currents set up between the boxes and the plates to reduce the solvent action. Assuming, however, that the life of the plates is only two years, and that the plates themselves are only half the size of those generally adopted, the cost per ton becomes insignificant.

The recovery by amalgamation now as compared with the old system, illustrates clearly the effect of taking the fine gold into solution, the respective figures being: Recovery by amalgamation in January, 1905, 50.124%; recovery by amalgamation in January, 1906, 45.202 per cent.

There is one aspect of amalgamation worthy of mention, and that is the danger of theft, for it is a fact that there is a big illicit trade in amalgam, and the writers are of opinion that amalgamation could be dispensed with altogeher under this scheme. The point has not yet been proved, but it can easily be put to the proof, and this will be done in the near future. It would be safer to have all the gold in one place, namely, the extractor boxes.

Originally it was hoped that filter-pressing would be unnecessary. This hope was based on the assumption that a product could be discharged continuously from the last vat, which would contain not more than 50% moisture. After some years of experience the conclusion was reached that this was impossible, and at the Van Ryn mine, two years ago, an arrangement was built for returning solution from the slime-dam. After careful consideration, however, and after giving due weight to the maintenance of slime-dams, cost of pumping back solution, losses by absorption, and the danger of the dams giving way, there could be no further doubt that filter-pressing must be adopted to complete the operation. The slime-plants at the New Goch and Meyer & Charlton are now simply being used as storage-tanks; that is to say, the whole of the available gold is in solution before the slime reaches the slime-plant, and, therefore, it is not called upon to do anything in that direction. At the Van Ryn, where solution is not circulated, the slime-plant is showing over 93% of the total gold in solution in the first four tanks. At the Charlton, however, with the circulating solution, this gold has already been absorbed before the slime-plant is reached. The superiority of the design of this plant over the ordinary type of decantation plant, is that the inter-handling can be done simply by the regulation of a valve and at no cost; and as a storage plant, to be worked jointly with a filter-press, it has many points of advantage.

In view of the fact that the circulating solution has already been in contact with the sand before it reaches the sand-plant, the value of charges under present conditions shows a big reduction, and from an 8-dwt. screen-assay the present condition gives a return in the settlers of under 1.5 dwt., and in the treatment-tanks of 1.4 dwt., which, under old conditions, would, in the latter, have been in the neighborhood of 5 dwt. The sand-plant consists of eight vats in all, four of which are superimposed and are called the 'settlers,' the four below being called the 'treatment-tanks.' As a matter of fact, however, solution is being pumped to the settlers as well as to the treatment vats almost constantly, and by this means, instead of a treatment of four days, which would have obtained under the old system, a treatment of nine days is obtained. The sand plant is only responsible for 10% of the total gold recovery, which when set against the cost of operating the plant and the interest on capital outlay, at once proves it to be a most expensive department. If the interest on capital outlay and the actual operating costs of this plant be allowed as a credit toward the cost of all sliming, it leaves a big margin in favor of fine grinding, and the intention of the writers on this evidence is entirely to discard the department so generally devoted today for the treatment of sand. Not only this, but the residue from the sand-plant has neither the consistence nor the low value of the filter-press product, and sufficient investigation has been done to indicate that if coarse sand be reduced to 150-mesh it can be filter-pressed just as well as ordinary slime. The average residue then, instead of being in the neighborhood of 12 gr., would be reduced to, at most, six grains per ton.

On a basis of 96 tons per day passing through the tube-mill the extra revenue is £15 15s. 7d. per day for each mill in use. The deduction generally is that fine-grinding is a necessary corollary to any metallurgical treatment of Rand ore, and the evidence points to the further extension of this principle.

Experimental data show that filter-pressing is one of the most necessary operations that has yet been introduced into local practice. The points are the great saving of water, the increased extraction, the saving of cyanide, the economy of time in treatment of slime, and the production of a better product for handling on the slime-dump. The operation is one of the very simplest, and experienced white men who have been filter-pressing for many years in Western Australia are quite satisfied that the native is just as good a filter-pressing hand as the white man. The reason why this method was not adopted earlier was due to the fear of excessive cost. Our figures not only disprove this idea, but show that filter-pressing is the most important operation in the whole range of treatment. The average residue from slime during March at the Charlton contains 0.2886 dwt. The fineness of the gold was 753.-962, and the amount of acid used 4,988 lb., equal to 1.304d. per ton treated; and the amount of zinc used 3,834 lb., valued at £63 18s.

The addition of a filter-press plant to the mill settles one point that is usually subject to inaccuracy, and that is the tonnage of slime handled. The relation of extraction to tons multiplied by screen-assays has not, in the past, been so close that this point has come up for serious discussion, but in the future there is no doubt that more attention will have to be given to it:

The cos s for the n. :th of March are as follows:

	Total cost Pounds.	Cost per ton milled. Pence.	Cost per ton treated Pence.
Tailing-elevator	279	6.23	6.23
Sand-treatment	447	9.99	14.08
Slime-treatment	362	8.08	27.82
Extractor-house	436	9.73	9.73
Total cost	1,524	34.03	57.86

It will be noticed from this statement that the cost of elevating the pulp is high, figuring at 6.232d. per ton milled. Apart from this cost, the actual expenditure is 27.811d. per ton milled. When it is considered that this covers the whole cost of sand-treatment and slime-treatment, including filter-pressing and the cost of dumping residue, this result must be regarded as eminently satisfactory for a new plant.

As already indic: d, the sand-plant is only responsible for 10% of the gold recovered; while the slime-plant is doing nothing, and therefore is not required. The highest residue is in the sand-plant, and the costs of running it are also high. The final conclusions on actual data are (1) that the credit of the cost of re-grinding to be established by discarding the sand-plant, and, allowing for the interest on the capital outlay and the actual working costs of the sand-plant, is more than is required to cover the whole of the cost of re-grinding; (2) that the extraction on the one-product basis, where everything would be filter-pressed, shows a further improvement in favor of all-sliming; (3) that a plant on the all-sliming basis could be erected complete at somewhat less than 50% the cost of the ordinary sand and decantation slime-plants of today, to give a much higher and far more consistent recovery, and to be worked at a much lower cost. In the design of the all-sliming plant the stamp battery is left out, the principle being crushing and grinding in stages.

The grinding analyses of the products show some curious features. One in particular is that the grading of a settled sand is quite different from the grading of the charge which that sand represents when transferred from the top to the bottom tank. It is difficult to determine exactly how this change is brought about; but, as the results have been confirmed in many investigations, there can be no question that the change does take place. It may be said that work could not be carried out satisfactorily without grading analyses.

Shortly after starting the plants, the residue value, both in the slime and the sand plants, occasionally was high. In every case, it was found that high residues were due to reasonable causes, such as could easily be controlled when the operators had a full knowledge of the conditions. There is no reason why the filter-press residue should ever be higher than 6 gr. or the sand-residue be higher (given that the tube-mills are running) than 12 gr., and this quite apart from the screen-assay. Taking the case of the Meyer & Charlton, however, which in March had an average screen-assay of 10.871 dwt., the average residue in the sand was 144 gr., and in the slime 6.91 gr. The percentage of the total pulp represented by sand was 70.763, and the slime 29.050. This gives an average residue of 12.39 gr., equal to an extraction on the screen assay of 95.302%. The total gold cleaned up shows an actual extraction of 95.5%. In this plant 96% theoretical extraction should be easily and consistently realized, and allowing for 1% loss in the smelt and clean-up, etc., the actual gold in the bank should be 95 per cent.

THE TREATMENT OF DESERT ORES

(June 23 1906)

The Editor:

Sir—I have read Mr. Bosqui's article in your issue of May 26, in reply to one of mine in your issue of Arpil 28, and crave space to rectify a few points.

The term 'desert ore' was not used by me as a term of classification of ore; it meant simply ore found in desert regions, and referred particularly to the conditions of treatment there. I did not refer specially to Mr. Bosqui's installation of the Combination mill or to the ore of any particular district, but referred generally to the treatment of soft oxidized ores. Mr. Bosqui's description of the Combination ore, as one in which the oxidized portion is much harder than the sulphide ore, shows it to be an unusual occurrence.

"The latest and best practice in cyaniding," quoted by Mr. Bosqui, would be more correctly termed the "last fashion in cyaniding in South Africa." It is notorious that what was called "double treatment" was boomed some years ago in South Africa, and in consequence a number of double-treatment plants were installed in this and other countries; and they were failures. More recently, the South African practice was to divide the pulp into 'sand' and 'slime' to be separately treated, and this appears to be the ideal treatment for "every modern mill operating on a logical basis," according to Mr Bosqui. The most recent practice, however, at Johannesburg, is to fine-grind and filter-press the whole pulp, and thus get "one product in one operation," the advantages of which are most clearly and interestingly shown in the abstract of Messrs. Denny's article given in your issue of June 2.

I shall not take up space by replying to Mr. Bosqui's statements on ore treatment seriatim, but I must traverse one statement regarding "the impossibility (from an economic standpoint) of reducing ore to a sufficient fineness to ensure the best recovery" by referring to the fine-grinding of cement-clinker, which is done on an enormous scale and at a low cost.

It appears to me that the "best practice" is to be able to interpret laboratory results on any ore, so as to install the kind of plant economically best suited to its treatment, without reference to what may be the practice in South African or other fields, otherwise than to take advantage of new discoveries in grinding machines, etc. In the case of porous oxidized surface ore it appears to me that the best practice would be to crush dry to a fairly coarse mesh and leach direct in vats, the residue being subsequently passed over amalgamating plates or concentrators, if found to pay. In the case of ore which requires fine-grinding to release the metals, I would fine-grind either dry or with cyanide solution, agitate, and filter-press the whole pulp, thus getting "one production in one operation." The point on which I wish to lay most stress was that when cyanide is used at all, it is more logical to treat with cyanide first and amalgamate (when necessary) afterward, than to follow the ordinary procedure.

Finally, I wish to deprecate most strongly the introduction of the personal element, as so patently shown by Mr. Bosqui. I take it, these discussions should be purely impersonal and for the advancement of the art of metallurgy and the interest and instruction of your readers, but not for the ventilation of any personal feeling between professional men.

BERTRAM HUNT.

San Francisco, June 9, 1906.

PROGRESS OF CYANIDATION

(Editorial, July 28, 1906)

The comment on current practice at Kalgoorlie by Mr. Alfred James, will interest cyaniders. We also publish a description of a new method of lining tube-mills, as devised by Mr. H. P. Barry, of the great Waihi mine, in New Zealand. One of the practical problems, especially to millmen operating at a distance from manufacturing centres, is the cheapest and most durable lining for the tubes. At first everyone depended upon Iceland pebbles and silex lining from Europe; and to those in charge of mills on the other side of the world—in Australia, New Zealand, and Mexico, for example—it was a serious handicap to depend on supplies from so great a distance. Iron linings of special composition were not much better, for they also were made by methods not available at the ordinary mining town. Both in Mexico and New Zealand the mill-managers have succeeded in breaking away from the tyranny of a special material and they have been able also to procure flints suitable for their purpose. The device invented by Mr. Barry is one of the best improvements in tube-mill practice, and, as it is applicable to other localities, we hope it may prove extremely useful.

In regard to cyanidation generally, Mr. James' remarks are much to the point. The frequent comparisons between the work done by the pans and tube-mills at Kalgoorlie have evidently been of little value as indicating the general applicability of the two machines, simply because the peculiar conditions obtaining at Kalgoorlie have been overlooked. Mr. James sets that matter right. His other criticisms are welcome, and any reply to them will be given a courteous hearing.

It appears likely that tube-mills will be discarded at Kalgoorlie because the tellurides occurring in the ore require roasting; the idea used to be to comminute excessively so as to dissolve these tellurides raw, but now solubility is secured by obtaining the porous condition due to a roast. Filter-presses are already a back number in Western Australia, and elsewhere. Filter machines of the Moore, Butters, and other types are displacing the old devices. In fact, cyanide practice is undergoing a continual and rapid development, one device after another being elbowed out of the way to make room for something better. Just now it is manifest that it is cheaper to treat slime by agitation than to treat sand by percolation; it is not only a question of relative cost between re-grinding *plus* slime-treatment on the one hand and percolation of the coarse stuff on the other; the assertive factor is the increase of crushing capacity in the first instance—whether under stamps or between rolls—due to relieving the first crusher of the work of pulverizing. On these subjects we hope to hear from our friends. Experiments are continually under way; the exchange of experience will prove mutually helpful.

TUBE-MILL LINING

(July 28 1906)

Those who operate tube-mills will not need to be told of the want of a cheap and durable form of lining, for this is one of the practical problems in this branch of milling. At the Waihi mine in New Zealand, the tube-mill has proved most effective for re-grinding and the practice has been developed under the direction of Mr. H. P. Barry, the superintendent of works. Formerly at Waihi they crushed under stamps to 40-mesh, and the finer the pulp discharged from the battery, the better the extraction in the cyanide annex; but there was a limit to fine crushing, for beyond 40-mesh the duty of the stamps was decreased until excessive pulverization was no longer economical. Now the stamps crush only to 15 or 20-mesh and the battery product is re-ground in tube-mills to 150-mesh. There was lots of trouble and expense in connection with the lining of the tubes for the distance of New Zealand from Europe represents a maximum length of transport. The silex from Iceland had to be laid with a special cement and the fitting of the pieces was troublesome. It cost £80 to line one tube-mill, and the time of service was only three months. Now Mr. Barry employs an invention (patented) of his own, whereby he utilizes the flinty portions of the Waihi ore and lays them in portland cement; the lining lasts six months and costs £40—the service is twice as long at one-half the expense, it is four times cheaper.

This lining is illustrated herewith, by courtesy of Mr. Charles Rhodes, general manager of the Waihi Gold Mining Co., Ltd. It is called Barry's honeycomb liner and it is formed of cast-iron segments, curved to the shape of the tube, in sections having four or six divisions, each four by six inches. In actual operation it has been found that the costs are:

Honeycombed casting, 28 cwt. @ 15s.	£21
Broken quartz, 5 tons @ 20s.	5
Portland cement, 1 ton at mine @ 11s. per cask	4½
Sand (rough tailing), 1 ton	1
Labor @ 9s. per day	8
	£39½

We are informed that the Waihi company is making the casting in its own foundry for 12s. per cwt. and that the cost put down for broken quartz is rather excessive, so that the liner can be made for £30. The saving is about £260 per tube per annum, as compared to the old method. As the Waihi company is about to use 10 or 12 tube-mills, the saving will be important, about $15,000 per year.

We quote herewith from the specification of Mr. Barry's patent, and accompany it with two of the diagrams that accompany that claim for patent.

The purpose of this invention is to provide a cheap and effective grinding surface within tube-mills and other grinding-machines.

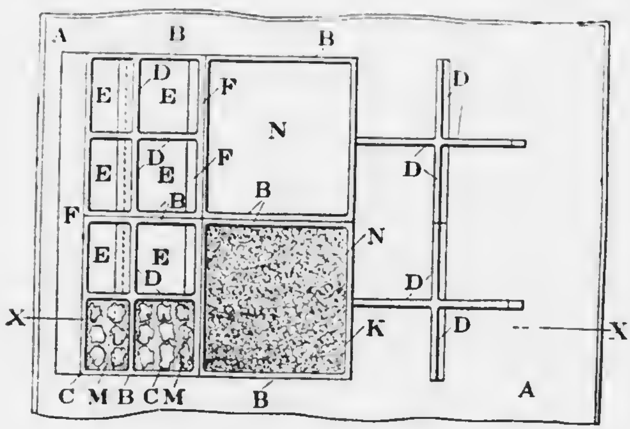

Fig. 1. Tube-Mill Lining. Plan.

This purpose is attained by fitting within the tube-mills shallow frames or boxes of hexagonal or rectangular form shaped as is hereafter stated, or in any other shape which will conveniently fit, and in fixing therein a material composed of rough blocks of stone or other substances suitable for the purpose. The accompanying drawing shows 16 figures of which Fig. 1 is a plan illustrating a portion of the inside of a tube-mill and showing different formations of frames or segments with material fixed in parts, while Fig. 2 is a cross-section through X-X of Fig. 1.

The tube-shell A of the grinding-mill is shown in Fig. 1 and 2 in part and broken, as it is only necessary to show a portion to illustrate the different formations or segments *B* fixed and which may be fixed therein. Fig. 2 being a cross-section only, the depths of the formations or segments *B* are shown, but Fig. 1 being a plan, their shapes are shown more fully so that the idea of their purpose can be fully seen while the rough stones *C* shown in both figures quite illustrate the advantage of their use.

The frame and box formations *B*, hereinafter called segments, are made with or without bottoms of iron, steel, or other suitable material and in convenient sizes shaped to fit inside the tube-mill, as shown in Fig. 1 and 2, and to retain the lining material in position. The segments *B* may have ribs or bars *D* preferably shaped across to help hold the lining material *C* in place and so as to limit and reduce the size of the segments *B* into smaller compartments *E* to suit the particular nature of the lining material used, or rib or bar formations *D* may be used by themselves without outer frames.

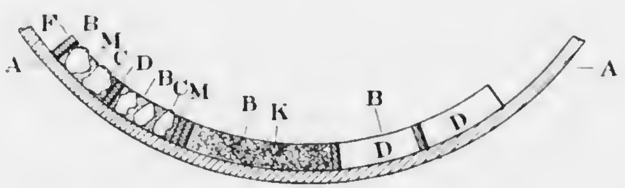

Fig. 2

Fig. 2. Tube-Mil. Lining. Section.

The outsides *F* of the segments *B* may be formed plain and made to the radius of the tube-mill and so form an arch which will key each segment *B* in position, or the outsides of the segments *B* may be rabbet-formed so as to interlock with grooves therein into which locking bars of iron or steel or other suitable material may be put. The segments *B* may be bolted or otherwise suitably connected to the tube-mill, but as a rule this will be unnecessary.

To prepare the segments *B* for use they are placed upon a firm surface approximately of the same radius as the inside of the tube-mill when they, the segments *B*, are well filled with the lining material either in the form of concrete *K* or of rough blocks of stones *C*, the stones being jammed in tightly and the interstices between them being filled and rammed with cement, concrete or cement mortar *M* or suitable material which can be poured in, in the shape of groating, if the spaces between the stones are small. The lining material may be fixed to be quite rough as shown at *C* in Fig 2 and it will preferably be made to project in irregular formations above the edges of the segment *B* and ribs or bars *D*, as it will thereby protect the edges from undue wear, and by its irregular surface assist the grinding capacity of the tube-mill or machine. The lining material may be of quartz or any hard rock or material that will do the work required of it

After giving the cement concrete or the cement mortar *M* or concrete *K* or other lining material in the segments *B* ample time to set hard, the segments *B* are built into the tube-mill or machine with cement or other suitable binding material and the locking-bars are driven in when found necessary to use them. If found advantageous to do so, the segments *B* may be built into the tube-mill or machine empty in the first place and then the different compartments can be filled with the lining material in the manner above described. When the different segments *B* have been fitted within the tube-mill or machine and the lining material filled and fixed therein, the tube-mill or machine will be worked in the usual way but with a much more effective and marked result than is now obtained without the use of the lining material herein specified. After the tube-mill or machine has been working for some time, the lining material should be examined, and if any of it is found to be faulty or unfit for use, the segment *B* into which it is fitted or fixed can be re-fitted by having the faulty lining chipped, broken, and taken out and fresh lining material filled in, in the manner already described.

CRUSHING AND GRINDING PRACTICE AT KALGOORLIE

By Alfred James

(July 28, 1906)

At a moment when one hears so much as to the relative advantages of stamps and the ball-mills, tube-mills, and pans, one is apt to overlook the particular practice which has led to the introduction of a special type of machine to effect a certain definite reduction on a definite class of ore. Thus, for example, pans were introduced at Kalgoorlie to collect coarse gold from the furnace product, to get rid of soluble salts before cyaniding, and to grind coarse on sintered particles of ore. For this purpose pans have proved remarkably successful and if the all-roasting process had continued unchallenged, probably they would have maintained their superiority unquestioned.

But the advent of Dr. Diehl and bromo-cyanide changed all that, or rather, for a time, threatened to do so. Dr. Diehl claimed to be able to do away with roasting entirely, and to attain this object a new type of ore-reducing machine—a fine slimer—was necessary, and he introduced the tube-mill for this purpose.

It is necessary at the outset to bear this distinction in mind, for if the all-roasting process becomes once more universal or if it is possible by closer or improved concentration to reduce the refractory material (that is the sulpho-tellurides) in the pulp to an inappreciable or immaterial amount, then it may be reasonably expected that, with the absence of the necessity for fine-sliming, pans will prove advantageous rather than tube-mills on account of their lower horsepower per unit, their greater convenience in working, their capacity for amalgamation, and the granular nature of their product. These remarks refer to the Kalgoorlie practice only and not to districts where, for example, the finest comminution possible per horsepower unit is necessary for effective amalgamation or for a high extraction generally.

Therefore, similarly, given an ore which yields the most profitable result by roasting, we can scarcely expect stamps to be preferred for crushing purposes, because the question, for example, of drying the ore is of greater moment than the less repairs required by stamps as compared with ball-mills. The latter, moreover, appear to have the further advantage of requiring less power per ton crushed, yielding as they do an output of two tons of hard ore per h.p. day through 30 to 40 mesh or 2½ tons per h.p. day through a 25-mesh screen—a record unapproached by stamps, which require re-grinding appliances absorbing considerable power when large outputs (7 to 10 tons per stamp) are to be attained. The best practice known to the writer in this direction with the ordinary gravitation stamp is 336 tons coarse crushed by 40 stamps (1,250 lb.) for 120 h.p. plus an extra 60 h.p. for the tube-mill necessary to crush the 240 tons coarse product to, say, 30 to 40-mesh, or under two tons per h.p. day.

It thus appears that stamps will be preferred only when the ore yields a considerable proportion of gold on the plates and when roasting is not to be adopted. At Broken Hill, where there is no amalgamation, the crushing being preparatory to wet concentration, they even prefer to wet-crush in ball-mills rather than stamps, but sufficient data are not yet available to establish the economy of this practice.

As to the relative efficiency of the ball and Griffin mills, apart from the questions of cost, power, and repairs, the regular granular product of the ball-mills appears so much superior for roasting purposes to the cement (fine plus coarse) granules of the Griffin mills, that it looks as if the latter would not continue in favor, though one can quite understand their installation years ago when dry-crushing results were less established.

And now as to pans versus tube-mills: If better concentration can show a reasonably free cyaniding product, then the necessity at Kalgoorlie for tube-mills disappears with the theory of a complete non-roasting treatment, which, as stated above, was the primary cause of their adoption. But it is by no means yet decided—rather the reverse—that concentration, as practised in this district, is yielding a product capable of being bromo-cyanided without the very finest sliming, such as it appears tube-mills only can as yet accomplish. It must be remembered that even the finest sliming gives only comparatively fair results on the material that is bromo-cyanided. It was the amalgamation recoveries from the Old Lake View and Hannan's Star plants which brought the percentage recoveries to so high a figure, and an attempt to obtain Dr. Diehl's bromo-cyanide extractions by bromo-cyanide alone without Dr. Diehl's bromo-cyanide method—tube-mill fine-sliming—is apt to yield as poor results as are shown by one of the best known mines at Kalgoorlie. Even though the ore of this mine is supposed to be comparatively free from refractory minerals, the residues are reputed to be the highest in the locality—2½ dwt. per ton.

Surely, in comparing the relative efficiencies of pans and tube-mills, an extraction test would have been more useful, more accurate and more practical than the pan versus tube-mill test published by the Ivanhoe Company. Of what value is a 150-mesh minimum when dealing with the classification of tube-mill products?

But with regard to these tests, they were not referred to more definitely in my remarks published in *The Engineering and Mining Journal*, because it was evident from the immatureness of the methods that the results of the tests were unworthy of serious consideration. One's chief regret was that names of such high standing should have been tacked onto tests so badly carried out. To those conversant with tube-mill work, it is unnecessary to insist on the absolute need of keeping the feed-pulp free from slime and for taking the finished product away from the mill immediately it is formed; but the Ivanhoe not only included a huge proportion of slime in its feed—returning the finished product to the mill instead of taking it away at once—but, by the intermediate working of the test on alternate days, actually had the flints coated with slime at the start of each daily test.

We have the extraordinary position of a man who discovers that an original feed or more than 26 tons per day chokes his tube-mill, actually feeding no less than 26.77 tons of finished slime (finer than 150-mesh), or more than he takes out (25.15 tons) for his daily output. Could any one expect a mill thus choked to do effective work? Can anyone longer wonder why the results show only a fraction of the work done per h.p. unit consumed, compared with that done elsewhere by the same h.p. unit? And so it happens that we have seen published all over the earth, results which must be due to the enthusiasm of inexperience, rather than to any attempt or desire to deliberately mislead.

The actual daily *production* of the tube-mill choked with fine slime was only 16.74 tons of '—150' slime, although the daily output—including the returned slime re-ground—was no less than 43.46 tons. Could anything be more misleading than figures made out on such a basis? One is tempted to wonder why the metallurgist responsible did not go a little further and reduce the position *ad absurdum*, by returning all his finished product, thus proving that the tube-mill was doing no work at all!

The tests were thus unworthy of serious notice mainly because the lack of opportunity for the free escape or removal of the finished tube-mill product reduced them to an absurdity. On the other hand the pans—especially the first pan—were allowed a much better opportunity of getting rid of their finished product, though even here the arrangements might have been improved. The first pan received only 7½ tons of slime with an original feed of 21.3 tons of sand, as against the tube-mill quota of 26.77 tons of slime with an original feed of 19.5 tons of sand only. Had some more efficient comparison of the product than the 150-mesh screen been available, the effect of the continuous re-grinding of the tube-mill finished product must surely have been apparent.

In the above remarks reference is made to the slime grinding tests only. I have already shown elsewhere that given the ordinary costs of flints and liners in place of the extraordinary costs stated, the advantage lay with the tube mill even apart from the considerations of limitation of output referred to above; and this remark refers also to the fine grinding (breaking down coarse sand) experiments. Setting flints and liners down at 1.75d. to 2d.—which is the ordinary cost—the relative costs per ton ground are 8.86d. for tube-mills, as against 10.32d. with pans.

In this connection it is curious that in Western Australia they should still use cast-iron linings, which appear to have become obselete in other places. In the recent discussion at Johannesburg Mr. Leupold, general manager of the Treasury mill (using the Krupp tube) stated that he entirely concurred with Mr. Dowling as to the superiority of flint liners, "which roughly cost one-half and last twice as long as the chilled iron ones."

Pans have proved themselves reliable, efficient, and convenient where roasting is employed, but there appears as yet no proof of such efficiency on unroasted sulpho-telluride tailing; on the contrary, the evidence to date seems all in favor of the use of the tube-mill on such material.

By the way, one cannot but be surprised at the difference of opinion obtaining locally as to the working of pans. No two neighboring metallurgists seemed to be of the same opinion. Thus at the Ivanhoe the compensating-weight rings are highly thought of, and much has been published about them, but at the next mine visited, I was informed that the compensating-weight rings had been thrown out, as having proved of no advantage whatever. At the next mine I was told the old original 8-ft. pans first introduced were the best pans on the field, and these 8-ft. pans are being adopted for the most recent plants. There is need for authoritative work to be done in this matter, and I hope that Messrs. Klug and Taylor will continue their valuable investigations into the subject.

A curious result of the introduction of electric power is that ball-mills (No. 5) formerly rated at 17 to 24 h.p. (steam) are now rated at 40 h.p. (electric). As a little over 200 h.p. (steam) is running nine of these mills at Kalgoorlie with three fans and all the shafting, and as 24 h.p. is about the highest figure indicated for a No. 5 Krupp mill with full charge and running 10% over normal speed and including shafting, etc., there must either be a lamentably poor efficiency of the electric motors or else the electric rating is inaccurate.

PANS V. TUBES

(August 1, 1906)

The Editor:

Sir—It has occurred to me, in the case of many of the controversies over the respective merits of grinding pans and tube-mills, that no real basis for argument would be left if more attention were paid to defining the work to be performed by each machine. For instance, is total sliming of the ore the principal consideration, or is sliming only to be the final stage in the crushing process?

In the Transvaal, total sliming—or at least total fine grinding—of the stamp-mill product seems to be the desired end, while in Kalgoorlie stamp-mill practice total sliming is wanted only after a preliminary concentration has been effected. Both grinding-pans and tube-mills are in use here, and each fills satisfactorily its own position in the scheme of treatment. The introduction of the grinding pan into the milling practice of Western Australia was undoubtedly due, in the first place, to a desire to lower the treatment costs in wet-crushing mills, which were using the bromo-cyanide process on sulpho-telluride ores. Its success in this particular function has led to its development as a grinding machine pure and simple on many different classes of ores. The conditions which introduced the pan were:

1. A high consumption of a high-priced chemical, namely, bromo-cyanide.

2. The knowledge that in this respect expenditure could be most easily reduced by eliminating by concentration as much of the gold as possible before treatment with bromo-cyanide.

3. An ore, the valuable portion of which was most susceptible to sliming.

4. The necessity of reducing the crushed ore from the stamps, to a size which would free the mineral particles for concentration, with the creation of a minimum percentage of slime.

These conditions seem somewhat contradictory, inasmuch as good concentration practice demands that the mineral be saved in as coarse a condition as possible, while on the other hand a high extraction from the roasted concentrates demands that all mineral particles be crushed sufficiently fine to free them from enclosure in gangue material. It is, however, this very conflict of demands which has made a place for the grinding pan, the construction and operation of which renders it possible to regulate with some accuracy the size of the product which it discharges.

While stage grinding, as a preliminary to concentration, is the work for which the pan was originally introduced, its field of usefulness is by no means confined to this particular duty. Its efficiency as a fine grinder is being steadily demonstrated, and the limit to which fine grinding in pans can be carried is not yet in sight

In regard to sliming of the total stamp-mill product, it seems by no means settled whether stage grinding in pans with final sliming in tube-mills, will not prove more economical than direct sliming of the whole product in tube-mills. There is no doubt, however, that as an adjunct to a stamp-mill for the purpose of giving increased efficiency to the stamps by allowing the use of coarser screens, the field for both these machines, used in combination with each other, is a wide one.

It is the successful development of the grinding pan which has enabled the wet-crushing mills at Kalgoorlie to survive the increasing competition of the dry-crushing and roasting mills; and it has been the chief factor in increasing output and decreasing costs of the stamp-mills at the quartz mines outside of Kalgoorlie.

D. P. MITCHELL.

Kalgoorlie, June 9, 1906.

ZINC DUST V. SHAVING

(August 18, 1906)

The Editor:

Sir: During the last few years I have had occasion to visit many cyanide plants in different parts of the country, and have noted with great interest the methods employed in different sections. While in the Black Hills a few years ago I observed with great satisfaction the apparent success of zinc-dust precipitation and noticed with what economy in labor great volumes of solution were handled. The question arises in my mind, Why is zinc-dust precipitation not more universally employed? All cyanide operators are well aware of the great amount of labor necessary to keep zinc-boxes properly dressed and in good working order, to say nothing of the labor involved in the monthly or bi-monthly clean-up. In point of economy in labor, zinc-dust precipitation certainly has the advantage over precipitation on zinc-shavings. If it is all that could be desired in other respects, why would it not be economy to design all plants of moderate size with this process?

I think a discussion of the subject by those employing the process, giving figures as to cost of installation and operation, efficiency, etc., would be read with great interest by cyanide operators.

CYANIDE MAN.

Goldfield, Nev., July 28, 1906.

Zinc-dust precipitation in cyanide work possesses distinct advantages over zinc-shaving, and has been employed with marked success at the cyanide plants of the Homestake Company in the Black Hills. This process was evolved there after diligent study, but all the details have never been given to the public, and it is possible that the success of the process at the Lead City plants may depend upon some important trick of manipulation which the management wishes to conceal. Many attempts have been made to use the process at other plants, but without success. It would be difficult to assign the cause for this failure. It may be due to the reluctance of operators to supplant a simple and reliable process for one comparatively complicated and untried; but it would seem to be due in some instances to unfavorable conditions for precipitation, or it may be altogether due to having missed the one feature essential to success, as known only to the Homestake operators. The reason for the continued popularity of zinc-shaving is obvious. The process is simple and appeals to the average intelligence. We join with our correspondent in his wonderment at the slow introduction of zinc-dust, and venture to predict that as soon as the process gets a fair start and the principles of its use are better understood, it will easily supersede the cumbersome method in vogue at the present time. We shall be glad to publish the views of cyaniders on this subject.—EDITOR.

THE TREATMENT OF DESERT ORES

(August 25, 1906)

The Editor:

Sir—I have been interested in the discussion by Mr. Bosqui and Mr. Hunt relative to the treatment of desert ores. As I have recently been required to devise a method of treatment for the ore of one of the mines near the Combination, at Goldfield, the following may be of interest.

The material is typical of the Goldfield district, but was the lower grade 'milling ore,' averaging about $20 per ton. When the samples were brought to the laboratory I was requested to make 'amalgamation, concentration, and cyanide test' to determine the best possible extraction obtainable. The test gave the following results: Recovered by amalgamation, 32%; by concentration, 16.5%; and by cyanidation of tailing, 46.5%, making a total recovery of 95 per cent.

The concentrate was very high-grade—being over $500 per ton—and represented ½% by weight of the ore taken. After completing this test, I decided to try direct cyaniding of the ore after dry-crushing the whole of it to 100-mesh, with the result that I obtained 94% extraction. After confirming these results by further tests, I recommended the following general method of treatment: Dry-crushing to 12 or 16-mesh, fine grinding in tube-mills with cyanide solution to 100-mesh, agitation, and filter-pressing.

The direct treatment of this ore by cyanide would have many advantages over the usual custom of amalgamation and concentration followed by cyaniding. In the first place water is very scarce; in the ordinary stamp-mill practice water is used in the ratio of from four to seven parts by weight to one of ore. In following the idea suggested above, it would be necessary only to use a sufficient amount of water as cyanide solution to get the best results from the tube-mill, and agitation tanks, and for washing in the filter-presses. According to Mr. Butters, El Oro practice shows that the proportion of water to ore used in tube-mills should be one to one, or less, to obtain the greatest efficiency. The plant would cost much less to install than the usual stamps and concentrators with sand and slime cyanide-annex; and, requiring but one crew of men, it would reduce the operating costs materially. I doubt the advisability of treating the majority of ores in both sand and slime plants; such plants in my opinion being suitable only to tailing piles where the crushing has already been done. In most ores if a plant is necessary to treat the slime when crushing coarse, it will pay to slime everything and treat altogether—thus combining all the work in one plant with one crew. Nor do I believe in the other practice—except in rare cases—of crushing all the ore to a 200-mesh slime. Those who have had to operate filter-presses or filtering devices realize the difficulty of filtering, washing, and removing cakes of 200-mesh slime; while if all the ore is crushed to 100 mesh, agitated as one product, and filter-pressed, the extraction is not materially reduced, save in exceptional cases and the cakes can be made thicker and dryer, and are easily washed and removed. In fact, some of the new presses are so constructed that material of this kind can be filtered, washed, and sluiced out automatically without entailing the expense of opening the presses between each change. This is not possible with a very fine slime, which becomes as tenacious as India rubber and must be almost chiseled from the cloths. In the majority of cases the mill clean-up will show a greater extraction than if all is crushed to pass 200-mesh; the possible exceptions are sulphides or perhaps a clean silicious ore, free from argillaceous material.

Mr. Bosqui, in advocating wet crushing and amalgamation, mentions the "great advantage of being able to secure daily from plates 50% of the extraction obtained from $50 ore, as compared with waiting for the tedious monthly or bi-monthly clean-up in a cyanide plant." I confess that I cannot see this advantage. Crushing 60 to 75 tons per day and recovering $25 per ton would be but $1,875 per day at most, and surely he would not make a melt every day for that amount. A bi-monthly clean-up would amount to but $28,000, which is not an unusual clean-up even in our California mills. A cyanide plant, if properly designed, can be roughly cleaned up every ten days without any great inconvenience; to be followed by a complete clean-up, say once a month. I must agree with Mr. Hunt regarding the practicability of crushing dry to any required fineness, as instanced in cement mills, but that is not necessary. It is my belief that the time will soon come when nearly "every new modern mill operating upon a logical basis" on the deserts will be so constructed that crushing to 10 or 16 mesh will be done dry by breakers and rolls or ball-mills, and the whole product will then be mixed with cyanide solution of a strength sufficient to obtain the highest extraction and passed through tube-mills or some other device to reduce to a fineness of 100 mesh; that the product will all be agitated together and passed to filter-presses so constructed as to permit of automatic discharging.

The objection to dust, I think, will be overcome as it is in the cement mills of Europe, where the laws compel the installation of exhaust-fans and dust-collectors. In a plant such as I have roughly outlined, the dust could be conveyed by pipes connected with an exhaust-fan to a hopper, where a spray of cyanide solution would collect and carry it to be mixed with the tube-mill product. Such an arrangement would be efficient, and the cost of installing and operating would be trifling.

Another point which I think is generally overlooked is that the coarse gold in any ore when subjected to the grinding action of the tube-mill or any other machine necessary to pass it through a 100-mesh screen, will cease to be coarse gold. All of us know how brittle gold becomes even upon rolling, as is seen in assaying bullion when rolling out the cornets, which, unless repeatedly annealed, will break to pieces. This same action occurs in grinding the ore and is aided by the sand grit, so that by the time the ore passes 100 mesh, practically every particle of gold is a very fine scale, a most ideal condition for cyanide attack.

Crushing to 10 or 16 mesh, no matter how hard the rock, is done cheaper in breaker and rolls dry, than with breakers and stamp-mills wet. The cost of installation per given capacity is less, and in desert regions, where water is of itself such a problem, the dry-crushing method must appeal to both the manager and the metallurgist.

I think it is becoming generally recognized that fine grinding of the whole product to 100 mesh increases the profits, and when this is so I do not think amalgamation should play any part in the process, except possibly in very rich (over $100) coarse gold ores. The stamp-mill as an amalgamator and when crushing to 30 or 40 mesh has long held its place in the first rank, and for that purpose can hardly be replaced But its success in this line seems "by the compelling force of custom" to lead metallurgists to its adoption in all cases. But new conditions have been presented. We no longer want a 30 or 40-mesh product, but a 100-mesh. Therefore let us adopt the machines best suited to these new conditions.

LOCHIEL M. KING.

Oakland, August 1.

CYANIDE NOTES

By E. A. H. Tays

(September 1 1906)

Successful cyaniding depends on two things: A working knowledge of the process, and common sense. A certain treatment may be successful at one camp and fail at another; in fact, the treatment successful with ore from one level of a mine, may fail with the ore from another level of the same mine. Consequently, to be successful, the cyanider must be ever alert.

Another thing he must bear in mind, and that is never to despise future possibilities. I call to mind a personal experience. We were treating with fair success, a refractory lot of tailing of high value, and because the mill-tailing was a little poorer than that from the cyanide plant, we ran all of it down the arroyo, as being worthless to us. Five years later, at the same plant, I treated at a profit, tailing carrying but one-half the values of that which ran into waste five years before.

We all recognize that oxidation of the charge during treatment is desirable in securing high extraction; but, so far, no economical method of obtaining this condition has been devised. I refer to the leaching of ordinary sand charges, for the agitated charges of slime are readily oxidized. It is generally conceded that in a vat charged with sand, when drained, the charge is affected by the oxygen of the atmosphere to a depth of but a foot or two below the surface. From experiments and accidental discoveries, I am led to believe that such charges are affected to, or nearly to, the bottom; time being the only essential.

I call to mind a small plant treating from 800 to 1,000 tons per month, it being an old plant remodeled. The material treated was a tailing from a stamp-mill, consisting of at least 25% clay slime; and, at best, only 80% could be extracted from it. It should have been arranged to separate the slime from the sand, in order to treat each separately, and this could have been done economically; but, as the plant was working to good advantage, it was deemed best to make no change. The treatment was designed to discharge one vat (about 36 tons) per day, and as there were ten vats, each vat was given a ten days' treatment, although the engineer told me, in turning over the plant, that full extraction was made in seven days. After a month or two some experiments were made, and it was found that as good results as could be obtained from the material treated could be obtained in five days; and, of course, after verifying these results, it was decided to double the output, as this increased the running expenses but little.

While we were waiting for extra supplies (two months) the five-day system was installed, and practice upheld the experiments, as long as we had an excess of vat capacity. When everything was ready to double the output, and every vat was put regularly into commission, our extraction dropped off from 6 to 8%, and continued low all one month, before the trouble was detected.

The treatment was as follows:

1.	Charge vat with sand	8 hours
2.	Charge strong solution (0.2%) from below, valve half open and let soak	14 "
3.	Discharge, valve wide open, until solution titrated 0.1%, then charge weak solution (0.1%) on top and let run	12 "
4.	Charge allowed to drain	12 "
5.	Strong solution charged from above, until vat filled, when valve was opened, and when the escaping solution titrated, 0.1%, weak solution was run on six hours	12 "
6.	Charge allowed to drain	18 "
7.	Strong solution charged (from above) until full, and left to saturate	6 "
8.	Solution drained off and when titrated 0.1%, weak solution was fed on top, feeding and draining at same time	12 "
9.	Let drain two hours, when seven tons fresh water were run on from top and charge let drain	18 "
10.	Sample and discharge tailing	8 "
	Total	120 hours

This treatment gave us good results in five days, during full charge experiments, when we had extra vats; but when all the vats were put into commission as already stated, the extraction dropped as much as eight per cent.

In going over the treatment step by step, everything checked up to the ninth step. It was found that while our scheme called for the treatment indicated, when we had vats to spare, the boys let the charge drain (oxidize, in fact), up to within the 14 hours necessary to fill the water-wash and drain dry enough to discharge, the vats remaining dry about four days before the water-wash

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was run on. This long period of rest allowed the latent reaction of oxidation to develop fully, and when the water was run on it readily took into solution the dissolved metals still in the tailing.

After this discovery, the treatment was modified, as follows:

1.	Charge vats with material	10 hours
2.	Charge strong solution (0.2%) from below at one-half valve capacity, and let soak	14 "
3.	Discharge solution, valve wide open, and at end of second hour, feed (on top) weak solution (0.1%); valve half open	12 "
4.	Let charge drain, valve wide open	12 "
5.	Charge strong solution from top, and let soak	6 "
6.	Discharge, valve wide open, and at end of second hour, charge (on top) weak solution; valve half open	12 "
7.	Let charge drain	18 "
8.	Charge weak solution from top, valve wide open, and let soak	6 "
9.	Let charge drain and oxidize	62 "
10.	Charge fresh water from top, until vat fills, and let soak	3 "
11.	Open valve wide, letting charge drain; this wash going to storage-vat leading to zinc-boxes	13 "
12.	Sample and discharge tailing	12 "
	Total	180 hours

This treatment gave good normal results; and, as a charge was run through in 7½ days, it still gave us a total of 1,500 tons per month, instead of 1,000 tons under former treatment.

I firmly believe now that oxidation takes place clear to the bottom of a vat full of sand, the vat having been filled with water and this then allowed to drain off, from the bottom. When full of water, the interstices are occupied by the water, the sand settling and packing as close as possible. As the water drains off, these interstices empty and form a vacuum, which naturally fills with air; the pressure of this being ample to fill the lower voids, as the water recedes. Once the interstices are filled with air, the chemical reactions that take place cause a practical oxidation to set in, if time be allowed; and it is probable that 80% of this reaction takes place within 60 hours. That the interstices in the charge do fill with air is proved by the fact that in running water on to a charge, much ebullition takes place, and for half an hour or more after the charge is covered with water; many air-holes forming all over the top of the charge.

Another proof that chemical reaction takes place in a charge treated as outlined, and then left to drain three or four days is (from my own tests) that if a water-wash be run through immediately, it will show but a few cents per ton; whereas, if the water-wash be run through several days after the charge has been allowed to drain, it will carry from $1.50 to $2 per ton.

Several years ago I discovered that newly turned zinc, if the shavings where stored were exposed to air, was necessary for perfect precipitation. Later, I have learned that solutions running through the zinc-boxes should not be too strong in cyanide, if a product high in the precious metals be desired. When the solutions run 0.1% or over, the reaction set up is so violent, that a larger proportion of zinc is destroyed (oxidized) than is necessary, the resulting slime (auro-cyanide) being low in gold and silver and high in zinc.

With weak solutions the reaction is normal, and metallic zinc is destroyed in proportion to the values in the solution; and the resultant slime is high-grade. I have secured slime running $30,000 per ton and high in zinc, requiring acid treatment; whereas, the next month, on the same grade of ore, slime, low in zinc, requiring no acid treatment, and carrying $57,000 gold per ton, was obtained, simply by not allowing the solutions to get beyond a certain strength, say 0.05%. For this reason, my practice has been, of late years, to run all solutions and water-washes into a common vat, the mixed solutions from which are run to the zinc-boxes. When the solution is above 0.05% in cyanide it can be run through the zinc-boxes a little slower; and when below 0.05%, a little faster, to obtain the normal results. Should the solution be very low in cyanide, say 0.01%, and yet rich in metals, a little cyanide can be added to the head box in each row, if it is found that solutions of 0.01% strength do not precipitate normally.

In the clean-up it is good practice to return all zinc, staying on a 40-mesh screen, to the precipitating-boxes. This zinc should be kept covered with a weak KCy solution until returned to the boxes. This prevents the violent oxidation that sets in under exposure to the atmosphere.

COPPER AND CYANIDE SOLUTIONS

(September 1, 1906)

The Editor:

Sir—I am on my way back to the mines again. Just had a letter from my cyanide man, which says we are getting a fearful lot of copper in the zinc-boxes. Will you send me, to my address at the mine, anything that has been printed of late on this subject, for all I know seems very meagre on the matter.

R. B. S.

New Orleans, August 14, 1906.

The presence of copper in ores causes decomposition or loss of cyanide when they are treated by the cyanide process, especially when it occurs in the form of carbonate, each pound of copper dissolved combining with about four pounds of potassium cyanide. A further trouble is met in the precipitation by means of zinc-shaving, owing to the tendency of the copper to form a firm metallic coating on the surface of the shaving when the cyanide solutions used are comparatively weak. With stronger solutions less copper is precipitated, and then in a more loose and spongy form, which does not prevent the deposition of gold and silver. The experiments of Von Oettingen (*Journal* Chem. & Met. Soc. South Africa, Feb., 1899, and *Proceedings*, Vol. II, pp. 557–570), and of Christy (*Transactions* A. I. M. E., Sept. 1899), show that the difference of potential between zinc and copper is greatest in very weak solutions of KCN, becoming very small in strong solutions. This is brought out in a graphic manner in Prof. Christy's Fig. 19. Experiments and working results quoted by Sharwood (*Proceedings* 13th Convention California Miners' Association, pp. 209, 210, MINING AND SCIENTIFIC PRESS, April 29, May 6 and 13, 1905) show that much more of the copper present was thrown down from weak than from stronger solutions. See also Browne (MINING AND SCIENTIFIC PRESS, Jan. 24, 1903.)

Whether it will pay best to use stronger solutions, or to add cyanide before precipitating, or to substitute some other method of treatment or of precipitation, will depend on a number of circumstances. This is discussed by Julian and Smart ('Cyaniding Gold and Silver Ores,' p. 209). The dipping of zinc-shaving in a solution of lead, patented by MacArthur, was intended to overcome the difficulty caused by copper in the zinc-box. Another patented process (Porter's) is said to be used at the Bagdad-Chase mine, the gold being precipitated by zinc shaving and the copper afterward removed by prolonged treatment with zinc-dust and ammonia. This was described in the MINING AND SCIENTIFIC PRESS of May 20, 1905.—EDITOR.

CYANIDE PRACTICE WITH THE MOORE FILTER—I

By R. Gilman Brown

(September 1, 1906)

The plant, of which the following is a description, was designed, erected, and brought to satisfactory operation during the years 1904–5. Much of the burden of the preliminary investigation and construction fell naturally upon the shoulders of Mr. Theodore J. Hoover, superintendent of the company, for which the writer was general manager. During the progress of the work a large mass of data, experimental and practical, was collected, which the two of us had planned to collate and publish in collaboration. This was all lost in the San Francisco fire. This paper and drawings have been prepared, largely from memory, but with the valuable assistance of the private notes of Mr. E. H. Nutter, who from his position as mechanical engineer and later superintendent for the Standard Co., had been fully in touch with the plant from the first.

Ever since the installation, in the early '90s, of the first cyanide plant at the Standard mine, at Bodie, California, under the initiative of Mr. Thos. H. Leggett, the treatment of the slime has been a problem of growing importance. The earliest practical attempt to solve it was in the nature of an evasion; namely, to break the caked slime from the ponds with a disc-harrow and mix it with sand. Apart from the cost, which came to an additional 30c. or more per ton on the whole tonnage, including the sand, the coarse material was only sufficient for the dilution of a portion of the slime product. Besides, this method took no advantage of the fact that by agitation the slime would yield a higher extraction in a shorter period. For want of a better way, the practice was continued for a time, as the only way of preventing the congestion of the tailing ponds with untreatable slime. But active experiments were made in other directions.

Agitation and decantation were discarded because of the flocculent character of much of the slime, that would not settle in 72 hr. to over 15% solids. Filter-pressing was tried and abandoned, because an eighth of an inch of pure slime would make the cloths impervious, even under 120-lb. pressure; and even if the slime was mixed with fine sand, the filtering was so slow that the sand settled out in the chambers, with the same result.

Dehydration of dry slime by roasting made both filtering and decantation possible, but with lower extraction and high cost; moreover, it was not applicable to the direct treatment of the wet product from a stamp-mill. More than one method of filtration in vats during agitation was tried and discarded. In the summer of 1903 the first crude tests with the Moore method were made, with no brilliant results. Later a personal investigation of the plant at Mercur made clear the inherent advantages of the method, while at the same time it exposed certain mechanical deficiencies. An experimental plant of half a ton capacity was installed at Bodie and from the first the results were good. Such was our success that by the middle of the winter we decided to apply it to all the tailing from the Standard mill. In designing the plant every care was taken to avoid the defects of the Mercur plant, but notwithstanding this, several months were consumed, after the plant was nominally completed, in alteration and modification, the major part, let it be noted, being in details outside of the Moore process proper. By the middle of 1905 operations had become fairly regular and the profit for the whole year amounted to more than half the cost of the plant.

The ore consists of quartz, iron oxides, and clay, the last coming from the decomposition of feldspar in the country rock. At times this equals 50% of the ore and as an average can be taken at 33⅓%. The gold is partly coarse and partly very fine, the latter portion amalgamating badly, to such an extent that at any time fine colors can be panned from below the vanners. The proportion of silver is high, the mill bullion being worth only $10 to $12 per ounce. Altogether it is a difficult ore to treat, despite the total absence of all minerals ordinarily classed as deleterious.

In conjunction with the introduction of the Moore method weak cyanide solution was substituted for water in the 20-stamp mill and the grade of the plates was increased from 1½ to 2½ in. per foot, in order to cut down the proportion of liquid to keep the plates clear. This steep grade, in conjunction with the hardening effect of the cyanide solution on amalgam, has diminished the extraction by free milling from 60 to 50%. After considerable experiment the strength of cyanide solution was settled at two pounds per ton, and with this at first no trouble was experienced with the plates, but of late the lower ones, where the amalgam lies thinnest, have shown signs of wasting, and some have been renewed. Possibly this can be lessened by using Muntz metal or by carrying a heavier coating of amalgam. Lime, at the rate of 10 lb. per ton, is added to the ore before it goes to the crusher. After passing over vanners, the pulp is raised 63 ft. to the high-level flume for conveyance to the slime-plant, 1,800 ft. away. The elevation is done by four Frenier sand-pumps, in series. The three lower ones, 10 by 54 in., have a lift of 16 ft. 4 in. each; the upper one is 8 by 48 in., and has a lift of 14 ft. The high level flume is four inches wide and nine inches deep, set on a grade of 7/16 in. per ft.; the pulp carries 17 to 19% solids and at this grade it flows freely under all conditions of temperature. A grade of 5/16 in. was found insufficient, particularly in cold weather; a broader flume was also found to give trouble by accumulations of sand. When received at the slime-plant, the pulp is again raised by a centrifugal-pump to the cone-separators. A word regarding the two methods of handling pulp: The Frenier pump, for a regular flow and for lifts within its capacity, is most satisfactory; the consumption of power is nominal and the wear is confined to the stuffing-box at the discharge; however, it requires more attention in operation, particularly in starting or stopping, and a great deal of pains in erecting.

The re-grinding is done in a 5 by 22 ft. trunnion tube-mill of Allis-Chalmers make. The feed is from the under-flow of two cone-separators. These are of wood, approximately five feet deep and with 60° slope of side. No extra solution is used to affect the separation, the degree of which, that is to say the proportion of overflow to underflow, being regulated by a simple needle-valve actuated by a screw and hand-wheel from the top. The routine test for separation is to catch the overflow on a 150-mesh screen and raise the needle-valve until no material is held on the screen. The underflow, containing the coarse stuff and a certain proportion of adhering slime passes to the tube-mill, being mixed with sand and slime from the ponds, automatically fed into the stream. The outflow from the mill is returned to the cones. The mill makes 26 rev. per min. The pebbles are Greenland concretionary flints, the charge being about 12 tons which fills the mill a short distance above the middle. This charge seems to be the best for grinding and most economical of power, but the difference is not great and the charge can wear to half this quantity with small diminution of grinding. The attempt was early made to use selected pebbles from local glacial drift, but their irregularity prevented free motion, and gave a high wear of liners, with increased power consumption. The insufficient hardness also of the pebbles made them expensive.

The linings originally furnished were of white cast iron, secured in place only by the arch of the shell. They were a constant source of trouble from dropping, and their life was short. Soft-wood blocks on end, six inches long, were next tried. There was no diminution of the grinding, but besides their short life—of about 10 days—they introduced an unexpected element in excessive foaming of all the solutions. It was assumed that this was due to a saponification of the wood oils by the alkali in the solution But whatever the cause, it produced an overwhelming mass of suds, overflowing all the launders and covering the vats with 18 in. to 2 ft. of foam. Mountain mahogany was tried next, diminishing this trouble, but with scarcely a longer life. Silex was subsequently used, and for a time answered well, but when partly worn there was continual delay from replacing worn-out blocks. This objection was entirely aside from the delay of four weeks or more demanded for the full setting of the cement that held the lining in place. The continuation of the use of silex meant the addition of a second tube-mill, that is, the duplication of the re-grinding plant.

Finally, the practice settled down to wrought-iron plates for liners. These are ¾ by 8 in., cut into 7 and 15-ft. lengths, and bolted through the shell. Some of the plates of this lining would be worn through in 90 days, but by replacing these the average life came to over 100 days and the duty to 4,800 tons of sand, ground to 200-mesh. The consumption of pebbles was 15½ tons; reducing this to pounds per ton, the lining wear was 2.44 lb. and the pebble consumption 6.47 lb. It seems likely that the wear of lining and pebbles is rather a function of time than of tonnage and in any case it is certain that the rate of wear of lining at least is much affected by the manipulation and proportion of solid in the feed; so that this somewhat anomalous success with the softer lining is probably in a measure due to the increased skill in adjusting the feed to the mill. It is to be noted as a point in favor of the wrought iron that there is little waste, as it can be worn down thin without breaking. Aside from this, the ease of inserting and securing the straps—all of which can be accomplished in 10 hr.—was the determining argument for their final adoption. The power for the mill is approximated as 50 h.p. when running and 100 h.p. at starting. The maximum grinding capacity of the single unit has not been definitely reached, but it is safe to place it at 60 tons of sand per 24 hr. This makes the duty of a horsepower month 36 tons. At normal cost of electric power in California the power cost comes to 17c. per ton; at the Standard mine, which produces its own power, the cost is 5½c. An interesting comparison can here be made between the cost in power of stamping from 2½-in. size to 30-mesh and grinding from 30-mesh to 200. The consumption of power in the Standard mill of 1,000-lb. stamps is, for stamping alone, 28 h.p. The monthly tonnage is 1,800, so that the duty of one horsepower-month is 64 tons. The reduction of linear size from 2½ in. to 30-mesh is given in the ratio 2.5:0.025 or 100.1. For the tube-mill the ratio between feed and product is 0.025:0.001 or 25:1. Summarizing: One horsepower in the stamp-mill reduces 64 tons per month at a ratio of 100.1, and in the tube-mill 36 tons, at 25:1 These figures are, for several reasons, far from accurate, but they serve to show in a sketchy fashion the largely increased power cost of fine-grinding. In connection with the recent notes* of Mr. Butters on the necessity of heavy foundations for the tube-mill, it is interesting to note that the foundations at the Standard are piers of heavy timbers, tied and bolted together with heavy angle pieces of ¾ by 8 in. flat iron and carefully bedded on mudsills set in hardpan. Mr. Nutter tells me that after nearly a year's use, these were tested with a transit and found unmoved. As a final comment upon the tube-mill, it can be said that it has proved itself a highly efficient fine-grinding machine, but that when the whole plant depends upon a single unit, as in this case, any accident to it shuts down the whole plant. This virtually demands the duplicating of the entire tube-mill equipment—for a small plant, a heavy extra capital expense. This led to experiment with a modified pan for fine-grinding. It was found that a single five-foot pan had a capacity of about five tons per 24 hr. and consumed

*MINING AND SCIENTIFIC PRESS, May 26, 1906.

10 h.p. or from eight to nine pans were required to handle the coarse product of 20 stamps, at a marked increase of power-cost. No determination was made of wear of metal, but judging from experience with grinding concentrate it would be high, and probably the labor of attending to a battery of pans would be large. Altogether for a small plant, so far as this experience goes, three smaller tube-mills, any two of which could do the whole work, would be preferable to pans, and probably cheaper in installation than two large units.

The per cent of solids has been increased in the tube-mill discharge to 25%, or perhaps higher, by the addition of material from the ponds. The settling vats, of which there are nine, aggregate a capacity of 21,000 cu. ft.; they are flat-bottomed wooden vats in two sizes of 70 and 80 tons solution capacity, representing 2,200 and 2,550 cu. ft. respectively. On the basis of inflow of 400 cu. ft. per hour, the one size fills in 5.5 hr. and the other in 6.4 hr. While filling they are agitated, then allowed to settle 28 hr. and slowly decanted down to the upper surface of the slime. This occupies eight hours and two more hours are taken to mix and pump out the sludge, a total of about 43 hr. Forty-two per cent of the solution is recovered by decantation. It passes to an additional clarifying tank and then to the gold tanks. The remaining pulp carries from 34 to 40% solids. Returning to the mechanical details of the settling room: The agitators are drags, actuated by a vertical shaft, driven by overhead crown gearing, but stepped on a steel button, running in quicksilver, on the floor of the tank. Above any level at which the heavier pulp can collect, an 8 by 8 in. cross-arm is secured to the shaft and from this the drags hang by links or short chains. The drags are short lengths of 30-lb. T rail; the speed of seven or eight r.p.m. is ample and the power consumption is low. This drag type of agitator is essential, as otherwise the slime packing around the blades would stall the driving gear. After decantation, the pulp is made homogeneous by stirring with the drags and then transferred to the storage or treatment vats by centrifugal pump or air-lift, both devices being used. For decantation there are hinged pipes that are lowered into the vat, after the requisite clearness is attained and that can be kept near the surface so as to draw off the least turbid solution. These discharge through the sides of the vat a short distance above the bottom. The sludge is drawn off through pipes in the bottom.

CYANIDE PRACTICE WITH THE MOORE FILTER — II

By R. Gilman Brown

(September 8, 1906)

So little has been published concerning the Moore process that no excuse is needed for explaining it, with as much minuteness as is possible in the absence of notes and working drawings.

The points in common between the Moore and the Cassel-Butters process are: The filtration by vacuum, with the resulting adhesion of the cake to the outside of the filter; and the general type of filter, which allows a great area in compact and cheap form. It seems to me that the point of adhesion of slime to the filter, making possible its removal from the remaining pulp, is the real point of difference between these and other slime methods. Others in plenty have filtered by suction, others have used parallel closely spaced units to secure maximum area in minimum volume, but so far as I know the adhesion of a cake to the outside of a filter and its removal by this means from the unfiltered pulp is the essential novelty with which the Moore process should justly be credited, in practice, at least. The essential difference between the Moore and the Cassel-Butters method may be considered an inversion of the Mountain-Mahomet theorem: Moore transfers the filters, with their load of slime, by means of a traveling crane from the pulp to the wash-water tank and then to the discharge hopper, while the Cassel-Butters plan removes the pulp from the filters by pumps of large volume, substitutes wash-water and then discharges, the operation being conducted in the same vat.

In the Standard Co.'s plant the filters are of canvas of medium weight, 5 ft. wide and 16 ft. long; the canvas is double, sewed round three edges, and the fourth (long) edge is bolted between slips of Oregon fir (*QQ*, Fig. 3) 1½ by 6 in. in cross-section. In the bottom edge of each filter or 'plate' is a ¾-in. channel-iron (*RR*) that serves as a launder for the in-filtered solution. The filters are stitched through both sides vertically at four-inch distances and in the compartments thus formed ¼ by 1 in. strips are inserted to allow circulation. Within each filter a 1-in. vertical suction-pipe, flattened at the end so as to dip into the channel-iron launder,

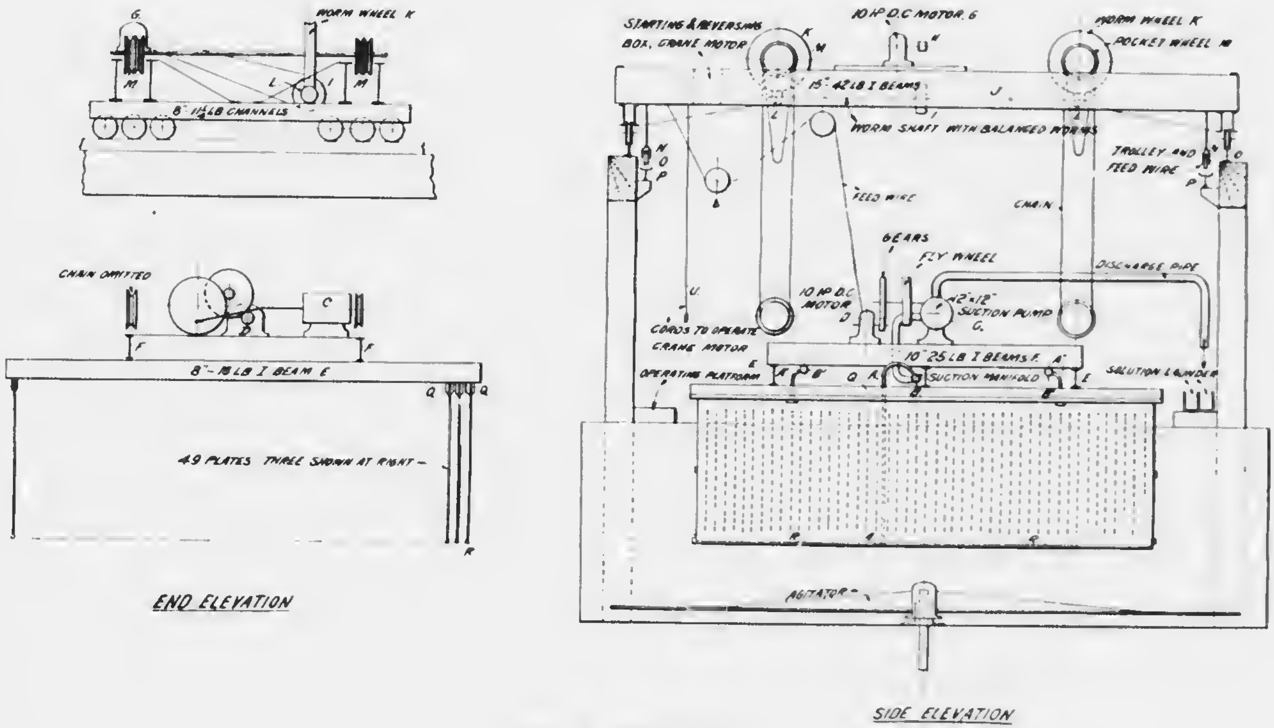

Fig. 3. Moore Filter

reaches to the bottom from the outside. The outer end of this is connected by a short length of suction-hose to a three-inch manifold, through which the suction is applied. The details of this will be better understood by reference to Fig. 3: *A* is one of the vertical suction-pipes, *B* the manifold in end-view and A^1 and B^1 the plenum-pipes and manifolds. In the Standard Co.'s plant there are 49 of these 'plates' hung from a frame of steel I-beams (*EEE, FFF*), the distance between centres being four inches. This constitutes the 'basket,' approximately 16 ft. square. On the top of this rests a suction-pump (*C*) 12 by 12 in. driven by a 10-h.p. D. C. motor (*D*), which derives its current from the crane overhead. With a vacuum of 20 in. the motor consumes 3 h.p. of current, but in the early part of the cycle when filtering is fast, there is a heavy rise in power at the end of the stroke due to expelling solution from the slender clearance space. To equalize this the pumps have heavy fly-wheels. Clearly this basket arrangement lends itself to great area of filtering surface, there being 7,840 sq. ft. concentrated in each unit. The basket is hung by four endless one-inch pitch-chains to the overhead crane and is raised or lowered by a 10-h.p. motor (*G*) through the medium of balanced worm gearing and differential chain-wheels. The driving pulley on the motor is indicated at *H* and the driven pulley at *I*, on the worm shaft *J*; *KK* are the worm-wheels, keyed on the shafts of the pocket-wheels *MM*. These worm-gears are right and left thread. The worms bear on collars on the ends of the worm-shaft and the direction of rotation is such that, raising the load, puts shaft *J* in tension; at the same time the worms being right and left, there is practically no end-thrust on the bearings. The pocket-wheels are of six and seven pockets respectively. The crane-motor and pump-motor get current from the trolley-wires *NN* through the forks and wheels *OO*; P is the insulated rest onto which the trolley-wire drops, when not held up by the wheel *O*. The crane-motor is operated through the starting-box *T*, which is controlled from the tank-floor by the cords *U*. The total maximum load of the full basket on the crane is 35 tons and this is raised about seven feet in five minutes. This represents about 3 h.p. The motor consumes 8.4 h.p. of current, so that the loss, even with the balancing of the worms, is heavy. Still, as the hoist is in operation not above one hour per day, this is negligible. Notwithstanding the differential gearing was furnished by

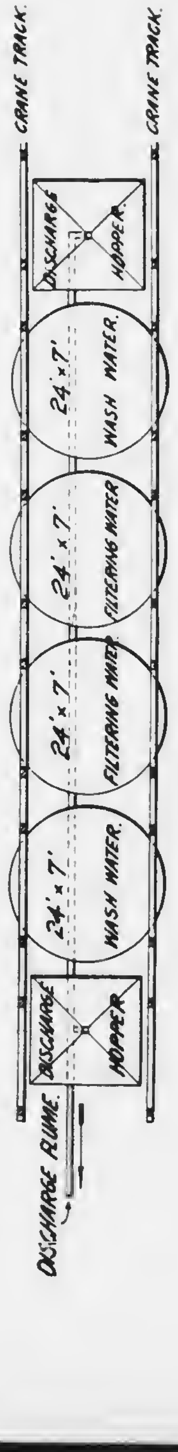

Fig. 4. Arrangement of Vats.

specialists, it gave trouble at first, largely because the pockets in the wheels were not deep enough or well fitted to the chain. Even now when this has been overcome, guide-wheels that were devised to make the chain grip the pockets better are retained as a measure of precaution. Probably any future design should have wheels with twice the number of pockets. The general construction of the crane is clearly indicated by Fig. 3 and the photographs, and needs no explanation. The pocket-wheels are placed at quarter-span points of the large I-beams, and all stress calculations were for a factor of safety of five. The supporting track is of 60-lb. T-rail and rests on heavy 12 by 18 in. longitudinal sills, the caps of the supporting sets. To prevent spreading of track, the sets are framed back to the main posts of the building. All of this construction could well be of structural steel where transportation cost is not prohibitive. The traversing device for the crane is a $\frac{5}{8}$-in. plough-steel wire-cable, run over three and four-grooved sheaves, after the manner of the Koepe hoist, the main sheave being driven by a worm-gear from the main-line shaft. A grip on the crane engages the cable and the gear is started, stopped, and reversed by a belt-shifter through the medium of levers conveniently placed for operation by the basketman. The tail-sheave is on a carriage and adjustable to take up stretch of the cable. The rate of travel is five feet per minute. Fig. 4 shows the general plan of the filtering-vats and the discharge-hoppers. The filtering-vats (which, along with the others, belonged to the former sand-plant) are round, flat-bottomed vats 24 ft. diam. by 7 ft. deep. They have four-armed agitators (Fig. 3) running at 7 r.p.m. close to the bottom. The shaft to which these are attached is driven by crown-gearing below the tank. A special stuffing-box devised to keep the sand out of the bearing has proved satisfactory but could be improved upon. In addition to the agitators, each filtering vat is fitted with two six-inch air-lifts discharging onto two distributing launders on the tops of the baskets. These are run intermittently during the accretion period and serve to bring the coarser particles to the surface, whence, as they slowly settle, they are caught by the suction currents and deposited on the filter with the slime.

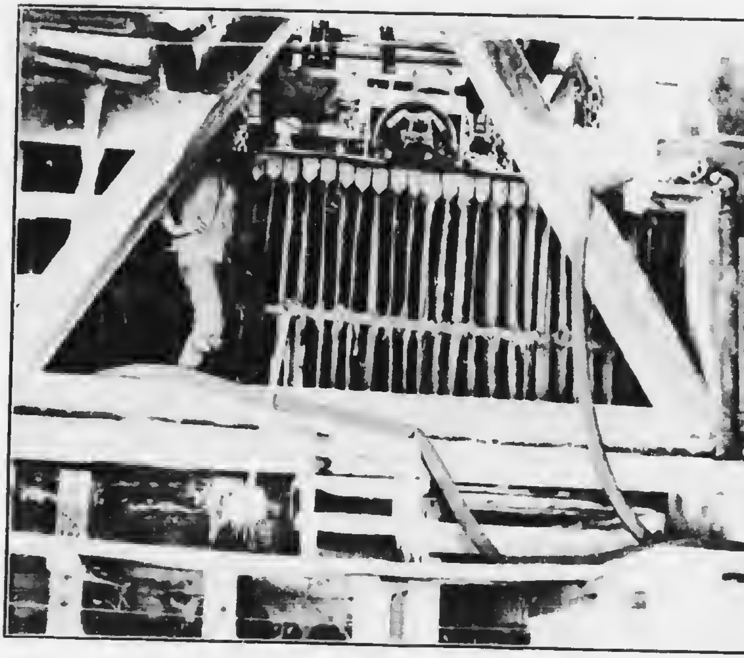

Fig. 5 Moore Filter, with Cake.

The action of the filtering process is as follows: The basket is lowered into the vat full of pulp, until the tops of the slats are submerged, and the suction pump is started. The first solution coming through muddy is turned back into the vat. It soon clears, however, unless a filter leaks. Should there be a leak, the identity of the faulty filter is quickly established by an inspection of the glass nipples connecting each individual 'plate' with the manifold, and that plate is cut out by closing the proper service cocks in the manifolds. Suction is continued with intermittent agitation until a sufficient coat is obtained, which in this case averages ¾ in.; then the basket is raised, the suction-pump still running, and traversed, with its load of slime adhering, over the wash-water tank and lowered therein. Here suction is continued with frequent titration of the filtrate toward the end, till the solution has fallen to the predetermined minimum net strength of cyanide. The displacement is good, about 0.7 ton wash-water being needed per

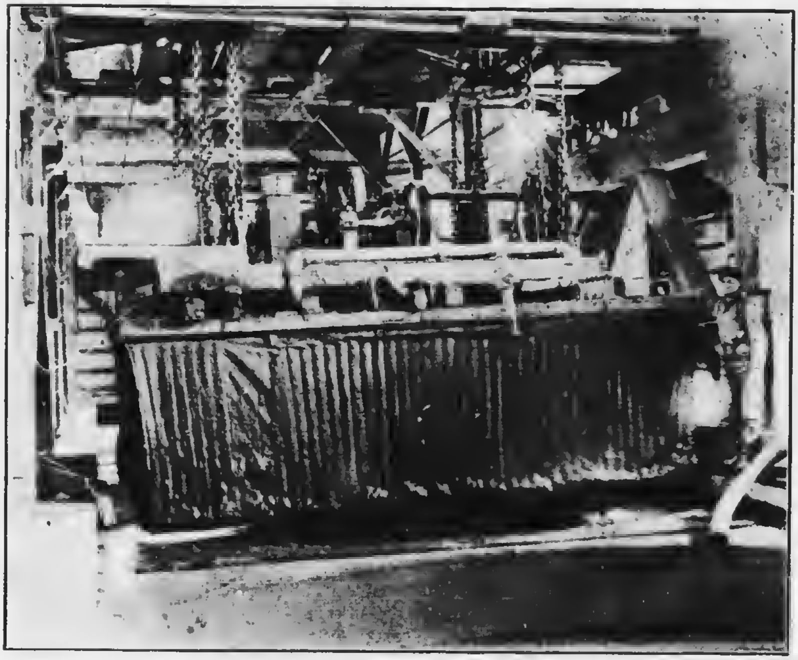

Fig. 6. End View of Moore Filter-Basket.

ton of dry slime; the cake in this condition carries about 40% moisture, so that every ton of slime is accompanied by 0.67 ton solution. The limiting net cyanide content of the filtrate is the difference between that in the wash-water, which always contains a little that has been soaked off from previous charges, and that of the filtrate. This is in the neighborhood of 0.15 lb. per ton or 0.0075 per cent.

After washing, the basket is raised again and run over the discharge-hopper; the suction is continued till excess moisture is removed and then an air-pressure of 35 lb. is turned on, in successive blasts of a few seconds each. This causes the cakes of slime to drop off, the discharge averaging about 85% of completeness. Every alternate day the filters are cleansed by substituting water, under 20-lb. pressure, for air. On each basket there are two special air-manifolds connecting with the inside of each filter, at points four feet from the ends. Probably these could be omitted, and the vacuum manifold be used for the plenum as well. The discharged sludge averages 68% *solids* and is sluiced from the hoppers into the waste-flume, about a ton of water being used for each ton of slime. The grade of flume for this class of material is 9/16 in per ft. The time consumed in a single cycle depends primarily upon the thickness of the pulp; as an instance, a pulp of 20% solids will scarcely build up a one-half inch coating in 10 hr., while pulp of 40% solids will give an inch coating in two to three hours. The average thickness for last year was 0.74 in. and the maximum for any one month 1.14 in. The time of the accretion period is about three and one-half hours; washing and discharging take the same, making about three full cycles in 24 hr. with time for emergency matters. This is considerably longer than is the experience elsewhere and must be attributed to the large proportion of clay in the material. During one month, when clay was notably less, the average fell to 2.35 hr. for accretion and 2.9 hr. for washing and discharge. The average load handled in each cycle is close to 18 tons of dry slime, so that the two units in this plant have an average capacity of 108 tons per diem. This has not been obtained in practice over any extended period, but the limiting factor has not been the filtering, so that the monthly capacity of the plant has been conservatively placed at 3,000 tons per month. Fifty-eight per cent of the total solution recovered from the plant has come through the filters. Beyond this point the process is the same as in any cyanide plant except that the sump solutions, after being standardized in storage-tanks, are returned by a three-plunger pump to the mill.

Taking up some of the details not yet touched upon the following is to be noted: The question of the wear of the filters is an important one. An extreme life of 10 months has been noted in some cases, but, if badly made or carelessly handled, they require constant attention, repairs, and renewals; half a cycle per day can easily be lost in this way. Six months can be taken as a fair average life; this, at a monthly tonnage of 1,500 per basket, makes the cost 5c. per ton. In other plants where smaller filters are used, the canvas is stretched on an internal frame of pipe and so kept taut, the pipe at the same time, serving for suction and discharge-pressure. In the Standard plant no trouble has been experienced from letting the filters take their natural hang, and when the plant was being designed, the stretching of the canvas over a frame did not appeal to us, both because of added expense of construction and of shorter life of the canvas, which would be under greater tension when the 'plates' distend under air-pressure and become distorted. Whether the latter is a valid objection, only unbiased comparative work can tell. The individual 'plates' are spaced one from the other by light wooden rods to which their edges are attached by twine. There are four of these, spacing the ends of the 'plates' at middle and lower corners. As a certain amount of slime gets through the filter at times (when a leak develops when the basket man is not at hand) the solution passes from the pump through a launder first to a clarifying-tank from which it is decanted to the gold-tanks. An addition, recently installed as an improvement on this, is a permanent set of 'plates' in the clarifying vat, through which the solution is drawn. This device is used elsewhere with success.

From tailing averaging about $8 per ton the average return has been 83% of the gold and silver, the gold alone being close to 90%. Higher results have been at times attained, and the indications are that the average for the present year will tend to approach these figures. The total extraction from the ore is as given herewith; the third column gives the results for a $20 ore, this being about the average for the mine:

	Per cent of product.	Per cent of crude ore.	Value.
Recovery in the mill	54	54.0	$10.80
Recovery in the slime plant	83	38.2	7.64
Total recovery		92.2	$18.44

Comparing this with the old method by percolation, we have the following:

Recovery in the mill	64	64.0	$12.80
Recovery in the cyanide plant	70	25.2	5.04
Total recovery	..	89.2	$17.84

This gain of 3%, or 60 cents per ton is small and hardly more than enough to cover the extra cost which on the basis of work during last fall is 22c., so that it does not furnish a strong argument for fine grinding in cases where the whole product is amenable to percolation. But so far as this ore goes, and many like it, the comparison is by no means fair. Under the former method of treatment 30% of the tailing was left in the ponds, being unfit for any but slime methods. The revised comparison then would be as follows:

Old method.	Percentage of product.	Percentage of crude ore.	Value.
Recovered in the mill	64	64.0	$12.80
Discharged from the mill	36	36.0	7.20
Left in the ponds	30	10.8	2.16
Going to cyanide plant	70	25.2	5.04
Saved in cyanide plant	70	17.6	3.52
Total saving		81.6	$16.32

This shows a gain of $2.12 per ton, ignoring any difference in cost. In our work last fall this came to $0.22 increase, so that the net gain is $1.90 per ton. Roughly, the plant cost $60,000, though it could probably be duplicated for one-fourth less, so that on the basis of the tonnage of 20,000 per year the gain represents 63% return per annum on the actual cost of the plant, fully justifying the investment. In addition to this gain is the ability to treat some 40,000 tons of accumulated slime, which has thus become an asset. The above comparison is on the figures of cost actually attained; using the cost of $2, estimated for the future, the gain would be $2.37 per ton.

The following table gives the segregation of cost for the various departments:

General expense, including superintendence, watchman, assays, insurance, taxes, chemicals, supplies, etc.	$1.202
Re-grinding	0.572
Moore process	0.314
Zinc room	0.261
Total in slime plant	$2.349
Ponds, collecting and handling, with team	0.120
Grand total	$2.469

By the disc-harrow percolation method, formerly followed, the cost was $2.25. This figure of $2.47 is for a restricted tonnage, and careful estimate on a basis of 3,000 tons per month brings the cost down to $2. Future work should improve even on this. Mention has been made of the accumulated tailing. This material has been made available for immediate treatment by the provision of a stock-bin outside the grinding-house. It is connected on the one side with the ponds by an inclined track and a stationary hauling system and on the other with the tube-mill by a steep launder, into which the tailing is fed from the bin, mechanically. In the case of ore of this kind with qualities of very slow settling, the addition of dry material is a decided advantage to the plant, as it increases the percentage of solids and lessens the time of settling. It is probable that this outside stuff, not having participated in the violent agitation with cyanide solution accomplished in the mortars, and moreover only coming into contact with the solution that reaches the tube-mill already weakened from its original strength of two pounds by the 'cyanicides' in the ore, does not yield as highly, and so cuts down the average extraction pro rata.

To the experienced cyanide man a question naturally presents itself as to the accumulation of weak solution. The addition to the bulk of solution comes from moisture in the ore and from the wash-water, the losses from evaporation and leakage and from the discharged tailing. On the basis of the dry ton, the additions are 0.09 ton + 0.7 = 0.79, while the determinable losses are 0.47 ton. This indicates an increase that is not found in practice. But if it were, the extreme weakness of the solution would make such a gain not of first moment. Just how the difference is to be accounted for, is not clear, but it is due in part at least to the difficulty of correctly determining the moisture in the discharge.

For years the concentrate has been treated in a pan, with a lye and bluestone charge, giving results that were only satisfactory because no way had appeared of bettering them. Soon after the slime-plant had been put in operation, the experiment was tried of treating the concentrate with strong cyanide solution, and with such good results that the method completely supplanted the old. The details are as follows: The concentrate consists largely of iron oxides with a little pyrite. This is charged in one-ton lots into an ordinary five-foot silver pan. 25 lb. lime is added and enough water to bring the pulp to about 45% solids, giving the consistence of thick cream. The charge is ground for 48 hr. and then cyanide is introduced, to bring the solution up to a strength of 24 lb. per ton. Grinding is continued with the addition of lime and cyanide at intervals, the one to insure alkalinity and the other to keep up the strength to the original. After about 72 hr. it passes to the settler for 24 hr. further agitation and finally is diluted with mill solution and turned into the slime-plant flume. Toward the end of the operation, samples of the pulp are taken at intervals, washed clean of solution and assayed, as a check upon the extraction.

An interesting feature is that during the grinding with lime, an oxidizing action—a sort of wet roast—appears to take place, the pulp changing from a dirty green to a brownish red. The results of this work for some 30 tons—one half the year's product—with an assay-value of over $150 per ton, are: Extraction of gold, 96.8%; of silver, 84.1%; total, 94.9%; the cost is $7.91 per ton. The consumption of cyanide is 18 to 20 lb. per ton. An excellent comparison of the new and the old method is obtained by considering the work of the first half of the year, which was by pan amalgamation: Extraction of gold, 87.1%; of silver, 74.9%; total, 84.4%; cost, $13.38 per ton. On $150 concentrate the gain per ton amounts to $15.75 in extraction and $5.47 in cost, a total of $21.22.

To the advocate of fine grinding the results I have given may appeal with too great force. Stamping, if followed at all, must almost of necessity be in solution, and the low cyanide content demanded by more than one factor is apt to give low results as regards the silver. In an ore which, by reason of absence of coarse gold, will yield creditably to cyanide alone, amalgamation can be dispensed with and probably less solution used. But it is doubtful if even then a solution strong enough for good silver extraction could be used without excessive loss. Certainly, however, in ore of the Standard type fine grinding gives higher results for the same cyanide strength and the same extraction for a lower strength. Between the two there is intermediate strength that gives the maximum commercial result, and in all tests with fine grinding the aim must be to determine this point. As a comparison between the general Moore-Cassel-Butters method and filter pressing, it seems that the former has great inherent advantages in cheapness of installation and operation, and growing familiarity with the details should make this increasingly evident. In comparison with decantation, apart from the fact that for some ore decantation is entirely unsuited, it does away with the enormous bulk of solutions, from the repeated washings. In fine, it is believed that once metallurgists become better informed on the principles of the system, it will be found worthy of more consideration than heretofore.

TUBE-MILLING IN KOREA

(September 22, 1906)

The Editor:

Sir—We are at present experimenting on a practical scale with re-grinding our concentrate in a tube-mill with cyanide solution. It occurred to me some months ago that cyanide solution could be used just as well as water, when reducing our concentrate to slime, and incidentally some of the gold could be extracted while

Fig. 7 Experimental Cyanide Plant

grinding. In other words, the tube-mill could be used as a grinder and agitator combined.

This company (Oriental Con. Mining Co.) has constructed an experimental plant consisting of one tube-mill (2½ by 12½ ft.), two mechanical agitators with plow shoes (8 by 6 ft. diam.), three filter and settling-boxes (4 by 5 by 5 ft.), 48 separate-compartment zinc-boxes, two sumps, and one stock-solution vat. An electric motor furnishes the necessary power. The tube-mill was constructed in this company's shops.

The method is a continuous one; the concentrate is put into the hopper with cyanide solution and upon passing through the tube-mill is nearly all ground to slime and a good percentage of the gold is extracted. At the end of the mill is a spitzkasten (3 in. wide by 1½ ft. long by 1½ ft. deep) supplied with clear water from the bottom. What discharges from the bottom of the spitzkasten is coarse concentrate and clear water, that passing over the spitzkasten is the cyanide solution and slime. The coarse con-

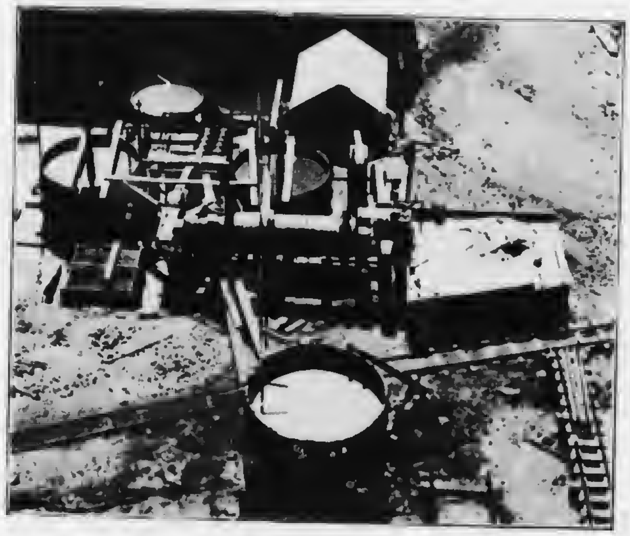

Fig. 8. Another View of the Same.

centrate that may escape from the tube-mill is caught in the settling-boxes and again re-ground, while the overflow contains very little value (tests 0.005 to 0.01% KCN and assays six cents per ton). We cannot arrange to run this overflow or waste-water solution through a separate line of zinc-boxes. In this small plant the value of this waste solution will not amount to more than $2 to $3 in 24 hours. The slime concentrate and cyanide solution pass on to an agitator which has been set in motion (13 rev. per min.) and the muller lowered. Here the product is agitated for about 15 hr., the muller is then raised two feet, the agitator stopped and the concentrate allowed to settle. It takes about one hour to settle clear; by the addition of some milk-of-lime it could be made to settle in half an hour. I prefer to use as little lime as possible because the strong alkaline solution consumes a large amount of zinc. The clear cyanide solution is decanted by a float siphon into filter-boxes and from there it is run through the zinc-boxes and the gold precipitated. Next, the strong precipitated cyanide solution

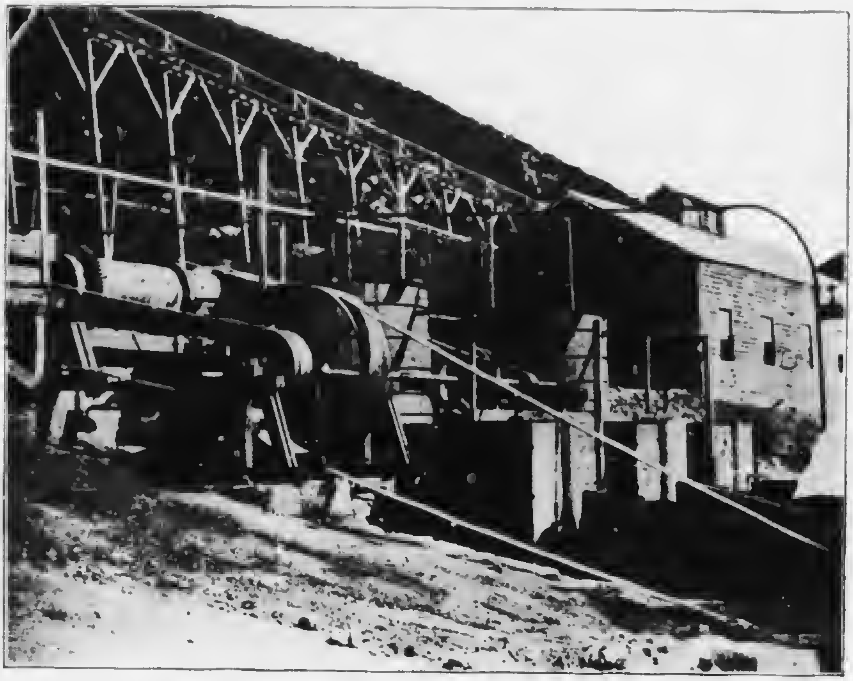

Fig. 9. Tube-Mill and Experimental Cyanide Plant

from the sump is pumped up to the agitator as a wash to partly remove the gold remaining within the settled concentrate. The muller is allowed to work down on the charge and agitate for a few minutes, then settled and decanted. Following this come two successive weak cyanide washes to remove cyanide and gold, and finally a water-wash which is run through a row of zinc-boxes to waste. Now the charge of about five tons of clean concentrate is ready to be discharged into the creek. A little water is added, the muller set in motion and lowered, the discharge-hole opened, and now the agitator will discharge itself readily and prepare for a new charge. The total treatment takes 24 hours, both agitation and decantation being done in the agitators

By feeding the concentrate and cyanide solution with a consistence of one to one through the tube-mill, we are able to get the finest grinding and the best results. The agitators will only have to be filled once to contain a full charge. While one agitator is being filled, the other is decanting and getting ready to be discharged At the head of the launder (leading from mill to agitator) a pipe was run to the stock cyanide vat above, so that we can add enough solution to keep the launder clear. With clean concentrate it is necessary to have a steep grade.

The strong cyanide solution (0.43%) is used in the tube-mill and agitators, while the weak one—employed for washes—tests 0.1%. No cyanide is added to the weak soluton. Lime is added (about two pounds per ton) with the concentrate as it is fed into the mill.

The old method of cyaniding concentrate here is by mixing with 48% sand and percolating in vats from 20 to 30 days. This method gives an average extraction of 80 per cent.

The extraction obtained at present with the tube-mill and agitators on clean concentrate is 93%. By gradually making improvements we may possibly better these results.

A. E. DRUCKER.

Chittabalbie, Korea, July 20, 1906.

CYANIDE PRACTICE AT EL ORO—I

By T. A. Rickard

(September 29 1906)

The development of the milling practice at El Oro is full of interest. In 1873 a *hacienda de beneficio*, or reduction plant, was erected to crush ore and treat the accumulated tailing from a still older *arrastre*, and to this plant further addition was made in 1885. The mill then included 25 stamps with amalgamating tables. In 1890 the accumulation of tailing made by the stamps was sold to a man from Butte, named Albertson. The tailing he handled was richer than the ore being mined today. Nevertheless, the contract for the treatment of it was cancelled after the purchaser had installed four amalgamating pans with settlers and had started to ship bullion. This was under the regime of General Frisbie. In 1894 a Chilean mill was brought from Chicago, to grind the ore after it passed through a Comet crusher. The Chilean mill did finer grinding than the stamps, which at that time were also preceded by crushers, of the Blake type. The mill in turn left a dump which, eventually, as methods improved, it became profitable to re-treat. Late in 1894, James B. Haggin bought control. In the following year the old-mule stable was converted into a cyanide annex. Redwood tanks with 4½-ft. staves and 24 ft. diameter were erected; the sump-tanks were larger, with 6-ft. staves. The tailing was carried, in boxes on the backs of *peones* and in hand-barrows, to the vats. Cyanide solution was first introduced by upward percolation through a false bottom, the succeeding water-washes being applied from above. This was followed by precipitation on zinc shaving, with acid treatment for the zinc 'shorts,' the bulk of the precipitate being carefully washed and melted forthwith. The bullion thus obtained was of extraordinary fineness—960 to 980—without the use of any nitre in the melting. This was one of the first successful cyanide plants in Mexico. With only the addition of the small cyanide plant just described, the mine paid $1,000,000 in dividends up to May, 1898, besides meeting the cost of various installations, including part of the 100-stamp mill taken over by the English company, which now controls the property.

The first 100-stamp mill was designed under the Haggin-Frisbie regime and was only expected to crush 4,500 tons per month through a 60-mesh screen. When the property was purchased by the Exploration Co. in 1898, this mill was too near completion to be altered. The slime-plant was added in 1900, after the present company had been formed. W. K. Betty had conducted a series of experiments for the new owners and double treatment was then adopted for the slime-plant; it was only making the best of conditions as they were found; hence the pile of stored tailing now about to be re-treated.

The general plan of treatment was as follows: From the stamp-battery the pulp passed over copper plates and was then divided, by spitzkasten, into 'coarse sand,' 'fine sand' and 'slime,' each product receiving individual treatment. The sand underwent double treatment, in South African style; it was first cyanided in collecting vats and then dropped into cars which removed it to the treatment vats. The slime was caught in a settling vat and thence went to the treatment house, where it was agitated by jets of compressed air. After treatment, the sand was dropped into cars underneath the vat, while the slime was flushed out with water in the ordinary manner.

In the meanwhile the capacity of the mine grew, not only by reason of the discovery of new orebodies, but indirectly through the cheapening of operations, so that further enlargement of the mill became prudent. In 1905 another, and the last, addition to the reduction plant was made. The new mill of 100 stamps, with its up-to-date cyanide equipment, differs from the old one in five respects:

1. Mechanical handling of the ore.
2. Heavier stamps.
3. Re-grinding in tube-mills.
4. Mechanical handling of sand by distributors, excavators, and belts.
5. Mechanical agitation of slime by stirrers and centrifugal pumps.

The new mill contains 100 stamps, each weighing 1,180 lb., falling 102 times per minute, with a 6-in. drop. The depth of discharge is 2½ to 3 in. with a new die, and 3½ in. when the die is worn out. Woven brass wire-screens of 35 mesh are used.

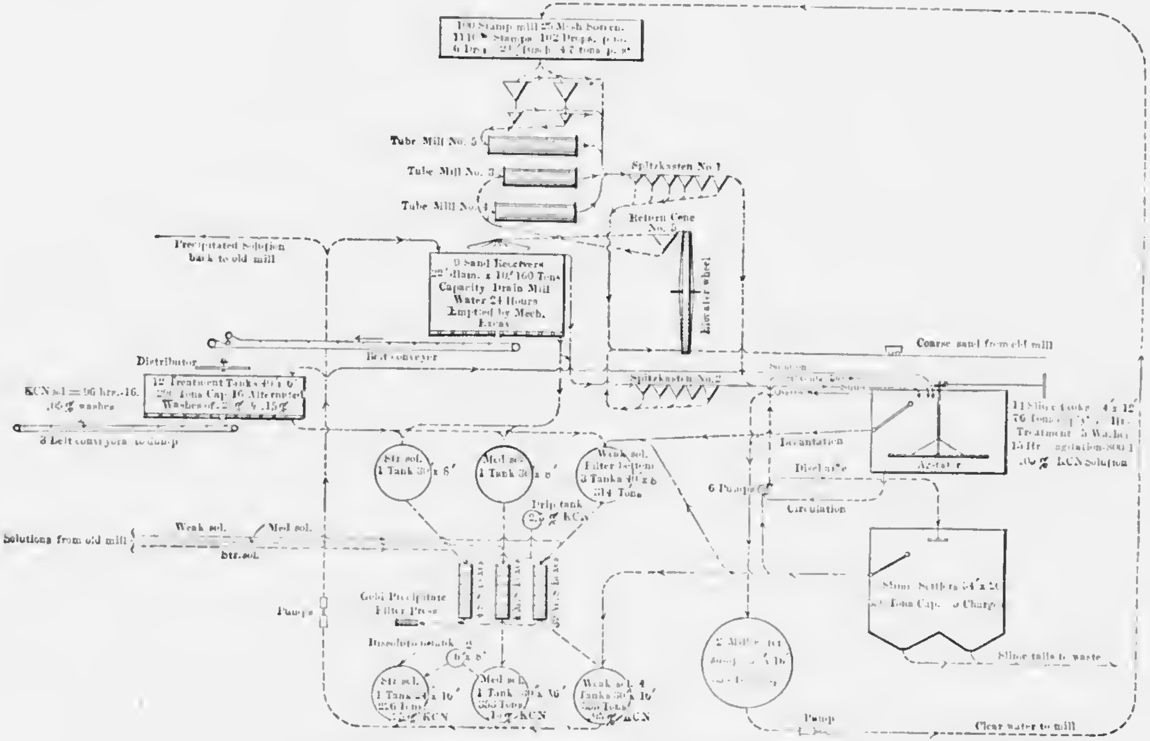

Fig. 10. Diagram of Cyanide Treatment at Plant No. 2.

The accompanying diagram* (Fig. 10) illustrates the process. From the stamps the crushed ore goes to a system of cone-classifiers and spitzkasten which separate the coarsest sand and send it to the tube-mills for re-grinding. The fine sand from the stamps combines with the similar product from the tube-mills and is elevated by the raff-wheel to the sand-collecting vats. Any slime which may have escaped complete separation and accompanies the sand, overflows from these vats and passes to the slime-plant, joining with the rest of this product that has been eliminated from the sand by the classifiers. The sand is distributed by a revolving mechanism of the Butters & Mein type. There is no chemical treatment in the sand-receiver, the idea being to keep the mill-water free from cyanide while effecting a final separation of slime, so as to get a clean product. The water and slime are drawn off through gates or slots on the side of the vats; these gates are closed by a roll of canvas as the vats fill. The sand, when thus finally freed from the last trace of slime, is removed by a Blaisdell excavator which drops it through a central opening onto a Robins belt-conveyor. This Blaisdell excavator is like a revolving disc-harrow and it has proved a most efficient machine. It uses comparatively little power and works smoothly. The belt conveyor takes the sand (containing now only from 10 to 11% moisture) to the treatment vat, which is fed by a revolving distributor operated by a variable-speed motor, the centrifugal force being so regulated as to throw the sand to the sides or centre of the vat, as required. The charge is 265 tons, dry weight. Ten washes of alternately medium (0.1%) and strong (0.2%) solution are introduced, six hours apart. This treatment is followed by no less than thirty 'weak' washes, such a lengthy operation being specially designed to extract silver. These 'weak' washes are four to six hours apart and contain 0.03% KCy. Each wash is equal to 13 tons of solution. After treatment, the residue, again using the Blaisdell machine, which moves on rails, is discharged onto a conveyor that takes it to the dump. Here the distribution of tailing is regulated, as the accumulation grows, by a hinged belt-conveyor in two lengths, the last one being swung round according to the contour of the ground.

*Borrowed by permission from 'The Grinding of Ore by Tube-Mills, and Cyaniding at El Oro, Mexico,' by G. Caetani and E. Burt. Trans. A. I. M. E., February, 1906. This is a conscientious and most valuable paper, giving a detailed account of the cyanide practice at El Oro.

The slime goes to a collecting vat, from which the thick mud is drawn off at the bottom and thrown into one of the treatment vats. There are twelve of these, each 34 ft. diam. and 12 ft. deep. Here it is agitated with a proper proportion of cyanide solution, which is introduced simultaneously. The apparatus for stirring consists of two long and two short arms made of oak bolted to a steel star. The oak arms are solid; they taper outward from a cross-section of 4 by 6 in. to 4 by 4 in. The thick end is bolted to the steel star, which is set on a vertical shaft. When the vat is charged, lead acetate is added immediately. Tests have shown that a beneficial result ensues forthwith, particularly as regards the dissolution of the silver.

Lead salts when added in excess to the cya[illegible]lution, give a precipitate of basic lead cyanide, but when present in small proportion the lead remains in solution, presumably owing to the formation of an alkaline plumbite (K_2PbO_2) by reaction with the caustic alkali, thus:

$$PbAc_2 + 4KOH = K_2PbO_2 + 2KAc + 2H_2O$$

Mercuric chloride is sometimes employed for the same purpose, producing a reaction with the KCy so as to form a soluble double cyanide, thus:

$$HgCl_2 + 4KCy = K_2HgCy_4 + 2KCl.$$

The most useful effect of these soluble lead and mercury compounds is the removal in the form of insoluble HgS and PbS, of any soluble sulphides that would otherwise retard the solution of gold and silver, and which may even re-precipitate silver already dissolved:

$$K_2S + K_2HgCy_4 = HgS + 4KCy$$
$$K_2S + K_2PbO_2 + 2H_2O = PbS + 4KOH.$$

The double mercuric-potassium cyanide also acts as a solvent, attacking gold more readily than simple KCy; and this aciton is independent of the presence of oxygen, gold replacing mercury:

$$K_2HgCy_4 + Au = K_2AuCy_4 + Hg.$$

Silver is similiarly dissolved. These reactions have been amply verified. The action of mercuric-potassium cyanide on gold is the basis of patent secured by Keith and Hood; the latter also claims the use of lead as facilitating the solvent effect of cyanide solutions. De Wilde has a patent involving addition of lead oxide to cyanide solution. These compounds also influence precipitation beneficially if they remain in the solution up to the point of entering the zinc-box, as in that case the lead and mercury are precipitated on the zinc, forming zinc-lead and zinc-mercury couples of high electro-motive force. In this precipitation the zinc simply changes places with the mercury or lead, as is also the case when zinc shaving is dipped in lead-acetate solution.

The charge is 60 tons (dry weight) of slime; this is mixed with a solution in the proportion of 2½ solution to 1 of slime, by weight. The solution contains 0.05% KCy*. Agitation continues for six hours. The vat is then filled until there is 3½ of solution to 1 of slime; this is well stirred and then allowed to settle. Settling and decantation consumes eight hours. This part of the process is hastened by the use of lime, which is added to the feed of the tube-mills.

The lime has two functions, one of them chemical, the other physical. By virtue of the first it neutralizes the sulphuric acid and decomposes the ferric sulphate contained in the ore, and due to oxidation. Such oxidation may have occurred in parts of the lode before it was mined, or it may have been developed by subsequent contact with the air in its passage to the mill or during treatment. The lime serves in this way to protect the cyanide of potassium or sodium, as the case may be. In slaking, the calcium oxide (CaO) takes up water to form the hydroxide ($Ca(OH)_2$) which dissolves in water to the extent of one part in 800. Lime is preferable to caustic soda, for this particular purpose, because the calcium carbonate is insoluble in water, while the sulphate is but slightly soluble, so that they do not accumulate in the cyanide solution, as is the case with the corresponding sodium salts where $NaOH$ is used as the neutralizing agent. Soluble carbonates are also precipitated by it, leaving caustic alkali in solution, thus:

$$CO_2 + Ca(OH)_2 = CaCO_3 + H_2O.$$
$$Na_2CO_3 + Ca(OH)_2 = CaCO_3 + 2NaOH.$$

By reason of its physical function in the mill, lime coagulates slime, so as to cause settling of the particles. The effect is complex.

*Sodium cyanide is used, but all calculations are made in terms of the equivalent potassium cyanide. 100 lb. NaCy is equal to 128 lb. KCy, therefore in practice eight-tenths of NaCy does the work of one unit of KCy. The chemical action is the same, the lesser freight on the more concentrated form of the cyanide making the sodium preferable to the potassium salt.

Much of the material classed as slime is of a colloid nature—indeed slime has been recently labeled a 'colloid hydrate.' Such matter when brought into contact with pure water becomes almost gelatinous, and therefore impervious to solution. There are several substances, notably alum, acids, soap, and lime, which, when added to the turbid water, cause the gelatinous matter to coagulate or flocculate, so as to produce a separation into distinct agglomerations. Further, minute particles of ore, whether slimy or not, if suspended in water and refusing to settle, develop a tendency to subside when lime, alum, and other substances are introduced. Although imperfectly understood, these reactions are used largely both in metallurgy and in agriculture.

The slime settles rapidly; within two minutes there is an inch of clear water. This clear solution is decanted and passes to the filter-vat, the bottom of which is provided with two or three feet of sand on the top of burlap. This removes any remaining trace of slime, cleaning the solution so that it is fit to go to the precipitation-house.

Returning to the treatment-vat; the slime remaining after decantation undergoes further agitation. The vat is filled with a 0.03% solution and agitation ensues for 1½ hours. Then follow three more successive washes. The vat is then filled for the fifth time and the mixture is thrown by a centrifugal pump into a deep settling-vat. Five of the treatment charges go to one of these vats, of which there are six, each being 20 ft. deep and 34 ft. diameter, with a capacity of 450 tons. The successive charges from the treatment-vat are fed into one settling-vat until it is full of slime, for as fast as the solution gathers on top it is run off, just sufficient time being given for clarification. This clarified solution is so poor in gold and silver that precipitation is not attempted, the solution being used as the first of the washes in the treatment-vat.

The new mill contains three tube-mills. All of them were made by Krupp, at Essen. The No. 3 mill is 19 ft. 8 in. long with 3 ft. 11 in. diam.; No. 4 is 4 ft. 11 in. diam., and 23 ft. 9 in. long, while No. 5 is of the same diameter as the last, but 26 ft. 3 in. long. The smallest of the tubes is found to do most work per horsepower required. In Western Australia the tubes or grit-mills (as they are often called) have been cut down to a length of 13 ft., but the ore at Kalgoorlie is softer, so that grinding is more quickly accomplished than at El Oro. The time required is determined directly by the hardness of the rock, for the ore is fed at the upper end and makes its exit at the lower, through a screen. Of the three types of tube-mills, the Abbé can be filled more than half full; this cannot be done with the Krupp mill because it both fills and discharges at the centre. The Davidsen has central feed but peripheral discharge, while in the Abbé mill this is reversed, the feed being peripheral and the discharge central. The last mentioned is built in sections and the driving is done on tires and by gears which circle the exterior of the shell, like a Bruckner furnace. The Krupp tube is made of wrought-iron sheets, welded; it runs on trunnions placed at one end, so that the shell does not come into play as regards the driving of the machine.

The lining of tube-mills is an important matter. Chilled cast-iron, both that imported from Krupp's works and that made by El Oro company itself, has been tried; the latter costing one-half the former and giving equal wear weight-for-weight. Krupp's lining is from ⅞ to one inch thick. El Oro lining is 1¼ in. thick. Nevertheless, it is the intention of the manager* to substitute silex, a natural flint with characteristic conchoidal fracture; it is whittled into shape in Germany before shipment, arriving in sections 2½ in. thick, 4 in. wide, and 6 in. long. The pebbles that do the grinding come from the coast of Denmark. They vary in size from that of an egg to that of a fist, the average being about three inches in diameter. They wear well, 6 lb. of pebbles being abraded during the grinding of one ton of sand; the consumption of lining being 1.6 lb. An attempt is being made to select some of the flinty quartz, such as occurs in the low-grade ore of the mine, to serve as grinding material. This seems wise; if the hard portions of the ore can be used to grind the soft, the economy is obvious.

At the time of my visit, No. 3 tube was being driven at the rate of 31 rev. per min., while No. 4 and No. 5 made 29 revolutions. The duty of the individual tube-mills cannot be stated; 172 tons of the coarsest sand from the new 100-stamp mill is re-ground from 35 to 150-mesh, or finer, by the three tubes. In addition, 85 tons per day of the coarsest of the 40-mesh sand coming from the old 100-stamp mill is reduced to the same condition, making the total work of the three tubes 257 tons.

The tube-mills get everything above 150 mesh, as separated by cone classification. The aim is to grind to 150 mesh and this

*Robert M. Raymond, to whom I am indebted for much valuable information is accomplished as nearly as the capacity of the plant will permit. Any oversize is returned as already described for re-grinding. The cyanide treatment is based on making a product of sand as nearly 150-mesh as possible, while the 200-mesh and finer are treated as slime. This tube-mill practice has steadily gained in importance, the tendency being to treat a larger proportion of the product from the stamps and to augment their crushing capacity, while enlarging the cyanide annex. This is a proper way of meeting the necessities of a mine, the output of which increases in tonnage as the grade declines.

CYANIDE POISONING

September 29, 1906

*In cases of poisoning, everything depends on prompt action, for the chance of recovery is extremely small after the lapse of a very few minutes if a fatal dose has been taken. One person should be dispatched for the nearest medical assistance that is available, but no delay in treatment should be permitted to occur on this account. The first care must be to neutralize the rapid poison by the antidote, and then to empty and wash out the stomach as soon and as completely as possible.

The antidote consists of two solutions sealed up in bottles, and a sealed powder. The two solutions are to be first mixed together in the tin vessel in which they are packed, by breaking off the sealed ends of the bottles. The tube containing the powder is also to be broken and the whole of the powder added to the mixture, and the dose is to be administered as soon as can possibly be done. If the patient is still conscious he must drink the antidote at once without waiting for the insertion of the stomach tube, but if not conscious, or not responsible, then a small gag must be firmly inserted between his teeth, so as to prevent the stomach tube from being bitten off, and the tube is then to be passed down his throat and into his stomach. The antidote is to be poured down the tube, and is then to be followed by some water.

In any case, either before or after the antidote has been taken, the stomach tube is to be inserted, and about half a pint of water is to be poured down it, the patient being placed in a reclining position, a little raised from the ground. The insertion of the tube may produce vomiting; this, however, is entirely favorable to the course of the treatment. When the last of the water is placed in the funnel, and before it has all descended into the tube, the funnel end of the latter is to be lowered so as to cause the tube to act as a syphon, and the stomach emptied as much as possible of its contents. Fresh water is to be poured down the tube, and the stomach again emptied, and this is to be repeated several times, so as to thoroughly wash out the stomach. When this has been done, the tube can be withdrawn.

If the tube be not at hand, every endeavor must be made

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*From the *Monthly Journal* of the Chamber of Mines, Western Australia.

to induce vomiting after the administration of the antidote, while an equal endeavor must be made to cause the patient to swallow more of the antidote between the intervals of vomiting, if the administration be not already and completely made.

Vomiting may be induced by an emetic, like mustard, or by tickling the back of the throat with a clean feather, or a piece of clean india rubber tube, or the finger. An ample quantity of warm water should be swallowed and vomited, so as to wash out the stomach as in the previous case. Should warm water not be at hand, cold water may be employed in its place.

As soon as the stomach has been satisfactorily emptied and washed, and the stomach tube withdrawn, steps should be taken to bring about artificial respiration. Should the patient appear to be in a state of collapse, and his breathing have ceased to be noticeable, the application of smelling salts or of ammonia to his nostrils may itself induce breathing again; but if this be not immediately successful, the patient should be treated as is done in cases of partial drowning or suffocation.

Medical assistance should by this time have arrived; indeed, if the patient be conscious, there is now great hope that the worst effects of the poison will have passed off, particularly if the details of the treatment have been all carried out, for cyanide poisoning is usually fatal within twenty minutes.

The package for treatment should consist of a tin vessel with lid, in which are packed:

(a) A hermetically sealed bottle, containing $7\frac{1}{2}$ grams of ferrous sulphate dissolved in 30 c.c. water; and

(b) A hermetically sealed bottle, containing $1\frac{1}{2}$ grams of caustic soda dissolved in 300 c.c. water; and

(c) A tube containing two grams of magnesia.

There should be also a gag for the purpose of opening the clenched mouth of an unconscious person, and a stomach tube, that can be passed through the gag and down into the oesophagus into the patient's stomach. This is very easy to effect, but several persons in charge of the plant should receive instructions from the nearest medical man as to how to insert a stomach tube so that they may know how to use it should occasion at any time arise.

The apparatus should never be allowed to be removed from its place, but always kept complete and ready for an emergency. It is advisable to keep it in duplicate, in prominently marked positions in the works.

CYANIDE PRACTICE AT EL ORO—II

By T. A. Rickard

(October 6, 1906)

A few scattered notes on El Oro mill may be worth recording. The bolts of the battery frames are coupled by washers; these are 6 to 10 in. long and from 2½ to 3 in. wide; they connect two bolts and hold them firm. If one gets loose, the other holds it in grip and prevents movement.

The guides are made at the company's foundry, of cast iron; instead of being sectional with bolts, they are one solid piece. Each stamp has its own guide and a right-angle plate, to keep it in proper place and line. The wear is slight and therefore the stamp works smoothly; there is less heating than with wooden guides.

The mortar is a development of the anvil-block. This is an excellent mode of construction, if properly done. I know of one case—not in Mexico—where trouble was caused by the anvil-block being constructed so that it did not rest perfectly true on the cement foundation; to remedy this it was the custom to shim the concrete block with a little cement; when this last broke and crumbled, there was a movement of the mortar itself. At El Oro, the mortar-block is made extra heavy, combining to some extent the anvil in itself, with a base three feet wide and a bottom 13 in. thick; this is placed upon a concrete foundation, with a piece of quarter-inch rubber belt between.

At El Oro, cones are superior to spitzkasten; the sizing tests have proved this abundantly, the separation by the cones being much sharper. The circulation and agitation of slime are aided by six pumps which are the Butters modification of the Gwynne pump, such as is used in the London dock-yards. They are of the centrifugal type; compressed air is introduced to effect aeration of the solution. The chief advantage of the Butters modification is that all wearing parts are readily removable. Each pump makes 1,390 rev. per min. and in that period handles 4½ tons (3½ tons being solution) of slime.

The vats are all made of steel plates, 3/16 in. thick on sides, with ¼-in. bottoms. Redwood laid down at El Oro comes to the same cost, but the steel is more durable and makes a tighter vat in the climate of central Mexico. The vat does not dry if empty, there are no staves to check, and no absorption of solution.

In the precipitation house, there is used a device introduced independently by W. K. Betty in South Africa and by Alfred F. Main at El Oro. I refer to a drop-drip of KCy ($2\frac{1}{2}$% solution) over the head compartment of each zinc-box that is precipitating from the weakest solution namely, that coming from the treatment of slime. This 'drip' makes the zinc more active, so that a precipitation of precious metal is obtained in a manner usually unattainable from so weak a solution, that is, one containing only 0.02% KCy. Still weaker solutions are successfully precipitated in which the quantity of KCy is so small as not to be detected by the ordinary silver nitrate test.

The method of dipping the zinc shaving in lead acetate (to aid precipitation) is not employed at El Oro because lead acetate is used at another stage of the process, as already explained. Zinc fume was tried, but it was ineffective with such weak solutions. Great care is taken with the zinc shaving, to cut it in thin but tough filaments, not so crinkly as to break easily in handling. The shaving is laid in the boxes most carefully, so as to avoid any tendency to channeling. The El Oro plant is the only one of its size where acid treatment is not used. From the boxes the zinc is sent through launders, to be carefully screened, while it is also being washed with fresh water. Then it is pumped into two filter-presses until they are full, the charge being equivalent to 19,000 oz. of bullion. The effluent solution is returned to the sump, the cakes in the press are washed and then dried by steam, the steam heating the iron, of the frame sufficiently to dry the cake inside. The cakes are dried to such a consistence as will facilitate fluxing before briquetting; they fall into a car and are then mixed with the fluxes needed for melting; the mixture is fed into a briquetting machine, making round bricks $3\frac{1}{2}$ in. thick with 3 in. diam. These are dried before being thrown into the melting pot, from which bars of 1,000 oz. are cast. The Mexican workmen are compelled to remove their clothes after work, before passing to the outer room. The precipitation room has a cement floor and the furnace has a dust-chamber.

The development of milling at El Oro emphasizes the relative importance of the cyanide annex in the modern wet treatment of precious-metal ore; the annex to the new mill required an expenditure a little over twice the cost of the new 100-stamp mill itself. The tendency is to increase the percentage that is re-ground, the perfection of the extraction being closely determined by the fineness of comminution. At the time of my visit the aim was to make two products; sand, as near 150-mesh as possible (and a decreasing percentage even of that) and slime, that is, all below 200-mesh. Of course, the sand, even when re-ground, is different from the clay despite equality in size of particles; the grains of 'sand' are sharp as against those of a mud (slime) rendered impalpable by absence of sharp edges. 'Sand' however fine, filters well, while 'slime' will not filter at all; it packs like glue. On the other hand, by reason of the relatively larger surface presented by minute particles, chemical action on the precious metals is almost instantaneous. How necessary re-grinding is, was shown by a simple experiment made by Mr. S. H. Pearce. Sand, after ordinary cyanide treatment at the old mill, where there is no re-grinding, was dissolved in *aqua regia*, but the 'purple of Cassius' test, with stannous chloride, gave no precipitate whatever, the gold being effectively locked within the grains of sand. The assay of the sand gave $4.50 per ton. Hence the need for re-grinding.

The accompanying record of tests will prove interesting to those engaged in cyanide work. Looking at Fig. 11, it will be noted that the legend explains the graphic representation of two sizing tests. At the time of these tests, a 2½-in. chuck-block was used, but it was too low to have much effect on the degree of fineness of the product; during the test the stamp-discharge was as through 28 mesh. Under these conditions the load on the tube-mills and on the plant became too heavy, so that finer screens were substituted shortly afterward. In the diagram (taken from the paper by Caetani and Burt, already mentioned) the ordinates represent the size of the screen and the abscissae the percentage retained on each of the screens. The legend "Thro' 250 mesh" should read "through 200 mesh."

The use of the term sand-index, to be seen in the note appearing on the diagram, requires explanation. Caetani and Burt employ it, and it represents one of the most valuable features

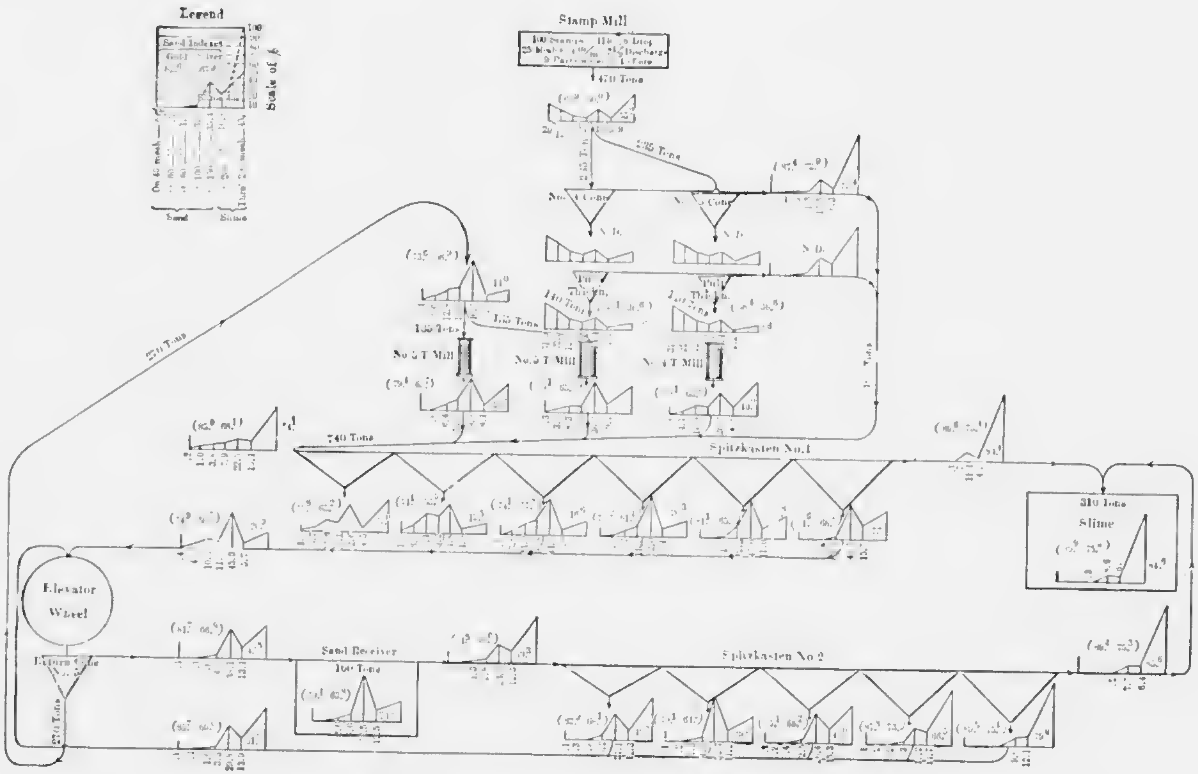


Fig. 11. Diagram of Sizing-Tests at Mill No. 2

of their paper. The problem was the following. Given two sands of the following analysis:

Mesh	On 20 %	On 40 %	On 100 %	On 200 %	Through 200 or slime
1st sand	10	30	25	5	30
2nd sand........	5	15	45	15	20

Which of these two sands is the finest? Caetani answers the question from the economic point of view, thus: It is desired to know the fineness of a sand for the reason that the finer the sand, the better the extraction obtained. Therefore the maximum possible extraction on a sand of given composition *is a number proportional to its fineness, considered from an economic standpoint.* As at El Oro one can *a priori* calculate exactly the extraction from a sand when a sizing test has been made, therefore one can calculate the index and represent thereby with *one* number what would otherwise have to be indicated by a tabulation consisting of 14 numbers. In the examples quoted at the beginning of the paragraph, the second sand is finer than the first, although it contains less slime.

REPORT OF CYANIDE DEPARTMENT, SEPTEMBER, 1905.

MILL NO. 1.

Classification –	%	Tons treated.	Gold: Assay value per ton. $	Gold: Indicated extraction. %	Silver: Assay value per ton. $	Silver: Indicated extraction. %
Coarse Sand	29 23	2,552	9 46	54 33	1 73	27 47
Fine Sand..........	25 57	2,233	7 86	72 44	4 57	45.86
Slime	45.20	3,947	9 04	93.58	2 03	82 26
Total	100	8,732	8 86	76 50	1 83	58 99

REMARKS. Old mill built before re-grinding was adopted. Fine sand poorer than coarse because it contains less gold open to attack. Slime richer in silver by presence of sulphide.

MILL NO. 2.

Classification	%	Tons treated	Gold: Assay value	Gold: Indicated extraction	Silver: Assay value	Silver: Indicated extraction
Sand	24 12	2,527	8 28	83 94	1.59	65 41
Slime	75 88	7,949	7 68	92 45	1.64	78 05
Total	100	10,476	7 82	90.28	1 63	75 08

REMARKS. New mill includes systematic scheme of re-grinding, as hown by increased proportion of slime. Better extraction on slime raises general result to a satisfactory figure.

The Esperanza mill had 120 stamps when the present company took it over, in 1904. It was deemed advisable to increase the capacity at the least possible cost, so 15 Huntington mills (each of 5-ft. diam.), were added, with the idea of re-grinding before cyanidation. This was tried, but it was found necessary to place the Huntingtons above the stamp-batteries, which necessitated elevating the pulp. It being therefore difficult to distribute the pulp to the Huntington mills, it was finally decided to use the latter machines for first grinding, in association with, instead of in succession to, the stamps.

The crude ore passes over a 1½-in. grizzly before it reaches the rock-breakers; after being crushed by them, the ore goes over a ¾-in. grizzly, the undersize being allotted to the Huntingtons and the oversize to the stamps. The batteries are provided with 60-mesh screens; while the pulp issuing from the Huntington mills goes through an angle-slot screen equivalent to 60-mesh, but 65% of the product will pass 200 mesh.

Of the 15 Huntingtons, six are now used as first grinders on low-grade sulphide ore, the product being sized and distributed to six Wilfley tables, the tailing from which, after classification, passes down blanket sluices before finally reaching the cyanide vats. The concentrate from the Wilfleys and that washed from the blankets, goes to the smelter at Aguascalientes.

The other nine Huntingtons treat oxidized ore, which, after being ground, goes to the cyanide annex. The cost of steel and repairs to wearing parts amount to 34 *centavos* per ton; labor averages 15 to 20 *cv.* per ton. The muller shells and die-ring are made of roll steel manufactured by the Midvale Steel Co., of Philadelphia. This is a soft metal and is susceptible of being kept to shape; it can be used until worn out, and is, therefore, economical. Each Huntington mill has its own motor; it has proved itself to be the best machine for reducing the ore to a certain point—say, 60 mesh—beyond which, for finer grinding, it is not economical.

The sand undergoes treatment for 100 hr; for it is found that extraction ceases then. Aeration is effected by a perforated pipe discharging over the return-solution vat; yet there is no such loss of KCy as might have been expected. The former collecting vats are now used for treatment; there is less aeration and less mixing, but there is a great gain in the capacity of the plant without interference with effective percolation. A vacuum-pump, for withdrawing the enriched solution, is used only at the close of the operation. Sodium cyanide, NaKCy, is the chemical employed; it is guaranteed equal to 125% active KCy, ranging from 124 to 128%. The enriched solution, before precipitation in zinc-boxes, is rarely higher than $2.20 in gold. Fresh cyanide, in crystals, is added to the head of the zinc-boxes, sometimes in quantities sufficient to keep the solution up to standard strength.

There are no amalgamating plates, and no mercury is used in the Esperanza plant. This is an interesting divergence from El Oro practice.

During September, 1905, the output of the mine consisted of 5,280 tons of shipping ore and 12,000 tons of milling ore, having together a value of $780,385 U. S. currency. The extraction in the mill was 91.64% of the gold and 52.92% of the silver in the crude ore.

COPPER IN CYANIDE SOLUTIONS

(October 6, 1906)

The Editor:

Sir—As to copper in cyanide solutions, finding facts here at variance with those published in your issue of September 1, I thought an account of our practice here would be of interest as well as our manner of making pure bullion from a zinc-box product high in copper.

The gold-bearing ore consists of hematite and clay in about equal proportions and 0.75-lb. copper. On account of the clay, roasting is required to effect dehydration. Without such dehydration the treatment of slime, which amounts to over 50%, would be impossible, either by filter-pressing or decantation. The latter is the method we employ, using shallow vats 19 in. high, which settle completely in four hours, including the decantation of the clear solution. The loss in cyanide is about four pounds per ton, with a solution containing 1½ lb. cyanide.

The zinc-boxes, of eight compartments each, show pure copper in the last two compartments, and gradually turn black toward the head. The copper is not loose and spongy, but adheres firmly, and to all appearances looks like pure copper shaving. I am inclined to believe that, in this instance at least, copper helps in the precipitation of gold and silver because a copper hue on the zinc has always been a proof that the sump assays only a trace in gold. These coppered shavings when treated with sulphuric acid, and then cupelled, yield gold 1,000 fine; so that the sump must be free from silver.

We have found it necessary on but two occasions during the past two years to dress the zinc in the boxes and then the last compartments were not coated with appreciable amounts of copper. We simply clean up the first and part of the second compartments every month, unless previously gold slime has so accumulated that it retards the proper flow of solution.

The short zinc is not removed. We find in this fact a contradiction to many authors who claim short zinc to be inert. The fact that we can return short zinc with no accumulation of such product, is a proof of its precipitating qualities. Only what can be washed through a 20-mesh screen is treated with acid and this before acid treatment is separated by stirring in water and decanting the suspended slime until only a coarse granular mass is left. The latter is treated with sulphuric and the former with hydrochloric acid. The slime contains a large percentage of calcium carbonate, which forbids the use of sulphuric acid on account of the regulus that would be formed in melting and on account of adulteration of precipitate with sulphate of lime.

Ten parts of the coarse or 12 parts of the slime after acid treatment are melted in a graphite crucible with about eight parts of a mixture of 26 litharge, 20 borax glass, and one nitre. The melting must be quickly performed to prevent formation of too much lead from the action of the graphite in the crucible, and unless considerable experience is obtained as to the proper time and heat, a flux consisting of equal parts of bicarbonate of soda and borax glass would be safer. The use of litharge, on account of its rapid shrinking before complete melting, allows of several fillings before fusion. The flux and charge are mixed after wetting sufficient to prevent dusting.

The smaller bars resulting from different melts are now ready for refining as follows: Place inside of No. 35 graphite crucible a Battersea P crucible so that the bottoms of each touch. To accomplish this, the graphite crucible must be slightly clipped inside for about an inch from the top. Melt the bars in the clay crucible and then add, from time to time, a little nitre until the metal is covered with slag, which should then be skimmed by means of a small crucible (10 grams) held with tongs. Repeat the adding of nitre and skimming until the skimmings show but little lead and the bullion looks bright. On an average, the bars which before refining are 700 to 800 fine, yield by above treatment bars of over 900 fine.

In case the soda flux is used, about five per cent metallic lead should be melted with the bar, as without lead the copper is difficult to remove. The refining takes about an hour after melting. If all the melts are made, using a clay crucible direct, bullion of 985 fine can be produced in the original melt, using either litharge or soda flux. However, as I was not able to obtain larger clay crucibles than the Battersea P size and as the heat must travel through two crucibles, clay breaking in direct contact with coke, I found it preferable to refine in one melt.

C. A. Arents.

Copperopolis, Cal., Sept. 17.

ZINC-DUST PRECIPITATION

(October 6 1906)

The Editor:

Sir—After reading your remarks concerning zinc-dust precipitation I have been inquiring among my professional friends and from one of them I have secured information which I feel sure will interest others, as much as it did me. This engineer was superintendent of a 100-ton sand and slime-plant. Slime was treated by decantation and sand by percolation, the ore containing both silver and gold, with over 50% of the value in gold.

The precipitation plant consisted of three 15-ton flat-bottomed wooden tanks, a six-inch self-acting compressor, a four-inch triplex-driven plunger-pump, and two 2.5-ton filter-presses. One of the latter was a Johnston press with six-inch frames. The leach from sand and slime ran to a sump whence it was pumped to one of the three agitation vats. This pregnant solution was worth about $2 per ton. The compressor was started and air pumped through a ¾-in. line to a grating of half-inch pipes in the bottom of the vat. These branch pipes were perforated. Two to four pounds of zinc dust were added (an average of 3.5 lb.) and the agitation with air continued about twenty minutes. The charge was then pumped by means of the triplex plunger through the press. One press held about one month's product. In order that the press should fill completely, it was the custom to close the discharge-cocks of the frames in the half nearest the inlet, gradually opening them as the farther frames became filled. The plant treating the ore produced a product worth $20 per lb., while the plant treating old tailing from a former pan-amalgamation mill made a precipitate worth $8 per pound.

This precipitate was treated with sulphuric acid, roasted in an iron muffle and melted in pots. The zinc dust costs 5.5 to 6c. per pound at the works. No attempt was made to remove the zinc oxide with ammonia. The quantity of zinc used was governed by the value of the barren solution. Sometimes in increase of zinc made a better precipitation and at other times an increase in the strength of solution used on the ores attained the same result. A twenty-cent tail-solution was considered very high.

The precipitation vats were flat-bottomed, and this necessitated an occasional clean-up, which consisted of hard labor with hammer and chisel. This difficulty could be avoided by having conical-bottomed vats. About 225 tons were precipitated in 24 hours.

The presses would stand a pressure of 40 lb., although it was the custom to fill the presses with as low a pressure as possible, pumping the solution at a rate barely sufficient to keep up with the plant.

No naked flame should be allowed around the press while opening, and the cigarette smokers must be kept at a distance. A flame applied to the charge when the press is first opened will explode the hydrogen mixture, separating the frames and scattering mud.

Some years ago I was shown the plant at Mercur and no secret was made of the method of precipitation, and the same statement applies to the attitude of those in charge at Lead, South Dakota.

MARK R. LAMB.

Goldfield, Nevada, September 4.

ORE TREATMENT AT THE COMBINATION MINE, GOLDFIELD, NEVADA

By Francis L. Bosqui

(October 6 and 13, 1906)

I. METALLURGY

The Combination mine, situated about one-half mile northeast of the town of Goldfield, consists of ten full claims and three fractions, aggregating 200 acres. The original discovery on Combination ground was the first of any importance made in the now famous camp. The property was acquired from prospectors in 1903 by the representatives of two Eastern exploration companies, who, after a visit to one of the outlying districts, happened to be passing through the present site of Goldfield on the way to Tonopah; and thus, at the very outset of its career as a gold producer, the Combination was blessed by the happiest accident that can befall a mine—it passed into good hands. The property has since occupied a unique place among the mines of southern Nevada. It has been well administered; it has had the benefit of the most approved and practical methods in mining and metallurgy; and its development has not been hampered by stock manipulations. Consequently, though the most interesting property in the district, whether we consider its varied metallurgical problems, or its ratio of output to small mill-capacity and small development, it is the least advertised and the least discussed.

The first shipments from the Combination were made in December, 1903. The gross output of the mine from the commencement of operations to April 1, 1906, is as follows

		Value	Total Value
Shipping ore (tons)	1,773	$438.24	$776,992.84
Stamp bullion (oz.)	13,584	19.48	264,506.30
Concentrate (tons)	230	352.05	80,972.07
Cyanide precipitate (lb.)	734	45.77	33,594.18
Cyanide bullion (oz.)	1,401	16.44	72,346.51
			$1,228,411.90

The property was a shipper from the grass-roots. Almost any grade of ore could be segregated by rough screening and sorting, the practice being to reserve the milling ore ($25 to $100) in graded dumps until the completion of a mill. In shipping it was at first necessary to haul the ore to the railroad, a distance of 60 miles. The costs of transportation and smelter treatment were so high as to emphasize the importance of treatment on the ground, and an investigation was at once commenced with a view to installing a reduction plant

The ore has been described as a highly silicified dacite occurring in zones of fissuring in the decomposed dacite constituting the country rock. In the more shattered portions of the ore-body the dacite is almost entirely altered into quartz, with stringers and patches of kaolinized material. During the progress of the preliminary tests, the ore showed certain freakish variations which made it difficult to decide upon a method of treatment. The fineness of the gold, and the almost entire absence of concentratable material in the upper levels indicated dry-crushing, and a testing plant for dry-crusning was installed. But even after a long leaching with cyanide solution, it was still found possible to pan an appreciable quantity of gold from the residue. These tests had scarcely begun before the gold in the mine became coarser, and the proportion of sulphides increased. A series of tests by amalgamation, concentration, and cyanidation, gave decidedly promising results, showing an average saving of 45% by the first, 5% by the second, and about 40% by cyaniding the residual sand and slime—a total of about 90%. Later, it developed that certain portions of the oxidized ore carried a disquieting amount of acid—free sulphuric and ferrous sulphate, with here and there enough alum to make the rock astringent to the taste. In some of the tests the ore was so acid as to require 50 lb. of lime per ton as a neutralizer. But this very acid ore was not found to be in sufficient quantity to affect the general treatment seriously; and in subsequent milling tests an average sample of all the accessible oxidized ore was taken, and the acid condition met by using from 10 to 12 lb. lime per ton.

The results obtained in ore tests made at Goldfield in the early months of 1904 were confirmed during the summer by mill-runs made at an ore-testing plant in San Francisco. The representative test, which gave the best results, was made as follows:

1. Crushing through 40-mesh wire screen.
2. Plate amalgamation.

3. Concentration on a Frue vanner.
4. Hydraulic separation of slime in cone-classifiers.
5. Leaching sand with cyanide solution.
6. Agitating slime with cyanide solution.

The best conditions for the sand were found to be eight days' leaching with a 0.2% solution; while the slime required four hours with a 0.15% solution. The following is a record of extraction from slime:

	Gold. oz.
Assay heads	1.20
Assay after 1 hr. agitation	0.46
" " 2 " "	0.20
" " 3 " "	0.14
" " 4 " "	0.12
" " 5 " "	0.12
" " 6 " "	0.13
" " 7 " "	0.12
" " 8 " "	0.12

The amalgamation plate and zinc-box were cleaned up and the following results obtained from the whole test:

Indicated extraction by cyanide, 83.4%.

Actual extraction by cyanide, 77.9%.

Indicated total extraction by all processes, 93%.

Actual total extraction by all processes, 91%.

On a small scale, better results were obtained by crushing to 50-mesh, and it was found that by sliming the whole product, a still higher recovery might be made. But the improved efficiency of American tube-mills at the time the tests were made and the high cost of power and labor at Goldfield, left the advantage in favor of sliming too small to justify the experiment.

The mill was originally designed to treat oxidized ore only, although in places in the oxidized zone there were found small quantities of sulphide ore which resisted mill treatment by ordinary methods. But as there was no indication of the development of a large amount of sulphide ore at the time construction commenced on the mill, the installation was allowed to proceed. It was while the mill was being built that large shoots of sulphide ore were opened up as the limit of the oxidized zone was reached, and before the end of the year an extensive dump had accumulated with an average content of about 3 oz. gold. This was reserved for special treatment, and samples taken for investigation.

The sulphide in the 'sulphide ore' of the lower levels is a simple iron pyrite, for the most part finely disseminated. The following is an analysis of the ore:

Silica, 70.4%; alumina, 17.0%; sulphur, 4.2%; iron, 8.5%, and copper, trace.

Direct cyanide treatment was tried in all practicable variations, but without good results.

After grinding through 200 mesh and agitating 15 hr. in a 0.25% cyanide solution, the recovery was only 60%, with a 4-lb. cyanide consumption.

Roasting and leaching a 20-mesh product gave an extraction of 91%, with a 7.2-lb. cyanide consumption.

Roasting a 20-mesh product, re-grinding to 200-mesh, and cyaniding by agitation in a 0.25% solution gave 93% extraction.

Pan amalgamation of the roasted ore gave an extraction of 54 per cent.

Oil concentration (Elmore process) of raw ore, with agitation of tailing in cyanide solution, gave 90% extraction with a consumption of 7½ lb. cyanide per ton.

Oil concentration followed by cyanogen bromide treatment of the tailing gave an extraction of 96% from heads assaying 3.48 oz. gold.

Chlorination by leaching, using an aqueous solution of chlorine produced by bringing together an 0.8% sulphuric acid solution, and a 0.7% chloride of lime solution, and leaching 36 hr., gave a recovery of 40 per cent.

Chlorination by the barrel process, after four hours' treatment, yielded 78 per cent.

These various methods were tried before wet concentration because the aim was to treat all products on the ground and avoid shipping. A combination of concentrating and cyaniding, however, was ultimately considered the most suitable to Goldfield conditions, and was adopted.

The ordinary concentration of 30 or 40-mesh product was found ineffective. It did not make a close saving of the fine sulphide, which had to be removed on account of the poor cyanide recovery from the raw sulphide. It was necessary to evolve some closer method of recovery which would leave nothing for cyanide treatment except the finest particles that might elude the most efficient concentrating machinery. The only way to accomplish this was by a series of reductions, and by following each stage by appropriate concentration. The sulphide freed at each stage of grinding was at once removed before the ore passed to the next and finer stage of grinding, and thus an unnecessary comminution was avoided.

The following mill test forms the basis of the method adopted in practice. The ore, assaying 3.01 oz. gold, was crushed in stamps to 30 mesh and passed over a small Wilfley concentrator. This yielded 6% (by weight) of concentrate, assaying 27.4 oz. gold. The tailing was re-ground through 60 mesh, and re-concentrated, yielding 2.11% concentrate, assaying 19.8 oz. gold. The 60-mesh tailing was re-ground through 100 mesh and concentrated, yielding 0.4% concentrate, assaying 20.4 oz. gold. The residue was then ground to 200 mesh, and passed over a canvas table, yielding 10% of silicious concentrate, assaying 3.97 oz. gold. The final tailing from the above operation assayed 0.52 oz. gold, showing an extraction by concentration of 80%. The extraction by canvas alone was 13%, showing the marked adaptability of canvas to ore of this character containing so much extremely fine sulphide. Cyanide treatment of the slime resulting from this series of successive reductions and concentrations reduced the tailing to 0.22 oz. gold, making the total extraction 93 per cent.

This method was adopted because the required plant could be conveniently added to the mill already installed, and the same system of crushing used. Though a little complicated, the process was the least so of any of the methods considered. The high cost of fuel and supplies in Goldfield barred roasting. Besides, so long as the ore concentrated well, the advantage gained by segregating the roastable portion of the ore in small bulk was obvious.

The concentrate from the sulphide ore is now being shipped. It may later be treated by chlorination on the ground. The concentrate from the oxidized ore is about to be treated in the mill by fine grinding and prolonged cyanidation.

II. THE MILL

In considering a mill for the combination mine, it was thought that the size of the property would hardly justify the initial installation of more than 10 stamps. The plant, as completed, is unusually large for such small capacity. An extensive and costly equipment was made necessary by the elaborate process required for the best recovery, and by the decision of the management to reserve one-half of the mill for the treatment of custom ore. The latter required one complete and separate unit; thus the whole plant was spread over a greater area than would otherwise have been necessary. The outlay, however, has been amply justified by the efficiency of the machinery, and the high recovery obtained.

The lack of grade for millsite introduced several problems into the construction which have necessarily affected the cost of treatment, namely, the elevating of ore to the bins, and the elevating of pulp to classifiers. The water problem, however, was solved at the start. The water is obtained from springs situated about ten miles west of the mine, and is pumped to the mill against a head of about 800 ft. at a cost of 0.1c. per gal. It is hot as it comes to the surface and carries about 0.1% sodium sulphate and a trace of sodium chloride. Though slightly brackish, the water is potable; and for milling purposes the sodium sulphate is beneficial in assisting the settlement of the slime.

The ore is trammed from the shaft to storage bins, from which it is delivered to a platform above a 10 by 16 Sturtevant roll-jaw crusher, where it is mixed with the required amount of lime, varying from 5 to 10 lb. per ton. The crusher reduces the ore to about one-half inch size. The lack of fall made the interposition of a sorting grizzly impossible. Everything passes to the crusher and is delivered direct to a 12-in. belt-conveyor set at an angle of 20°, which elevates the ore to the mill-bins. There are four of these bins for the four units of five stamps each. (Ten more stamps have recently been added for treating sulphide ore.) The point of discharge at the top of the mill is shifted by means of the usual form of adjustable carriage. The ore is fed from hanging feeders, attached to the bins, into the low mortars of the Boss cantilever battery system. This type of battery frame and mortar was described in the MINING AND SCIENTIFIC PRESS, of February 17, 1906.

In the following description of the milling system, I shall first take up the treatment of the oxidized ore, which is carried on by the original ten stamps and cyanide annex. As already explained, the mill was divided into two separate units for the purpose of treating custom ore. Very little custom ore, however, has been received; and recent favorable developments in the mine have decided the management to devote the entire mill to their own ore. In the following account, a complete five-stamp system will be considered. The scheme of treatment is exhibited diagramatically in Fig. 12.

Inside amalgamation is carried on by means of a curved plate screwed to the chuck-block. The ore is crushed through

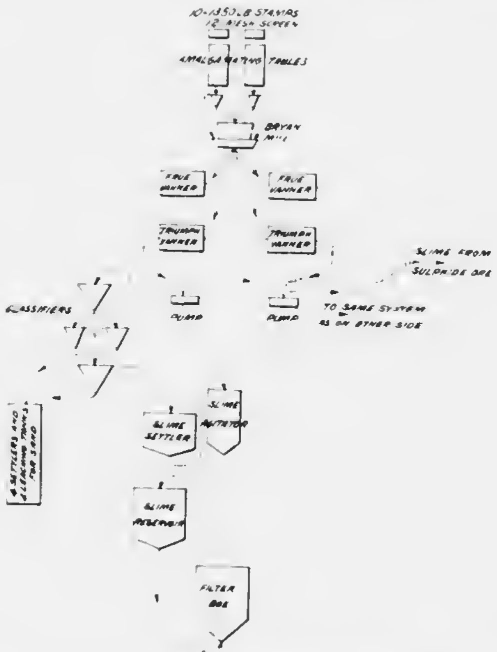


Fig. 12. Treatment of Oxidized Ore.

12-mesh wire-screen. Outside the screen a splash-plate is used. From this plate the pulp falls to a lip-plate about 12 in. wide, with the front edge slightly bent down, giving the pulp a gentle drop to the apron-plate. There are three plates to each mortar, arranged in steps, giving an amalgamating surface 53 in. wide and about 12½ ft. long. The whole tray, by means of wheels and track, can be shifted during the clean-up of the battery, as shown in Fig. 13 and 14.

At the bottom of each tray is a small cone hydraulic classifier, which separates the coarse mill-pulp into two products: (1) Fine sand and slime, which passes to the outer discharge lip of the Bryan mill and thence direct to the concentrators; (2) coarse sand, which passes to the Bryan mill for re-grinding.

The ore, being extremely hard and tough, is crushed with 1,350-lb. stamps, falling 100 times per minute, with a 6-in. drop. In spite of this, however, the stamp-duty is only 3½ tons, using a 12-mesh screen. One of the 5-ft. Bryan mills, running at half speed, takes all the coarse sand, from 10 stamps (approximately 20 tons per day), and crushes it through a No. 9 slotted screen, equivalent

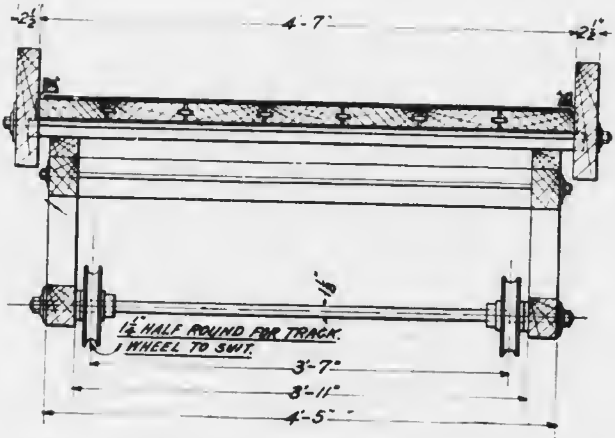


Fig. 13. Cross-Section of Amalgamation Table.

to 40 mesh. The final product from the Bryan is passed over two 6-ft. Frue vanners and two 6-ft. Triumph tables. From these concentrators the pulp is raised by two 54-in. Fremier sand-pumps to two sets of cone classifiers. This system is a modification of that introduced by Mr. Merrill at the Homestake cyanide plants. The top cone takes the intermittent discharge from the sand-pump and is so adjusted by valves that it sends a fairly uniform flow of pulp to the two smaller cones. The top cone is not a classifier. The first rough classification is made in the small hydraulic cones, from which a stream of sand flows direct to the sand vats. The overflow from these small cones, consisting of slime and fine sand, flows to the larger lower cone, where a closer clas-

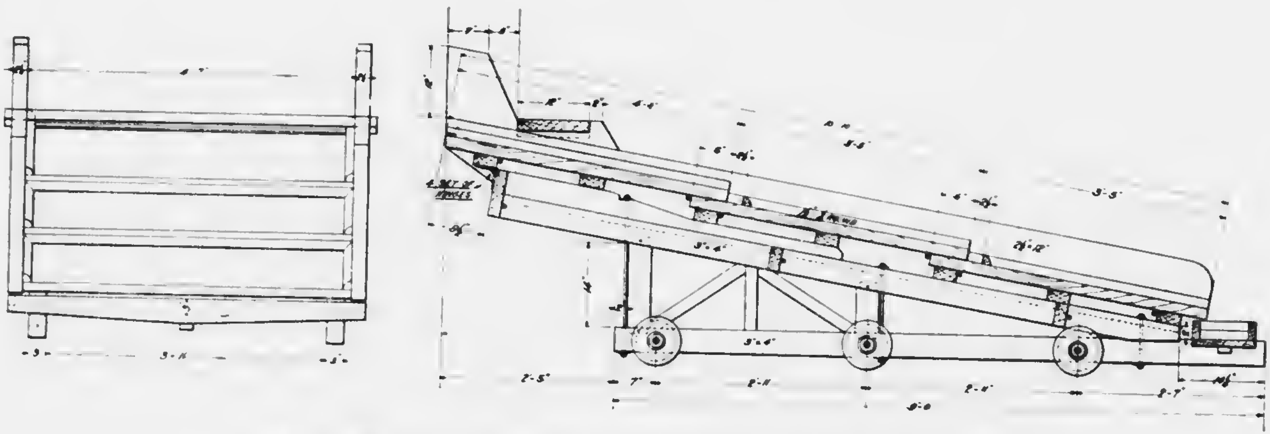


Fig. 14. Movable Amalgamation Table Used in the Combination Mill.

sification is made. The stream from the bottom of the latter also flows to the leaching vats. The overflow from this large lower cone passes to the slime settlers.

The pulp from the Bryan mill may, therefore, be said to consist of two products, namely:

1. Sand	Retained on 100 mesh	41%	67% of total.
	Passing 100, retained on 200	35%	
	Passing 200 mesh	24%	
2. Slime	Passing 200 mesh	90%	33% of total.

This is not considered an ideal separation, inasmuch as the slime carries a large amount of fine sand. In spite of this, however, the recovery from the slime has been very good (over 95%) since the introduction of the Butters-Cassel filter. A contemplated re-arrangement of the cones is expected to improve the extraction from slime.

In a small mill, classification requires constant attention. Slight interruptions, the suspension of one battery unit, or any variation from normal operating conditions, at once affect the flow of pulp in the classifiers, which are dependent upon nice adjustment for their efficiency. Obviously, the larger the mill, the smoother will be the operation of this system, and the less attention will it require.

From the cones, the sand flows to a pipe distributer and thence to a settling vat, of which there are four on each side of the mill. The fourth vat was added after the completion of the mill. These vats were at first provided with an overflow lip and a circular launder to carry off the surplus water. It was found, however, that occasional irregularities in classification resulted in the settlement of slime in these vats, which interfered with percolation. The settlers were then fitted with slime-gates, and the overflow from the sand-settlers, carrying a certain amount of slime, now runs to a centrifugal pump at the lower end of the mill, to be sent to the slime-settlers for clarification. This is an awkward arrangement, but was unavoidable owing to the small gradient of the millsite.

When the sand-settler is filled, the surplus moisture is removed by a Gould vacuum pump, and the charge shoveled to the treatment vat below. Here the charge is given an eight days' treatment with a 0.1% and a 0.2% cyanide solution, and the residue discharged by sluicing through a central bottom-discharge door.

The slime is delivered to the centre of a conical-bottom settler, provided with a rim overflow. There are two of these settlers on each side of the mill. Each is alternately allowed to fill and overflow for 12 hours, and allowed 12 hours for settling. A pipe decanter carries off the surplus water, leaving the slime with about 50% moisture. Sufficient strong cyanide solution is added to the charge of slime to make a solution of from 0.15 to 0.2% cyanide, and to give the pulp a consistence of three parts solution to one of slime. By means of a centrifugal pump, the pulp is transferred to an agitatior, with a steep cone-bottom. By means of valves, the same machine (a 3-in. Krogh slime-pump) is applied to the agitation, taking the slime from the bottom of the vat and throwing it back at the top. A supplementary agitation is given by means of a mechanical stirrer revolving slowly. The pulp and stirrer give an ideal agitation, being sufficiently complete for the best results. The pump, which has a lift of about 10. ft., is run at 375 rev. per min. Thus, at short intervals, the whole content of the vat passes through the pump, where it is aerated by means of a pet-cock on the suction pipe. The stirrer prevents the settlement of fine sand at the junction of the cone bottom with the staves, and keeps up the agitation during pump repairs. The only repair required in the pump is the replacement of the shaft about once per month. The objection urged against the centrifugal pump as an agitator—that it is apt to clog with slime during stoppages—is not a sound one. The 'slime' product at the Combination mill carries a large amount of fine sand, and the pump has repeatedly been started up after stoppages of several hours, without the least trouble.

After agitation, lasting from 12 to 18 hours, the pulp is discharged into a slime reservoir, a vat of large capacity provided with a mechanical stirrer from which it is drawn, as required, for filtration in the Butters-Cassel filter.

A filter-press was at first used for filtering the slime. This was an American machine consisting of fifty 42-in. plates and 2-in. distance frames, and had a capacity of 2½ tons of (dry) slime. It was evidently made of poor material, as it was the source of exasperating trouble through breakage. The plates were continually cracking under a pressure much below the guaranteed maximum, and the outlet-cocks getting out of order or breaking. Moreover, the operation of pressing, washing, and discharging was extremely slow, requiring about five hours, and the operating expense high, requiring two men on each of the three shifts at the prevailing laborer's wage of $4 per day, to say nothing of the cost of the filter-cloths. It is fair to suppose that one of the high-class foreign presses of the Dehné type might have given better satisfaction. At best, however, filter-pressing is not to be compared with the system now in use, especially as regards cost of operating, and the completeness of the washing operation.

The essential points of difference between the Butters-modification of the Cassel filter and the other vacuum-filtering schemes are the extreme simplicity in the design of the filter-leaf or frame, and the fact that these frames, throughout the operation, are always stationary. In the Combination plant there are 28 frames (5 by 10 ft.) set 4½ in. apart in a box 10 ft. square with a pointed bottom inclined at an angle of 50°. The slime-pulp is introduced at the point of the box, and a vacuum of 22 in. of mercury is maintained for about 20 min., during which time a cake is deposited on each side of the frame ¾ to 1 in. thick. The surplus pulp is then withdrawn to the slime reservoir and the wash introduced, consisting of a weak solution of cyanide. When the cakes are thoroughly washed, the weak solution is withdrawn into its proper vat, and water introduced, until the frames are completely immersed. The object of this final water is to assist in removing the cakes. More water is introduced into the interior of the frames under a low head. This causes the cakes to drop off clean, into the pointed bottom of the filter-box, whence they are finally removed by sluicing. The whole operation requires about 3½ hours, and about nine tons (dry) slime are treated at each charge. The plant, therefore, has a capacity of about 63 tons per day. It is operated by one man on each shift. The principal power required is for pumping the pulp and the various solutions in and out of the filter-box, and for operating the vacuum pump. In addition, the gold-bearing solution discharged from the vacuum pump is raised about 30 ft. and forced through the discarded filter-press now used for clarifying purposes. The whole consumes about 10 h.p. A 15-h.p. motor has been installed for this work, but has extra work to do not connected immediately with filtering.

The filter-plant has required no repairs since it was first operated in February of this year, and has worked in the most satisfactory manner. The cost of filtering (exclusive of power) has been reduced from 96c. per ton of slime in January (using filter-press) to 26c. in May (using the Butters system). The power consumption appears to be about the same in the two processes.

Treatment of Sulphide Ore.—This ore is crushed to 12 mesh, in two five-stamp batteries, and run over plates to take up the small quantity of free gold present. See Fig. 15. A classifier at the end of the plate-tray removes the coarse sand and sulphide, which go to a Wilfley concentrator; the overflow of slime and fine sand passing to the outer lip of a Bryan mill. One Wilfley table, therefore, removes the coarse sulphide from the product of ten stamps crushing to 12 mesh. The tailing from the Wilfley is raised in a bucket-elevator to a Bryan mill, which crushes it to 40 mesh (No. 9 slotted screen). The tailing from the Bryan

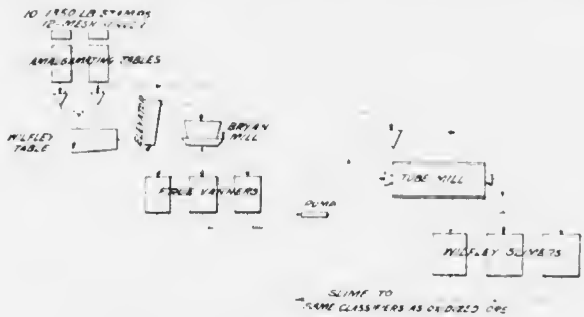


Fig. 15. Treatment of Sulphide Ore.

mill joins the stream of slime from the small classifier and passes to four 6-ft. Frue vanners. Here a large quantity of fine sulphide is removed. The Frue tailing is then elevated by a sand-pump to a classifier above a 4 by 12 ft. Abbé tube-mill of the trunnion type. The cone acts as a classifier as well as a de-watering device. The coarse sand passes to the tube-mill; the slime overflow joins the tailing from the tube-mill and goes to three Wilfley slime-tables of the latest pattern. The tailing from the last, consisting of slime and fine sand, is elevated to the cone-classifier in the original mill, where it is mixed with the oxidized tailing and treated in the cyanide plant.

This plant was only operated a few days, and then shut down, pending the installation of two Wilfley slime-tables, making three in all, the first having been set up experimentally. During this short run the results were very promising. Of the final product from the slime-table, 87% passed through 200 mesh, and the three stages of reduction showed a saving by concentration of over 80% of the contained gold. With the cyanide treatment of the slime tailing, a confirmation of the small mill-run is expected, namely, better than 90% recovery.

It is too soon to give the results of tube-mill work. The mill is lined with 2½-in. silex blocks, which will be replaced by blocks 4 in. thick.

Fig. 16 Tube-Mill and Wilfley Slime-Tables.

Precipitation is accomplished in the usual way by means of zinc shaving. The solutions are richer than are usually seen in cyanide mills, reaching as high as one ounce per ton in gold. Owing to the absence of silver, which undoubtedly facilitates the precipitation of gold, a very large zinc surface is required. The precipitate is refined with sulphuric acid and smelted in a pot-furnace, with gasoline, a powerful jet being maintained by means of a small upright Leyner compressor.

The following details of costs for milling and cyaniding indicate a few of the difficulties met with in an isolated district where the cost of supplies, and of labor, power, and water are unusually high. And then it must be borne in mind that the mill is of small capacity. The change from steam to electric power—the latter being furnished by a local concern operating a 90-mile transmission line—reduced the cost of power from a maximum

Fig 17 Slime-Agitation Vat.

of $1.73 per ton (January 1906) to $0.76 (May, 1906). We have already noted the great reduction in cost effected by introducing a new filtering system for slime. The marked decrease in operating expense during February was due to increasing the capacity of the mill by annexing 10 stamps, making 20 in all. While the total milling and cyaniding cost of $5.828 per ton seems high, it is really not so when local conditions are considered. It is expected that further retrenchment will soon cut this down to $5 per ton.

The recovery in the mill and cyanide plant has attained a maximum of 93% for several consecutive months, and has averaged over 90 per cent.

CYANIDING COST PER TON OF ORE MILLED (1906).

Month.	Ore tons.	LABOR.						SUPPLIES.								Total cost.
		Solution men.	Filter-press.	Shovelers.	Tailing dam.	Batters slime-filter.	Total labor.	Cyanide.	Zinc.	Acid.	Lime.	Filter cl[illegible]s.	Other chemicals.	Miscellaneous.	Total supplies.	
January	791 6	0 581	0 952	0 522			$2 055	0 362	0 121	0 031	0 155	0 016			0 685	$2 740
February	1,049 0	0 400	0 351	0 357	0 044	0 149	1 301	0 349	0 073	0 020	0 155	0 023	0 027	0 008	0 654	1 955
March	1,617 0	0 288	....	0 426	0 068	0 230	1 012	0 181	0 052	0 017	0 115		0 026	0 023	0 412	1 424
April	1,606 5	0 280	....	0 449	0 060	0 255	1 044	0 311	0 058	0 033	0 157			0 017	0 576	1 620
May	1,842 5	0 254	....	0 252	0 063	0 269	0 838	0 384	0 153	0 013	0 311		0 005	0 004	0 870	1 700

NOTE—The average consumption of chemicals in treatment of sand and slime during 1906 has been as follows: Cyanide, 1 27 lb.; zinc 0 6 lb.; and lime, 11 pounds.

MILLING AND CYANIDING COSTS PER TON OF ORE MILLED (1906).

Date.	Ore tons.	Mill superintendent.	Power.	Heating.	Water and pumping.	Crushing and elevating.	Stamping and amalgamating.		Concentrating.	Cyaniding.		Assaying and sampling.		Lubricating.	Tools.	Lighting.	Maintenance and repairs.		General expense.	Total per ton.		Total costs.
							Labor.	Supplies.		Labor.	Supplies.	Labor.	Supplies.				Labor.	Supplies.		Labor.	Supplies.	
January	791 6	0 303	1 734	0 795	1 263	0 303	0 592	0 241	0 597	2 055	0 685	0 378	0 003	0 097	0 090	0 009	0 442	0 679	0 002	$4 670	$5 598	$10 268
February	1,049 0	0 214	1 193	0 194	0 953	0 221	0 401	0 345	0 409	1 301	0 655	0 287	0 009	0 035	0 035	0 010	0 278	0 275	0 002	3 111	3 706	6 817
March	1,617 0	0 151	1 311	....	0 618	0 294	0 335	0 227	0 310	1 012	0 412	0 268	0 005	0 035	0 012	0 047	0 849	0 998	0 136	3 385	3 536	6 921
April	1,606 5	0 125	0 875	.....	0 622	0 291	0 377	0 139	0 317	1 044	0 576	0 332	0 012	0 040	0 020	0 027	0 753	1 520	0 152	3 351	3 871	7 222
May	1,842 5	0 130	0 763	.....	0 271	0.271	0 327	0 399	0 283	0 858	0 870	0 312	0 004	0 026	0 036	...	0 782	0 430	0 068	3 022	2 806	5 828

FILTER MACHINES

(October 13, 1906)

The Editor:

Sir—In view of the scarcity of literature touching the Moore and Butters processes, some comment on Mr. R. Gilman Brown's recent papers may be of interest.

The process at Bodie differs from that at Telluride (as described in *The Engineering and Mining Journal*, July 7, 1906) in a notable point, the cake being blown off by air into a hopper instead of by water into the wash-vat. The latter method is distinctly advantageous as the strain on the filters is considerable, while air quickly enlarges leaks. Further, a stoppage of a few minutes caused, for example, by a circuit-breaker being thrown out, will cause a considerable portion of the cake to drop into the wash-vat, which is both inconvenient and expensive, unless the vat is designed for pulp discharge. The photograph given by Mr. Brown shows that the baskets are likely to hang so unevenly that to use water, the basket being hung up, would consume much time or give a faulty discharge at the high end. The discharge of cake by water when the basket is immersed, is perfect, while discharge by air entails scraping at the edges. A great disadvantage of moving baskets is that an accident to the somewhat complex moving gear puts more than one basket out of commission, while, if the pulleys of the hoisting chains wear far, there is a chance of an awkward accident through a chain slipping one or more links. The ability to raise the baskets, however, facilitates replacing leaky filters. In the Butters process it is to be expected that the unmoved filters will outlast those raised and air-distended; but besides this we have an economy in labor. To raise and discharge a basket takes half an hour and the basket-man's whole attention, while in the Butters process his duty is mainly to turn the valves. With the charging and washing intervals given by Mr. Brown, two baskets would keep a man busy, while he would be badly overworked with quick charges like those obtained by Mr. Lamb on Goldfield ore.

H. W. GARTRELL.

Bisbee, Arizona, September 7.

WHAT IS SLIME?

(October 20, 1906)

The Editor:

Sir—Slime is now receiving so much attention and is so conspicuous a feature in ore treatment that there seems to be a growing disposition to designate some measure of fineness which shall define it and receive general recognition.

We know slime now as a condition of pulp that requires attention more or less different from that given to sand. In my own experience I have found that slime, as it appears to us, is not altogether a matter of extreme fineness, but the manner of creating it is also a factor. Aeration in process of crushing greatly increases the visual evidence of slime.

Many years ago I had been crushing an ore with stamps, using but little water, and the pulp went largely to slime, although a medium screen was used. Near-by a large slow-traveling Chilean mill was installed and began crushing on the same ore. Instead of screens they used float overflow, about 18 in. above the die. To my surprise, there was no evidence of slime in the vat which received the pulp, the water flowing away from it clear. In this case there was no possible contact of air with the pulp at, or after, its crushing, as the stream had but a gentle flow (without falling) from the mill to the tank. For concentration, it is plain that aeration of pulp is not desirable.

M. P. Boss.

San Francisco, October 5.

THE ACTION OF OXYGEN IN CYANIDE SOLUTIONS

By H. Julian

(October 27, 1906)

*A doubt has for some time existed as to the accuracy of the generally accepted idea that free oxygen is primarily essential for the dissolution of gold in cyanide solutions, accord[illegible], to the equation:

$$4KCy + 2Au + O + H_2O = 2KAuCy_2 + 2KOH.$$

Experiments are described which go to show (1) that free oxygen plays no primary part in the reaction, (2) that any assistance given by free oxygen is of a secondary nature, and (3) that free oxygen exerts a retarding influence.

The experiments show that the galvanometer points to the presence of free oxygen as having a retarding influence on the dissolution of the gold, whereas the balance points to it being of material assistance. The cause of the two instruments not agreeing is discussed, and is attributed to the formation of local voltaic circuits. These, in the first instance, deposit hydrogen and oxygen, which it may be assumed, become occluded at their respective electrodes until the systems are in equilibrium. It is pointed out that the cyanogen leaves the solution to combine with the gold rather than that gold particles pass into the solution, and it is shown that cyanogen does not leave the solution until the deposited oxygen has been occluded to a certain degree of concentration. The reason for this is that the expenditure of energy necessary to remove oxygen from the solution is less than that necessary to remove cyanogen; but when oxygen is occluded to a certain concentration, the expenditure of energy then necessary to cause the metal to occlude a further amount becomes as great as that necessary to begin to remove cyanogen from the solution. The available energy is obtained from the metal and solution, and it follows that when the solution is very dilute the available energy is too small to remove cyanogen, oxygen being then alone

*Abstract from *British Association Report*, prepared by the author and published in the *Journal* of the Chemical, Metallurgical and Mining Society of South Africa.

deposited. From this it may be conjectured that no metal actually combines with cyanogen until the solution has a certain minimum strength.

The presence of dissolved oxygen in the solution has a secondary effect in the process of dissolution, by oxidizing the occluded hydrogen produced through the action in the local voltaic circuits. This results in upsetting the equilibrium, and introducing into the circuits concentration gas cells, which soon bring about equilibrium again, but this time with oxygen at both electrodes at different concentrations, instead of hydrogen and oxygen. If, now, excess of dissolved oxygen diffuses to either of the electrodes the equilibrium is again upset, and an E.M.F. is generated by the gas cell in opposition to the E.M.F. generated by the metal couple; the net result being, of course, a current in the direction of the greater E.M.F. As the strength of the solution increases after a certain point, the E.M.F. due to the metal couple increases rapidly, whereas that due to the oxygen-concentration cell remains constant or increases only slowly.

The increase in the E.M.F. of the metal couple appears to be largely due to the formation of AuCy, a compound having a high potential, which acts as an electrode. This deposits in films, varying in density or thickness to a maximum with the strength of the solution. A couple results of Au-AuCN. After this stage of the process, when AuCy is formed, oxygen ceases to exert an influence. That is to say, the metal passes into solution by the AuCy dissolving in the potassium cyanide solution, as one salt dissolves in the solution of another.

The effect of the gas cell is best observed in highly dilute solutions at ordinary or low temperatures. After a certain strength is attained, dependent on temperature, the effect of the gas cell is entirely masked. At the higher temperatures the E.M.F. of the gas cell diminishes, with a corresponding increase in the E.M.F. of the Au-AuCy couple. At boiling point the retarding oxygen effect of the gas cell on the dissolution of the metal practically disappears.

SOME TAILING SAMPLERS

By R. Gilman Brown

(November 3, 1906)

It is hardly too much to say that of tailing samplers there should be no end, and the description of good devices of this class at least serves the laudable end of removing the underpinning

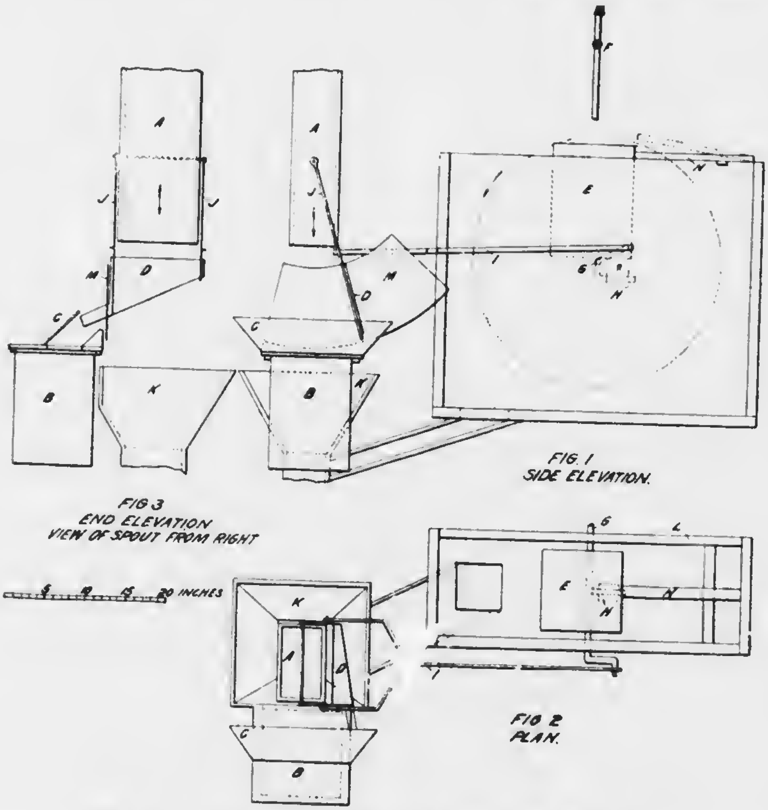


Fig 18. Tailing Sampler in Use at the Ymir Mill, B. C.

from those who still evade the use of such devices. It can not but be remarkable that, even at this late day when so much has been written of the need of a systematic check on milling work, it should be rather the exception to find automatic samplers installed in smaller mills.

It is not the every-day mechanic who can devise a machine which at regular intervals can be self-instigated to take a quick cut through a stream of tailing, and many of the devices furnished by manufacturers are open to objection on the score of high cost, limited adaptability, and complicated mechanism. The two illustrated herewith are practical machines, which have been installed by home talent and have stood the test of long operation.

The first, illustrated in Fig. 1, 2, and 3, samples the tailing from the Ymir mill, in British Columbia. It can be described as follows: *A* is a vertical chute down which the tailing flows, *D* is a sampling box with 1/16-in. slot suspended by the hangers *J J*, which are free to swing. From the spout of *D* the sample runs into the receptacle *B*; the apron *M* guards against splash, and *C*, with its inclined cover, further protects the sample from accidental salting. When the sampler is in the position shown, the tailing-stream falls into the hopper *K*. The actuating device is shown at the right of Fig. 1. *F* is a water pipe with a regulating valve, *E* is a water can secured eccentrically on a crank-shaft *G* and held vertical by an adjustable weight *H*. The crank, through its connecting rod and fork *I*, causes *D* to swing back and forth through the stream of tailing.

The operation is as follows: When the water dripping from *F* has reached the proper point, it overbalances the counterweight *H*, and *E* turns down toward the left, spills its water into the housing *L*, and is carried around by its inertia to its original position, raising the latch *N* as it passes and then falling back against it. By adjustment of *H*, this can be made to happen with little jar. It is easy to regulate this device to take samples from five-minute intervals up to 30. *B* and *E* are the ordinary 5-gal. coal-oil cans of commerce. It should be noted that in Fig. 3 the actuating device has been omitted, and in Fig. 2, the apron *M*. The other device is installed at the cyanide plant of the same company. Here *A* (Fig. 4, 5, 6.) is the tailing chute and the sampling box *B* is suspended in the same way as before, with the exception that the hanger is bent from a single piece of iron; this construction is demanded because the connecting rod *I* works only on one side. The actuating device is a flutter wheel G in the tailing-launder driving the grooved wheel E through the medium of a crossed cord. Hanging loose of the shaft of E is a heavy arm of iron C, connected by I to the sampling-box. The pin D carries C around with E until it reaches the position shown in Fig. 4. From this position C falls by its own weight to its lower position, swinging the sample-box through the stream. In the lower position it remains until D catches up with it, when it is carried around again to the upper position. During this movement it swings B back still further out of the stream. This is a very pretty mechanical arrangement and works admirably.

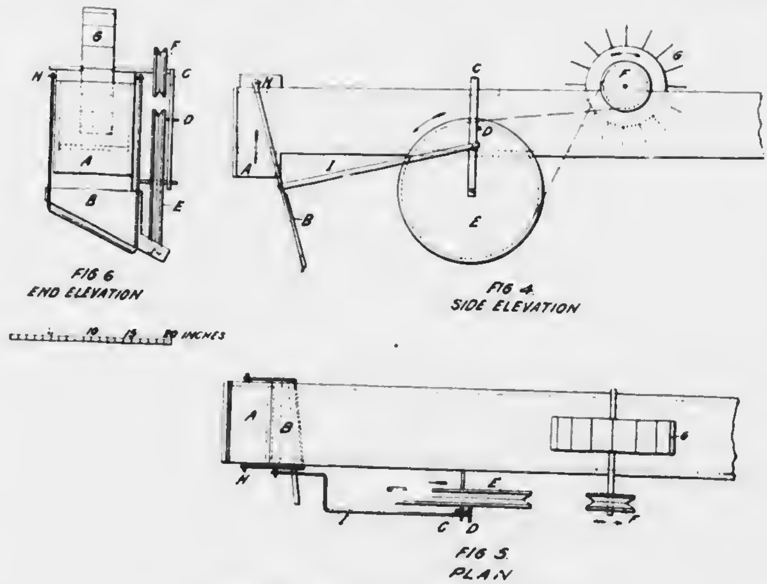


Fig. 19. Another Sampling Device.

The use of the flutter wheel is, however, open to objection in that the time interval at which the sample is taken varies somewhat with the quantity of pulp passing. Assume that half the mill is hung up; if under this condition the wheel G revolves half as fast, the sample interval will be doubled, and as there is but half as much tailing dropping from A, the amount of sample taken for this period will be one-quarter of normal, when to give true results it should be one-half. With half the flow, no doubt the wheel G would not slow down to the same extent, but it would slow down slightly, and observation of the sampler in operation shows this to be the case. Moreover under the best of conditions the speed of *G* is not uniform, so that a better arrangement would be to have *E* actuated through the medium of some reduction gearing from the mill-shafting or from a water-wheel driven by a stream that can be regulated.

COPPER IN CYANIDE SOLUTIONS

(November 3, 1906)

The Editor:

Sir—I note what C. A. Arents says about copper precipitating on zinc shaving, and, having had a similar experience, I can agree with him that copper may not be a hindrance to good precipitation.

Two years ago I was operating a cyanide plant in the desert country, treating a dun p of old pan-mill tailing. One mill had burned down there, and there were numerous small pieces of burned copper plate mixed with the sand. The copper did not begin to show until the plant had been running about four months. The concentration of copper in the solution was probably due to the fact that no wash-water or weak solution left the plant except that existing in the residues as moisture.

Copper began to show first on the fresh zinc that was put in the lower boxes and in a month or six weeks it had worked up to the head boxes. I was somewhat alarmed at first, and tried to keep it off by adding cyanide to the filter-box. The stronger solution would take it off in a few hours but only to be back again the next morning. I took daily samples (for a while) of the solution leaving the boxes, but they only assayed a trace to a few cents, when the solution entering averaged about five dollars per ton.

The gold seemed to precipitate on the copper-covered shaving so that there were shades from bright copper, bronze, to black. There was also mercury in the sand which was taken up by the solution and precipitated, causing some of the shaving to look like silver.

The value of the precipitate shipped was about $16 per lb. before the copper began to show, and gradually it decreased to about $8 per lb. But with the exception of making a lower grade precipitate, the copper did no harm whatever, and the precipitation was as good after as before.

E. D. Chandler.

Rochford, S. D., October 17.

TUBE-MILL LINING

(November 3, 1906)

The Editor:

Sir—In your issue of July 28, 1906, I read with pleasure the article on 'Tube-Mill Lining,' in which the Barry honeycomb is described and illustrated.

There is no doubt that this patent is a great improvement on the silex lining which the Waihi Co. imported with their tube-

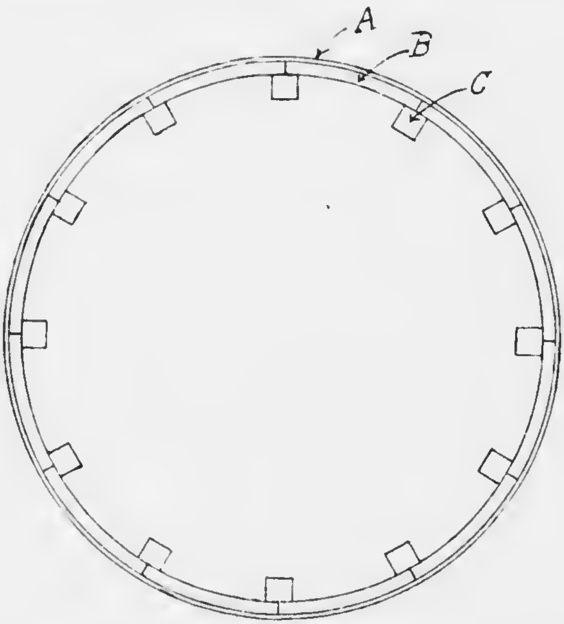


Fig. 20 A Tube-Mill.

mills. Anyone who has had experience with the silex lining knows how expensive and troublesome it is when used for wet grinding, and consequently will welcome any improvement in the matter of initial cost and maintenance. The ordinary iron or steel liner, with smooth inner surface, is also unsatisfactory, for, owing to the stones sliding and thereby becoming flat, much of their efficiency for grinding is lost.

Having run tube-mills for some considerable time, grinding the sandy portion of the pulp from stamps, crushing hard ore through all grades of wire-screen from 30 mesh down to 10 mesh, and having had trouble, owing to the use of smooth iron liners, I set to work and brought out an improved iron liner, which consists of segments of a special hard mixture of iron, costing 14s. per cwt. (say, three cents per pound), when manufactured into liners. These liners are 4 ft. long and 1⅛ in. thick, and are of such a width that 12 form a circle of 4 ft. diam., which is the diameter of the tube-mill. Instead of having holes cast for bolts, they have only two half holes on each long edge. Over the junction of each pair, a cast-iron bar 2½ in. square is placed and bolted by two square-headed bolts through the lining and outer shell of the tube-mill. Thus there are 12 cast-iron bars which run longitudinally through the mill, and the effect is to form a casing of flint stones which does not slip and which protects the iron lining.

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

The cost of liners and bars for a 4 by 16 ft. tube-mill is £84, and two men can fix them all in position in three shifts of eight hours each; the total cost with bolts would be, say, £90. The amount of sand passing through the tube-mill is 70 tons per day of the material discharged through the 10-mesh screen of the stamp-battery, and the finished product has the fineness that would be secured by the use of a 35-mesh screen on the battery. Thus the cost of lining is about seven-tenths of a penny, or 1.4 c. per ton of sand ground.

S. D. McMiken.

Komata, New Zealand, September 2.

COMPARATIVE TESTS BETWEEN COKE AND CRUDE OIL FOR MELTING PRECIPITATE

(November 24, 1906)

The Editor:

Sir—It may interest your readers to know that two experiments were made at the Butters Salvador mines, in Salvador, to determine the relative efficiency of coke and crude oil in melting precipitate from the cyanide plant; the first was on the ordinary San Sebastian clean-up from the acid refining box, and the other was on silver precipitate from Divisadero. The costs are calculated on a basis of coke at $0.0208 (gold) per lb., and oil at $0.252 (gold) per gal. laid down at San Sebastian; white labor at $3 per shift at 10 hours; and compressed air at 1c. per hour.

FIRST TEST

OIL.

	Sept. 11, 1906.
Precipitate	2,448 oz. Troy
Oil	15.2 gal.
Time	7½ hr.
Cost of fuel per oz. of precipitate	$0.00156
Cost of fuel per 1,000 oz. of precipitate	1.56
Cost of white labor per 1,000 oz. of precipitate	0.91
Cost of air per 1,000 oz. of precipitate	0.03
Total	$2.50
Balance in favor of oil per 1,000 oz. of precipitate	1.06
Total	$3.56

COKE.

	Sept. 13, 1906.
Precipitate	1,991 oz. Troy
Coke	221 lb.
Time	8⅓ hr.
Cost of fuel per oz. of precipitate	$0.00231
Cost of fuel per 1,000 oz. of precipitate	2.31
Cost of white labor per 1,000 oz. of precipitate	1.25
Total	$3.56

SECOND TEST

OIL.

Sept. 20, 1906.

Precipitate	5,664 oz. Troy
Oil	21 gal.
Time	10½ hr.
Cost of fuel per oz. of precipitate	$0.00093
Cost of fuel per 1,000 oz. of precipitate	0.93
Cost of white labor per 1,000 oz. of precipitate	0.55
Cost of air per 1,000 oz. of precipitate	0.02
Total	$1.50
Balance in favor of oil per 1,000 oz. of precipitate	1.03
Total	$2.53

COKE.

Sept. 21, 1906.

Precipitate	3,524 oz. Troy
Coke	283 lb.
Time	10⅛ hr.
Cost of fuel per oz. of precipitate	$0.00167
Cost of fuel per 1,000 oz. of precipitate	1.67
Cost of white labor per 1,000 oz. of precipitate	0.86
Total	$2.53

Thus, in both these tests, on entirely different classes of material, we get a uniform result, which shows strongly in favor of the crude-oil method of melting, at least as far as the Republic of Salvador is concerned. There are other advantages connected with the use of oil which do not appear above, but which should not be overlooked. Firstly, the elimination of a by-product in the shape of coke ashes from the wind furnace; and secondly, a great saving in labor and personal discomfort to the man who does the melting, which is a point of some importance in a tropical climate.

E. M. HAMILTON.

San Sebastian, Salvador, October 22.

TUBE-MILL LINING

(November 17, 1906)

The Editor:

Sir—After reading about the proposed tube-mill lining to be used at the Waihi mine, in New Zealand, I thought it would be of interest to your readers to know of my experience with practically the same lining. We have installed a 12-ft. mill, made in the shops of this company (Oriental Con. Mining Co., of Korea), and, being slightly out of the world, it would have been

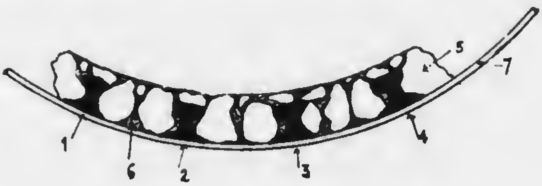


Fig 21 1, 2, 3, 4. Rails. 5 Quartz. 6 Mortar. 7. Shell

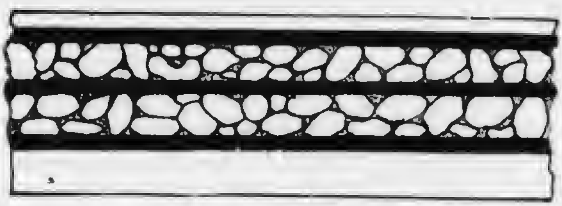


Fig. 22. Tube-Mill Lining.

a great saving to line our mill with ordinary hard quartz and cement such as described in the MINING AND SCIENTIFIC PRESS of July 28, 1906.

About nine months ago it occurred to me that if we could utilize our hard quartz, together with a proper cementing material, it would make an ideal lining for a tube-mill. The most important point was how to hold it in place within the tube. The mill was put together in three sections or sheets of iron and all well riveted hot. These sections we well knew would not be as good as one continuous sheet or tube, for they were liable to spring a leak around the rivets. This would mean a considerable loss in our case, as we are grinding concentrate to a slime in strong cyanide solution. Therefore an arrangement that would give strength to the mill as well as hold in place the 'bull' quartz lining was wanted. Sixteen-pound rails were riveted lengthwise in the tube, six inches apart and around the total inner circumference. Diameter of tube is three feet. Fig. 21 and 22 show the arrangement of rails and the way the quartz and cement was held in place.

The best quartz for the purpose was found at the Tabowie mine. It was hard but not brittle, which seemed to be an important point in such a lining. All cementing material was made up of equal parts of hard coarse quartz, sand, and cement. Special care was taken to have the sand clean and fresh. A layer of cementing material was first put between the rails next to the shell, and spread out evenly. Next, large pieces of 'bull' quartz, 2 and 3 in. diam. were pressed into the cement as compactly as possible, and between these larger pieces smaller quartz was filled in with the cementing material. It was all tamped well, so as to fill the spaces with cement. Another layer of cement was spread evenly over the top and another layer filled in as before. The two layers brought the quartz about two inches above the rails, the upper layer serving as a protection. Every little space between the large quartz was filled with quartz and cementing material. Finally, on top of all was placed the mortar; this also was well tamped. The surface was smoothed with a trowel and then allowed to set. A strip or one-third of the mill was lined at one time. After two days it became sufficiently hard so that the mill could be turned over and another one-third lined in the same way. When the lining was complete, it was allowed to set for six weeks. This was more than was really necessary to become hard, but we wanted to have it thoroughly dried out, so as to have it in the best possible condition to wear. The inner diameter of tube was now about 2 ft. 3 in. Ordinary bull quartz, the same as used for the lining, was broken up in 2 and 3-in. pieces and used as pebbles for grinding, which proved to be very satisfactory. The tube was filled just a little over half full, which amounted to 1½ to 2 tons of pebbles. The material ground in the mill was concentrate, cyanide solution being used.

Now, in regard to the wearing of the lining, I can say that it was by no means satisfactory. At the rate it was wearing it would have only taken about two weeks to cut away all the lining. It was rapidly undermining the larger pieces of quartz—that is, the cementing material was gradually cut out. Wherever there was a space of cement between the quartz that was possible to wear, it was sure to be gradually worn away and cut out, allowing the quartz pieces to fall out. You cannot imagine the beating and wearing action the pebbles have on that cementing material. The quartz in the lining will stand alright, but the thing is to keep it in place. Even with silex linings, where the best quality of diamond cement is used, and where all the bricks are made to fit as closely as possible, the pebble will, in spite of all, wear away some of the cement to a certain depth. Now, how can you expect ordinary or portland cement to stand where the spaces between rough pieces of quartz are greater? It may be easy to think in the first place that this lining is going to hold, but to make it is another thing. The lining appeared to be very hard before using and certainly ought to have been, for it had plenty of time to set. The principle of the lining is all right, but I am afraid we will have to find something a good deal harder than portland cement for a cementing material.

Cast-steel liners were the next to be tried, as we had on hand enough of stamp-battery liners (4 ft. by 9 in. and 1 in. thick) for our purpose. Before putting them in place the old quartz lining had to be removed from between the rails by means of a single jack and moil. It was no easy task, for it was hard and the rails held it in securely. The rails were left in and the steel plates put on top, being held in place by bolts with counter-sunk heads. The bolt extended out through the shell, and over this was put a washer made from a piece of old vanner belt. On top of this was an iron washer and then the nut. There were two bolts, one in each end of the plate, to hold it in place, and both were screwed down tightly. After a day or so of running, they were screwed up again. Probably it was necessary to go all over the bolts on the tube-mill a third time, but after this we never had any trouble with bolts leaking or getting loose. It was absolutely necessary to have vanner-belt washers next to the shell. Fig. 23 will show the method of holding the liners.

The rails give the tube great strength, especially where it is put together in three sections. With these in place, there need be no fear of leaks starting around rivets, especially if the riveting has been done well.

These steel liners have now been in use one month and show only a little wear. At the present rate of wear these liners should last at least six or seven months more. About ten tons of clean concentrate per day pass through our 12-ft. mill. Two tons out of the ten are caught by the spitzkasten box and passed through

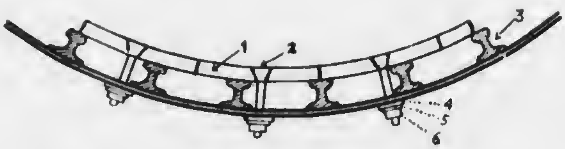


Fig. 23. 1. Steel Plate Liner. 2. Countersunk Head. 3 Steel Rail. 4. Vanner-belt Washer. 5 Iron Washer. 6. Nut

the mill again as coarse concentrate, making eight tons ground to slime and passing into the agitators per 24 hours. The question of a cheap lining in out-of-the-way places like Korea is certainly one of great importance to tube-milling. If more would give their experiences concerning different linings, it would be of great value to us all.

A. E. DRUCKER.

Taracol, Korea, September 29.

A SIMPLE SOLUTION METER

By E. H. Nutter

(December 1, 1906)

In the operation of cyanide plants, it is nearly always important to have a means of correctly determining the amount of gold-bearing solution entering the precipitating boxes. In some plants

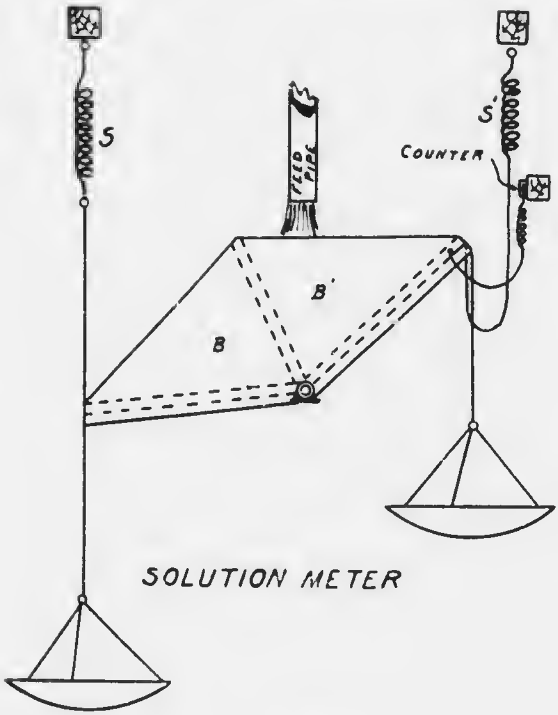


Fig. 24

this is done by alternately filling and discharging sump-vats of fairly good size, and recording the height of solution each time a sump is filled. This is a clumsy, though accurate enough, method, but has the objection that it requires nearly constant attendance.

Various water meters are manufactured which can be used for solution work, except that those of sufficient capacity are expensive, and are open to the objection that the lime which is used in practically all cyanide-plant solutions, soon deposits a scale in the working parts of the meters, which renders them inaccurate or puts them out of business altogether.

To meet these objections I adapted the familiar sampling box illustrated herewith, to the requirements of the case, and one was installed as a meter at the Liberty Bell mill. The device is simple and cheap, and as others may find it useful, a description of it is not out of place.

The arrangement can be seen by reference to Fig. 24. The compartments *B B'* alternately take the solution flow from the feed-pipe. As a compartment fills, the weight of solution overbalances the empty compartment and the box turns through a short arc, discharging as it turns, until it is stopped by the springs *S S'*. Each time *B'* discharges the counter registers 1. The meter at the Liberty Bell is set up over a vat, and the pans shown in the diagram are always submerged. Their retarding effect is necessary to avoid excessive jar. A dash-pot would probably do as well, or else the box could be made of metal with a pear-shaped cross-section, and hung close up to its centre of gravity. The latter form would not need the springs or pans.

At the Liberty Bell, the constant of the meter was determined by filling the vat under it and dividing by the number of double discharges registered. In this way leakage, splash, and all other irregular factors are averaged, and determined. The constant so determined was 0.32275 tons for each unit registered. This meter handles the flow from an 80-stamp mill crushing in cyanide solution. For this size of meter the pans should be at least 36 in. diam., and the springs made of not less than ¼-in. rod. A four-inch hole in the bottom of each pan tends to steady its motion through the solution.

PROGRESS IN CYANIDATION

(Editorial, December 15, 1906)

The article on 'Recent Improvements in the Cyanide Process' by Mr. F. L. Bosqui, will command attention, being written by one who is in the thick of current practice. Mr. Bosqui favors filtering machines of the Moore-Butters type and is inclined to consider, as Mr. Alfred James does, that filter-presses will shortly be relegated to the background. This appears open to doubt. Two large facts are against it, namely, the continued use of presses at Kalgoorlie and the successful improvements recently made by Mr. C. W. Merrill at the Homestake. From Kalgoorlie we hear good news of the Ridgway invention, a horizontal revolving automatic vacuum filter, but this machine has yet to be tried in the United States. At present, of all the sand leached at Kalgoorlie only a few thousand tons are treated by devices other than filter-presses. All the larger mines in the back country use them and the Ivanhoe is now erecting a new filter-press plant. Filter-pressing at Kalgoorlie now costs about 35 cents per ton, and leaves 12 to 15 per cent moisture in the cakes; it is believed locally that a new plant on a large scale could be operated for not more than 25 cents per ton Of course, in regions where water costs nothing and the original slime is docile to treatment, the filtering machines will make a better comparison with presses. Meanwhile, Mr. Merrill's work is bound to encourage the advocates of the older method. He has enlarged its scope and simplified its details. With 24 filter-presses, each weighing 65 tons, and each of 26-ton capacity (as compared to the old 6-ton presses), he expects the total cost of treating slime to be not more than 25 cents per ton. The mills make 1,600 tons daily of this product, and 2,400 tons of sand. A test has been made on 6,000 tons of slime. In charging, he uses a pressure of 35 pounds; while leaching, 12 to 15 pounds; and to sluice with, a pressure of 65 pounds per square inch. Out of a slime containing 91 cents, he extracts 83 cents. Mr. Bosqui gives other details; those we quote are from Mr. Merrill himself. It is a remarkable fact, worthy of note, that after 28 years working, 35 per cent of the Homestake ore still comes from surface excavations. Another matter under trial is the use of zinc dust as against shaving;

the former is gaining ground in America. We are informed that at the Waihi, treating 25,000 tons per month, it requires seven men continuously to clean the boxes, using zinc shaving; at the Homestake, treating 45,000 tons per month, only four men are employed a half-day, using zinc dust. In all these matters there is a constant effort to improve, and it is only by unprejudiced tests that the best method can be determined. On another page, Mr. E. H. Nutter adds greatly to the value of our Discussion Department by a letter in which he compares the use of the Moore and Butters filters. Such first-hand information from practical men will be of service to all those who use cyanide in milling. We trust other friends will not hesitate to put their experiences on record, where it can be of general service.

CYANIDE PRACTICE WITH THE MOORE FILTER

(December 15, 1906)

The Editor:

Sir—Mr. R. Gilman Brown's recent contribution to the literature of vacuum filtering, entitled 'Cyanide Practice with the Moore Filter,' appearing in your issues of September 1 and 8, is one of the few articles that have been published on the subject of increasing interest and growing importance to those who have to face difficult problems in cyanidation. Of the few articles so far published, Mr. Brown's is by far the most thoughtful, and contains detail that others have not attempted to give.

As the writer had more or less to do with the construction of the Bodie plant, and its subsequent operation, and since then has had the opportunity of comparing it with the Butters-Cassel installation at the Combination mine, Goldfield, Nevada, the Butters' plant at Virginia City, and the Moore plant at the Liberty Bell mill at Telluride, Colorado, he has gathered a few facts pertinent to the discussion that may be of some interest.

At the Standard, silex blocks for tube-mill lining were a failure; at the Liberty Bell they are a success. The difference in results can apparently be attributed entirely to the different pebbles used to the difference in the thickness of the material fed to the tube-mills and to laying the silex blocks edgewise instead of flatwise, in the later lining. The other factors affecting the life of the lining can be neglected, in this comparison, as the silex blocks are of the same grade, and look to be from the same quarry; the speed of the mills is the same, and it would be hard to find two ores more nearly the same (with regard to the percentage of hard quartz and soft slime-forming material) than the Liberty Bell and the Standard Consolidated ores.

At the time the silex was tried at the Standard, local pebbles of miscellaneous character, and largely of igneous origin were in use. Their consumption was about 50 lb. per ton of sand reground. To a large extent they wore flat, indicating a sliding rather than a rolling motion. At that time the tube-mill was fed almost entirely from spitzkasten underflow, carrying perhaps 20 to 25% solids. The silex was 2¼ in. thick, and the lining lasted six weeks. At the Liberty Bell, a 2¼-in. silex lining in a mill fed from spitzkasten underflow gave four months of actual service. As all other conditions were practically parallel in the operation of these two mills, except the use of local pebbles in one and flint pebbles in the other, the short life of the silex at the Standard can, it seems, be blamed entirely on the local pebbles used.

At the Liberty Bell there are three tube-mills, one fed by spitzkasten underflow as already stated, while the others receive the discharge from Dorr scraping classifiers, a coarse product carrying from 45 to 50% solids. In these mills, also, the silex blocks have been set on edge, forming a lining four inches thick. By setting the blocks edgewise, it seems likely that the wear is increased as the bedding planes of the original limestone seem to be parellel to the flat sides of the blocks. Capping is certainly not so bad with the edgewise blocks. The edgewise lining has already had nine months of actual service, and seems good for several months more. Roughly, the lining in the mill receiving spitzkasten underflow wore at the rate of one inch in two months, as there was some waste, of course, when it was finally removed. In the other mills the rate of wear has been just about half of this : one inch in four months. Here, then, are three mills on the same ore, and operating under nearly identical conditions, except for the matter of thickness of pulp fed. In the mills receiving the thicker pulp the wear of linings is about half that in the other. The duty of the four-inch lining has been something over 9,000 tons of sand re-ground per inch of wear.

These facts are again illustrated by later results with the tube-mill at the Standard. There, the first wrought-iron lining of ⅞-in. plates lasted 90 days and re-ground about 4,800 tons of sand. During its life only a small tonnage of pond-tailing was introduced to the mill. About the time the second lining of the same kind was put in, arrangements had been completed for introducing pond-tailing in addition to the daily mill tonnage, up to the full capacity of the filtering plant. The percentage of solids in the tube-mill feed during the life of the second lining averaged around 45%, as against 20 to 25% before. The life of the second lining was 170 days as against 90; the total tonnage of re-ground sand increased from 4,800 to over 10,000, and the pebble consumption dropped from 6.5 lb. per ton to 2.6 lb. All other conditions were the same. These results speak for themselves.

At the Standard, the cost for local pebbles at $9 per ton laid down at the tube-mill was $0.225 per ton of re-ground sand, while the cost for flint pebbles under the same conditions at $70.20 per ton of 2,000 lb. laid down at the tube-mill was only $0.233 per ton re-ground, and this has since dropped to $0.091 per ton. Good technical results have been obtained with the Moore plant at the Standard during the last half-year. The extraction has been increased and the cost lowered. These results are entirely to the credit of Mr. H. H. Kessler, who had been in charge of the plant since the first of the year.

At the Standard, during the time the tube-mill was shut down waiting for the cement to harden in the silex lining, the Moore plant limped along as best it could without re-grinding. The sand accumulated wherever it had a chance. The settlers, the agitators, and the filtering vats all furnished their quota of sand and grief. A crew of about six men was kept busy all the time repairing filters, and this one month alone made a decided increase in the costs for the whole year.

Mercury-step bearings for the agitator shafts were a good deal of a failure. The balls ground the iron and the mercury together into a nice iron amalgam, and their use was abandoned.

Mr. Brown has described the basket hoisting arrangement at the Standard. At the Liberty Bell, the baskets are hoisted for transfer by means of hydraulic cranes, operated under a pressure of 250 lb. These work well, and give much less trouble than the differential chain hoists at the Standard. Where high-pressure water is not available, an automatic high-pressure pump installation, such as is used for elevator service, could be utilized. At the Liberty Bell also, an independent electric motor drive is used for traversing the cranes. This is much the best arrangement where one crane serves a number of baskets. The long filter as installed at the Standard did fairly good work but there was, nevertheless, considerable trouble from twinning or coalescing of the cakes on adjoining filters, which prevented efficient washing. The filters continually broke loose from the spacing bars, and before the writer left Bodie he was seriously considering the advisability of dividing each of the 16-ft. filters into two units and stretching the canvas on iron frames. This was not done, but, from the experience elsewhere with the smaller filters, it would be an improvement. Mr. Brown has called attention to the greater efficiency of the filtration when the pulp is thick. This is a matter that should not be overlooked in the design of filtration plants, as it affects the capacity in a very large degree. The reason seems to be mainly that in the thicker pulps the sand is buoyed up, and a better and more permeable admixture of sand and slime is obtained in the cakes.

It is satisfactory to record that the estimated capacity of the Standard plant of 3,000 tons per month has been exceeded by 500 tons during the months when uninterrupted excavation could be carried on in the ponds. Mr. Brown's statement that there was no accumulation of weak solution in the plant, applied to the first period of the operations. After all sources of mechanical loss of solution had been eliminated there was a constant accumulation of weak solution, which had to be run to waste. Mr. Brown makes incidental mention of the Butters-Cassel process as being an inversion of the Moore process, and points out the common essential feature of the two, which is, the adhesion to the filter of a cake of slime, thereby making possible its removal from the unfiltered remaining pulp. In the Moore process the filters are transferred from loading to washing vat, etc., and back again, while in the Butters-Cassel process the filters are stationary, and the pulp, wash-water, etc., are transferred to and from the filtering vat. When large capacity can be obtained from a relatively small filtering plant, the Butters-Cassel process has several advantages over the Moore process, but where the filtering plant must be large the reverse is true.

It must be borne in mind that there is a wide variation in the slimes made from different ores and that they act differently in filter-plants, so that a plant designed to handle a certain tonnage at one mine will have a different capacity at another. Thus at the Combination mill at Goldfield, the loading period is only 30 minutes, while at the Standard it varies from $2\frac{1}{2}$ to 6 hours, depending on the proportions of clay slime and fine sand in the charge, and the thickness of pulp. Therefore, a plant designed to handle 60 tons per day of Combination slime would fall far short of handling 60 tons of Bodie slime. At the Liberty Bell the loading period is one hour while at Butters' Virginia City plant, the time varies from 30 minutes to one hour. At all of the plants it is the aim to build up a cake of from $\frac{3}{4}$ to 1 in. thick.

It will be assumed here that the essential operations of loading, washing the cake, and discharging in water, will require the same length of time with the two processes on the same slime, and this discussion is consequently narrowed to a consideration of the relative merits of transferring the filters and transferring the pulp, wash-water, etc. It is true that with the Butters process a somewhat higher vacuum can be maintained within the filter, on account of the possibility of arranging for bottom solution discharge from the filters, which cannot be done easily with the Moore process. The difference is not great, however, as the solution is raised in the suction pipe in the Moore filters on the principle of the air-lift, and there is not, consequently, a solid column of solution, reducing the vacuum foot for foot of lift in the suction pipe. The rate of loading, also, does not increase directly with the vacuum, but is greater at times with a lower than with a higher vacuum.

At the Liberty Bell, when everything is working smoothly, the Moore plant will just about hold even with the mill crushing 350 tons per day. In order to give the cakes a preliminary weak solution wash and allow also for repairs, another set of filters and another washing vat are to be installed. Besides the necessary settlers and agitators, the plant at present contains six vats, two of which are for loading, and five baskets or filter sets. The baskets are transferred by two overhead traveling hydraulic cranes, and each transfer takes ten minutes. Each complet cycle requires two hours and forty minutes, divided as follows: One hour and ten minutes loading and transferring; one hour and thirty minutes washing, discharging, and transferring back to loading vat The time required for a cycle with the Butters-Cassel process depends on the time required for transferring pulp, water, etc., which is dependent on the size of vats and pumps.

To arrive at a comparison, let it be assumed that the Liberty Bell plant is to be changed from the Moore to the Butters system. Instead of the present arrangement of two loading and four washing vats, these, and the additional ones necessary will be used for all operations. At Virginia City the time of filtering and emptying the filtering vats is from 15 to 20 minutes. At the Combination it is from 15 to 25 minutes. Each filtering vat at the Liberty Bell contains 3,050 cu. ft. A 6-in. centrifugal pump at normal speed would fill or discharge a vat in approximately 28 min., a 7-in. pump in 18 min., an 8-in. pump in 14 min., and a 10-in. pump in 7.5 minutes. The large pumps involve the use of large valves, which are a source of trouble in pumping sand, while the smaller pumps involve a larger plant. For a middle course, let us consider the installation of 7-in. pumps. It will practically be necessary to install a pump to each vat, in order to avoid undue valve complications. The cycle for each vat, then, will be as follows: Loading one hour, pumping out pulp 18 min., pumping in water 18 min., washing the cake one hour, discharging 15 min., pumping back water 18 min., pumping in pulp 18 min., total 3 hr. 27 min., say 3 hr. 30 min., as that much, if not more, extra time will easily be consumed. This cycle is longer by 59 min. than with the Moore plant. At 45 filter-charges per day—the usual run—there is a total time increase of 37 hr. 30 min. required to handle the tonnage, and provision will have to be made to handle 11 extra charges per day over what the present plant would take care of. This means the installation of three more filter sets and two more vats. In addition, two storage vats would have to be provided, one for water and one for pulp for taking care of the flux of material during operation. These should have a capacity of at least 6,000 cu. ft. each, and more than this would be desirable. We have, then, two filtering vats of 3,000 cu. ft. capacity, three additional baskets, eight 7-in. centrifugal pumps, with the necessary pipe connections, three-way and single valves, belting, line shafting, motor drive, etc., and a light crane for handling filters for repairing; and two storage vats of not less than 6,000 cu. ft. capacity each, all of which are balanced against two heavy traveling hydraulic cranes, with the necessary heavy track supports. Roughly, the first cost would be about $6,000 at Telluride in favor of the Moore process. The chief expense, however, amounting to more than the increased first cost, would be the power cost for driving the centrifugal pumps. The power required for pumping would vary from a minimum at the beginning to a maximum at the end of the transferring period, complicated of course by the workings of the various pumps, to the extent of their effect on the height of material in the storage vats. At the Liberty Bell, any arrangement that could be made without the reconstruction of the present filtering vats, would necessitate each pump working against a maximum head of 19 ft., and each pump would then take about 11 h.p. part of the time. As power is bought on peak load in the Telluride district and cost a minimum of $5 per horse-power month, it would be hard to say what the extra power bill would be. Should four pumps be working near the maximum lift at one time, which would not be unlikely, the bill for the month would be about $225. It would probably not be less than $150 any one month, and might run up to $300 or $400. There would be no saving in labor in one plant over the other, as one man on a shift would be required to attend to the filtering in either case.

EDWARD H. NUTTER.

Telluride, November 25.

RECENT IMPROVEMENTS IN THE CYANIDE PROCESS

By F. L. Bosqui

(December 15, 1906)

*It was nearly twenty years ago that two Glasgow chemists, MacArthur and Forrest, made the first practical application of the dissolving action of a dilute cyanide solution on gold. The process was at once adopted in New Zealand and South Africa. In the latter country all the conditions were most favorable to its success; and the enormous profits yielded by the pioneer plants at once established cyanidation as a process of the greatest commercial importance.

The procedure adopted in the Transvaal was simple in comparison with recent modifications of the process. The tailing from the stamps, after hydraulic concentration of coarse sand and sulphides, was gathered in leaching vats; the slime that overflowed was run into huge shallow vats, the surplus water decanted, and the slime subjected to a series of agitations and decantations, until the mineral that it was found economical to extract was finally precipitated from the solution.

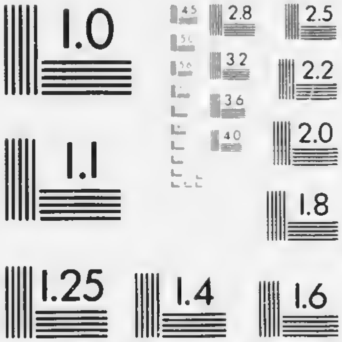
At first, zinc shaving was universally used as a precipitant, but this was superseded by the electrolytic deposition of the gold on sheets of lead. This was adopted in all the representative plants; but its popularity soon waned, owing to the production of troublesome by-products, the awkwardness of the clean-up and bullion recovery, and the unsatisfactory deposition as compared with that obtained on zinc. Its chief advantage was that it recovered the values from extremely dilute solutions, but this advantage was nullified by Betty's discovery that zinc shaving if dipped in a weak solution of lead acetate, would accomplish the same thing.

During recent years, no radical changes were made in the process in South Africa. This was due to the serious blow given the mining industry by the Boer war; and also, in part, to the conservatism of metallurgists on the Rand and their reluctance to

*Revised by the author from a paper read before the American Mining Congress.

MICROCOPY RESOLUTION TEST CHART

(ANSI and ISO TEST CHART No. 2)



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ad important innovations originating elsewhere. The brothers Denny were the first of the Rand metallurgists to recognize the importance of finer grinding, and their energetic advocacy of tube-milling and filter-pressing finally resulted in the acceptance of Australian methods.

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

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

The obvious objection to the old type of filter-presses is the high cost of installation and operation, but they nevertheless enjoyed a great success; and it is worthy of note, in observing the evolution of the process, that they were a means of emphasizing the importance of fine grinding and helped to establish the tube-mill. It has always been a truism in cyanidation that the finer the product, the higher the extraction. This is the case with but few exceptions. To apply this principle required two things; an economical machine for fine grinding and a filtering system that would be at once efficient and economical. You are all no doubt familiar with the tube-mill as now applied in cyanide work. It consists of a sheet-steel cylinder with cast-iron ends, varying in size (the largest mills are 5 ft. diam. by 22 ft. long), and supported either upon trunnions or upon steel tires revolving on rollers like a chlorination barrel. The interior of the mill may be lined either with cast iron, or a species of natural flint known as 'silex.' The latter is the more commonly used, and is sold in two sizes; blocks $2\frac{1}{2}$ in. and 4 in. thick. The silex lining is laid in cement and will last from four to eight months, depending upon the ore. When ready to operate, the mill is charged about half-full with flint pebbles. The product to be reduced is fed into the mill either through a spiral, or a device of the stuffing-box type, and the re-ground material is discharged at the opposite end, being finely comminuted by attrition against the flint pebbles and the lining during the slow revolution of the cylinder. The average speed of the tube-mill is from 25 to 35 revolutions per minute. The fineness to which the sand is reduced will depend upon several factors, chief among which is the amount of water used. The best proportion has been found to be one part solid to one part water. As a machine for economically reducing ore to an extreme fineness, the tube-mill has no equal. The cost of operation is variable. In this country and in Mexico it will range between 20 and 40c. per ton. The work at El Oro, Mexico, and at Telluride, Colorado, is representative of the best practice on this continent; while that at the Combination mine, Goldfield, Nevada, probably represents the maximum of cost, owing to high price of power and labor. A small 4 by 12 ft. trunnion mill is installed at the Combination for sliming the 40-mesh product from a Bryan mill. The product of ten stamps, about 35 tons of ore per day, passes to the tube-mill classifier; and of this product about 75% goes to the tube-mill, or 24.6 tons per day. The following figures may be of interest:

Cost of $2\frac{1}{2}$-in. silex lining laid in mill	$323.50
Life of lining	4 months
Cost of lining per ton of ore stamped	7.7 cents
Cost of pebbles delivered at Goldfield	$71 per ton
Consumption of pebbles	2.03 lb. per ton of ore stamped
Cost of pebbles per ton of ore stamped	7.1 cents
Power consumed	25 h.p. at $11.25 per h.p. per month
Cost of power	26.7 cents per ton of ore stamped
Summary, cost per ton of ore stamped:	
Pebbles	$0.071
Lining	0.077
Power	0.267
	$0.415

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

I have already referred to filter-pressing as an established practice in Western Australia. The press was never very popular in America, and few successful installations are recorded. The most noteworthy, perhaps, is that of the Gold Road mine, near Kingman, in Arizona, where two five-ton Dehne presses have been successfully operated for some time. About three years ago Mr. George Moore, after a series of failures in an attempt to filter-press slime at the mill of the Consolidated Mercur Co., in Utah, devised a vacuum-filter and installed a plant in the Mercur mill. This was the origin of the vacuum-filter, recent modifications of which are to be found at a number of mills in this country. Experiments recently made in Australia so far demonstrate the superiority of this method over all others, that it seems safe to predict the early disappearance of the filter-press.

The unit of the Moore filter is a rectangular wooden frame covered with canvas, and provided with a vacuum drain-pipe extending to the lowest point of the interior. These frames are grouped together in clusters or baskets, which are raised and lowered by means of a hydraulic crane. When lowered into a suitable compartment containing the slime pulp, the vacuum is applied to a common pipe connected with each frame. The solution is drawn through the canvas, and a slime cake varying from $\frac{3}{8}$ to $\frac{7}{8}$ in. thick is deposited on each side of the filter-leaf. The cluster of filter-leaves carrying the charge of slime, weighing several tons, is then lifted from the pulp, shifted automatically to an adjoining compartment containing the wash, where it is again lowered, the vacuum applied, and the displacing operation repeated. The load is again raised and shifted to a bin where the cakes are discharged by introducing air into the interior of the leaves.

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

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

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

Three men at $4; $12 per day	30 cents per ton
Twelve h.p. at $11.25 per h.p. per month	11 " "
Lubricants and incidentals	4 " "
	45 cents per ton

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

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

Driving a 4-in. Butters centrifugal pump.

Operating a 12 by 10 Gould's vacuum pump.

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

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

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

The power may be distributed as follows:

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

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

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

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

I shall not venture to describe the different problems encountered and successfully solved by Mr. C. W. Merrill at the Homestake in the cyaniding of a tailing averaging less than $1.50 per ton. The next and most serious problem to engage his attention was the treatment of the slime, of which 1,600 tons per day of an average value of 80c. per ton have been run to waste from the Homestake mills. Mr. Merrill has described a filter-press the unique feature of which is that it can be automatically discharged by sluicing without being opened, thus doing away with the chief objections of the old type of press, namely, the cost of operating. This press is of a common flush-plate and distance frame pattern, but consists of much larger units. The dimensions are as follows:

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

The pulp is admitted to this press through a continuous channel at the centre of the top of the frames. When the cake is formed, cyanide solution is forced into the cake through channels at the upper corners. At the bottom of the frames, there extends a continuous channel within which lies a sluicing pipe, provided with nozzles that project into each compartment. This pipe can be revolved through an arc of any magnitude so as to play a stream into any part of the cake, washing it down into the annular space between the central channel and the solution pipe. When the press is being filled and leached, the discharge ends of this pipe are sealed. The method of operation is as follows:

The slime, after partial de-watering, consists of about three parts of water to one of solids. In this form it is charged by gravity to the presses at about 30-lb. pressure. The leaching with cyanide solution is done in the press, the effluent solutions being conducted to four precipitating vats where the gold is recovered on zinc dust. There is no power cost for agitating or elevating, except for elevating the solution to the press. There will be only 0.6 ton of solution handled per ton of slime, of which only 0.3 ton will be precipitated. All filtering will be done by gravity at a cost of two cents per ton.

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

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

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

Estimated cost of slime-treatment	25 cents per ton
Estimated profit per year	$300,000
Cost of plant	$400,000
Cost of pipe-lines	$50,000
Area covered by plant	270 by 65 ft.
Area covered per ton of capacity	Two square yards
Cost of installation per ton of capacity	$300
Filter-press cost per ton of capacity	$60
Power required	1/10 h.p. per ton of slime treated
Water required for sluicing	4 tons water to one ton of slime

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

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

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

CYANIDE PRACTICE AT KALGOORLIE

(December 22, 1906)

The Editor:

Sir—I consider it a fortunate moment when a copy of your issue of July 28 came into my hands and enabled me to read with interest and also a little amusement an article by Mr. Alfred James on 'Crushing and Grinding Practice at Kalgoorlie.' As you mention that any reply to Mr. James' criticisms will be given a courteous hearing, I trust that it will be so in this instance.

With the exception of a few remarks in the first part of the article in question, there is little to disagree with, but it is when Mr. James leaves general views, and gets down on to more detailed work, that he appears to lose his balance, especially in regard to his criticisms on the tests made at the Ivanhoe mine, at Kalgoorlie, between grinding pans and tube-mills. Although I left the service of that company in March last, it is natural that an attack (like the one now under review) on those experiments, which were carried out by Mr. R. B. Nicolson and myself, should evoke a reply.

Taking that part of Mr. James' paper which relates to pans against tube-mills, he opens with a reference to a mysterious mine at Kalgoorlie, that is practising bromo-cyanide treatment without using bromo-cyanide methods, and in consequence the residues of this mine are reputed to be worth 2½ dwt. per ton. With the exception of the loss in residues, I have a shrewd suspicion what the name of the mine is, but would certainly advise Mr. James to call at headquarters and get correct figures for the past few months, which are, I dare say, considerably under his estimate.

The question of extraction was entirely left out of the Ivanhoe tests. The point to be proved was, which was the superior grinder, a pan or a tube-mill. Personally, I think very little difference would be found in the extraction obtained from the slime made by either of these two machines.

I am glad to see Mr. James regrets that names of such high standing should have been tacked on to tests so badly carried out as were those at the Ivanhoe. While appreciating the compliment in the first part of this sentence, I would also apply the same remarks to Mr. James and his article. This same gentleman, by a wonderful though erroneous deduction, then discovers that as the Ivanhoe tube-mill worked on alternate days, the flints were coated with slime at the start of each daily test! For Mr. James, benefit alone, I would inform him that at the end of a day's run of the tube-mill, the feed was shut off and clean water passed through for some time before the mill itself was stopped, and on re-starting for another day's run the mill was allowed to work for some time until its full load was reached before samples of any kind were taken. Setting aside this precaution, the fate of any slime left inside a tube-mill revolving at 32 revolutions per minute can be imagined.

Mr. James now comments somewhat sarcastically on the large amount of 'slime' (?) (150 mesh) which is allowed to return to choke the tube-mill instead of being got rid of at once, and shows that the Ivanhoe mill had more slime returned than was its daily output of this product. The statement made in the Ivanhoe report that the tube-mill became choked, has been taken perhaps in too literal a sense. It is obvious that as long as the inlet and outlet of a tube-mill is large enough, practically any tonnage of sand can be rushed through. What happens in a case of this kind is that the various spitzkasten used for separating the resultant slime and feeding the mill are gradually choked by the accumulation of fine sand returned for re-grinding, the effect of the tube-mill having more original feed than it can cope with, or, in other words, the mill may be said to be 'overloaded.' This is what happened during the Ivanhoe experiments. The tube-mill did not exactly choke, but the various spitzkasten employed did, and after every expedient failed the fine sand was only prevented from gradually gaining on the mill by reducing the amount of original feed. As there is a certain time at which a tube-mill must become overloaded, that particular point was naturally presumed to have been reached. The same remarks also can be applied to the grinding pans.

At this point it may be as well to get an idea of what Mr. James does consider good work by a tube-mill, and I have gone to some trouble in turning up a few statements made by him during the last few years, and in one of them in *The Engineering and Mining Journal* of January, 1905, I notice he makes the following remarks:

"I ask why is it that the Hannan's Star mill, the first one laid down, should be still doing the best work, while the Ivanhoe mill has been thrown out?" And further on we have: "The longest and oldest mills, those at Hannan's Star, and the old Brownhill exhibit the best results, etc."

Turning now to the *Journal of the Chamber of Mines* of Western Australia of March, 1904, we see some working results of this same Hannan's Star mill, which Mr. James appears to never tire in quoting. A glance at this latter paper reveals, in the light of present remarks, some astonishing figures, and I shall trespass a little on your space to reproduce them alongside those of the Ivanhoe. Before giving these it must be remembered that the Hannan's Star tube-mill was 16 ft. long, taking 30 h.p., and was reducing 38 tons of sand per day to a slime of -150, while the Ivanhoe mill was 13 ft. long, requiring 20 h.p. and bringing 19.5 tons of sand to a slime—also of minus 150. It may be added that of the original feed of 38 tons sent to the former mill, 20.5% remained on 40 mesh, while of the 19.5 tons sent to the Ivanhoe tube, 51% stayed above 40 mesh, or, in other words, the original feed to the Ivanhoe mill was 2½ times as coarse as that sent to Hannan's Star. As it was demonstrated, in tests carried out at the former mine, that coarse sand had a bad effect on tube-mill work, this fact, taking the size of the two tube-mills into account, explains most of the difference in output.

As 'slime' (?) in the return feed is the product that worries Mr. James so much, I give below the grading from the two tube-mills:

	Hannan's Star mill (16 ft.) 268 tons per day, being original feed and sand returned for regrinding.		Ivanhoe tube-mill (13 ft.) 142 tons per day, being original feed and sand returned for regrinding.	
	%	Tons.	%	Tons.
On 40 mesh	4.5	12.06	11.40	16.19
" 60 "	9.5	25.46	14.98	21.27
" 100 "	19.9	53.33	36.94	52.45
" 150 "	19.0	50.92	17.77	25.23
Through 150 "	47.1	126.22	18.91	26.85

A glance at the above figures shows that the Hannan's Star mill with an original output of 38 tons of -150-mesh slime per day contained in its return feed no less than 47%, or 126 tons of this same objectionable product, against the Ivanhoe mill with an original output of 19.5 tons, containing only 18.91%, or 26.77

tons. In other words, we have the extraordinary position of a man who for the last few years has been holding the Hannan's Star tube-mill up to the admiration of the mining world, executing a complete somersault, and condemning the tube-mill practice on a neighboring mine, when on his own showing this latter mill is doing far better work, especially as far as clean separation goes, than the one he is so proud of. It is unfortunate that Mr. James has allowed personal feelings to obscure his judgment on this occasion, and if he now puts the Hannan's Star mill figures in the place of the Ivanhoe, it is evident in what an unfortunate position he has placed himself.

As it is hardly necessary to comment any further on this, we will now go on to another paragraph in the paper under review where complaint is made that the pans — especially the first one — had an undue advantage over the tube-mill in matter of getting rid of finished product. Surely Mr. James does not condemn the practice of running the original or coarse feed into one pan and the return fine feed into a second to be slimed. This is one of the advantages of pans over tube-mills, as in the case of the latter one machine has to both reduce coarse sand and do the sliming as well. However, if Mr. James had looked at the feed entering the second or finishing pan he would see that out of a daily tonnage of 74.6 tons returned to this pan no less than 24.5 tons passed 150 mesh (being nine tons more than its actual output of this product), or, in other words, this pan had practically the same quality of feed as the tube-mill, so that both were laboring under the same disadvantage. It may be consoling to the gentleman in question that all tube-mills and pans grinding to a slime exhibit the characteristic of having more —150 'slime' (?) returned to them than is their actual output of this product and it is, I should say, the result of not having a proper definition of what constitutes slime, and a better plan for separation of the same. Numerous attempts have been made of late to define what slime is, but so far the results have not been satisfactory. The common and practical definition at Kalgoorlie is: All that product which passes a screen having 150 holes per linear inch. This, however, is far from satisfactory, for anyone conversant with grinding practice knows how large is the quantity of very fine sand that will pass a 150-mesh readily, but is yet heavy enough to sink in a spitzkasten, and be returned to the tube-mill or pans again and again. Recognizing these difficulties, it was stated at the outset of the Ivanhoe experiments that all that product which passed a 150-mesh would be called 'slime,' whether it came from a tube-mill or grinding pan. As some basis had to be made for comparison, this was thought to be the fairest and the returned slime which causes Mr. James so much worry is in reality very fine sand. In the Ivanhoe report the eturn feed is spoken of in every instance as 'sand,' and, as shown, both tube-mill and pans had their complement of it. If in place of writing contradictory statements, Mr. James would set himself the task of defining what slime is, and, having satisfied the mining community on that point, he then invents some simple appliance which would eliminate that slime, and only sends back to be re-ground that part which according to his definition required it, then perhaps further trials may be undertaken between grinding machines that would satisfy even the most critical.

The fact of the Ivanhoe tests being checked by a tube-mill expert sent by Messrs. Bewick, Moreing & Co., from the Oroya Brownhill mine, is sufficient to remove any chance of bias on the part of those responsible for the figures given.

Coming now to the cost allowed for flints and liners—which Mr. James remarks as being excessive—I can only state that the costs shown were based on the consumption experienced during the progress of the experiments and which agreed closely with results obtained on a large scale on neighboring mines, and that, as shown later on, the longevity of liners in the Kalgoorlie district is unique. Leaving out the difference in wear and tear of working parts, the question of a pan being superior to a tube-mill in breaking down coarse sand is so evident that it is hardly worth discussing.

It comes as a surprise to those unacquainted with fine-grinding operations at Kalgoorlie that iron liners should still be in use, but when we consider the records put up there on hard sulphide ore, one-inch iron liners lasting from four to six months, or longer than three-inch silex liners do on the Rand, it is perhaps no wonder that Kalgoorlie metallurgists are reluctant to discard such a useful friend. The reason for the great difference in wear in liners between two such important mining centres is worthy of careful investigation. From knowledge of both Kalgoorlie and Rand practice I would say that the sand of the latter place is much sharper, and also the fact that the product entering the tube-mills on the Rand is coarser on account of it practically only passing through once, while Kalgoorlie mills in grinding to a slime have a large percentage of original feed returned again and again, and in consequence the feed entering them is much finer. Another peculiar circumstance is that, while it is difficult on the Rand to find a tube-mill under 22 ft. in length, in the whole of Western Australia there is no mill over 16 ft., the majority being but 13 ft. long!

A practical proof of the esteem in which grinding pans are held is that about 200 are now engaged in Western Australia, 40% of this total in grinding raw sand, admittedly not to a slime, for those mines which are sliming their total product had tube-mills erected or on order before the remarkable grinding ability of pans was recognized; but where it has been decided to increase the capacity of these latter plants, pans have invariably been installed in preference. It is perhaps worthy of notice that about a dozen have also arrived or are on their way to South Africa.

It is a matter of sincere pleasure to those responsible if in the smallest degree the increased use of grinding pans has been due to the Ivanhoe experiments, and also the recommendation to millmen of a grinding machine that (to use Mr. James' own words) has the advantages over tube-mills of "using less horse-power per unit, their greater convenience in working, their capacity for amalgamation, the granular (?) nature of their product," and it may be added (until demonstrated otherwise) their undoubted superiority in breaking down coarse sand, and their more than equal success in sliming.

H. T. BRETT.

Sabiwa Mine, Gwanda, Rhodesia. October 4.

CYANIDE PRACTICE AT KALGOORLIE

(December 29, 1906)

The Editor:

Sir—With reference to Mr. Brett's long letter in your issue of December 22, of which you kindly sent me an advance proof, I notice that in spite of the space at his command Mr. Brett does not apparently deny my suggestion that in his so-called tube-mill versus pan tests he actually fed into the tube-mill with the sand to be re-ground more slimed (finished) product than sand to be re-ground, although he must have known that such a procedure would seriously diminish the duty of the tube-mill.

As Mr. Brett does not deny this, his long personal explanation calls for no other comment from me; but I would like to add that the pan work at the Ivanhoe has impressed me so favorably that were I laying down an installation of pans I should deem myself fortunate if able to secure the services of Mr. Brett for this purpose—so efficiently have the pans been worked at his late mill at Kalgoorlie.

ALFRED JAMES.

London, December 5.

PROGRESS IN CYANIDATION DURING 1906

By Alfred James

(January 5 1907

The making of slime and the treatment of it may be taken as the main directions of improvement in cyanidation during the year. The use of tube-mills has progressed by leaps and bounds, and is now rapidly becoming well-nigh universal, but with the adoption of these machines the question of slime treatment has forced itself into still greater prominence, and as stated by me last year, some of our keenest metallurgists have been attacking this problem, and it is now possible to state that their endeavors have met with a considerable measure of success.

As will be indicated later, progress has also been made in the treatment of difficult ores—including silver-gold ores, antimony-gold concentrate, and cupriferous tailing—and in crushing, roasting, and conveying we are ahead of last year's practice.

It is interesting to note how the treatment of gold ores has overlapped the metallurgy of the other metals. Fine sliming and prolonged cyanide treatment constitute a method that, in many cases, is ousting chloridizing roasting and the old patio process for the treatment of silver ores. Copper ore in Chile is now being leached in a cyanide plant—without cyanide—and the Merton roaster, so successfully introduced in Australia for gold ores has taken a hold on the zinc industry for the roasting of zinc blende, and for the treatment of the zinc-lead-silver concentrate of Broken Hill. Indeed one cannot fail to be struck by the way in which names well known in gold-milling, such as Argall, Marriner, and Simpson, have been attracted to the problems presented by zinc-lead ores.

It is difficulty to put a finger on any one part of the world as having made the most striking progress of he year. Western Australia still has the pre-eminence. Inventions that may have originated in other countries appear to be more quickly put to definite practical service there than elsewhere, and just as Western Australia has taken the lead in fine sliming and in circulating cyanide solutions through the mill, so now it seems to be the pioneer in slime treatment. Probably the emulation arising from the up-to-date methods of the leading firm of mining engineers on the one hand, and independent managers on the other hand, has tended to place Western Australia in the forefront in the matter of new methods, high extractions, and low costs. Certainly such men as Hamilton, Moss, Nicholson, and Klug, and the gentlemen associated with Bewick, Moreing & Co., have cause for congratulation on the results they have achieved and are achieving, and the adoption of their methods in other regions shows that the industry, not locally but generally, is under an obligation to them, as well as to the alert and progressive West Australian Chamber of Mines, which has published an excellent series of articles by Mr. Robert Allen on ore treatment as carried out at the various local installations.

American practice has chiefly busied itself with tube-mills and slime treatment. Argall, Bosqui, Butters, and Merrill have all been contributing to our advancement by getting out improved methods. In Africa the publication of the results obtained by the brothers Denny at the Meyer & Charlton and New Goch mills has caused much interest, for extraction and output have been increased and cost lowered—mainly by the use of tube-mills. Record gold outputs are again being produced from the Transvaal, which has more than recovered the set-back occasioned by the war.

Crushing.—The tendency to stage-crushing is in still greater evidence. Coarse breakers, fine crushers, heavy stamps, pans, tube-mills, are not an uncommon sequence. With reference to breaking of ore, however, it has not yet been proved that reduction to finer than 2½-in. cube is advantageous for the stamps. Experiments seem to indicate that finer crushing is more expensive than the gain arising from any increase in stamp duty will warrant.

As for crushers, the Gates type still holds the field for big work, but the Bigelow breaker, of the Blake type but with the pitman working in compression and having its weight assisting the crushing stroke, appears to be replacing the other kinds at Kalgoorlie, and to show a lower running cost than either the gyratory or the older reciprocating patterns.

Stamps of 1,500-lb. weight are now in use on the Rand, and it looks as if the limit of weight had not yet been reached. But little has been heard this year of anvil-blocks for mortars, although at the Waihi Grand Junction they seem to have made for a considerable increase in output beyond that originally expected. Judging by out-put per horse-power, ball-mills would appear to be the most efficient machine, with an output of over two tons per h.p. day on the best types, crushing all the ore through 27 mesh, but the cost of dry-crushing at the South Kalgurli appears to be about three shillings per ton as against roughly 1s. 9d. per ton crushing wet at the Oroya-Brownhill, Lake View, and Ivanhoe. On the other hand the latter crush through very much coarser screening (10, 17, and 15 mesh, respectively), reducing the over-sized product by their pans and tube-mills.

Fine Grinding.—The tendency has been to use stamps for rough crushing only, that is, to substitute coarser screens for those formerly in use, and from the pulp to separate the coarse product and to grind it in tube-mills or pans. There have been no notable contributions to our knowledge this year as to which is the best appliance for this purpose. Tube-mills seem to be generally employed, except at Kalgoorlie, where the roasted ore can be efficiently ground to the required fineness in pans. Not that pans are not effective on raw ore. On the contrary, the Ivanhoe results show very good work indeed, and it is to be much regretted that the comparative tests made at that mine between pans and tube-mills were carried out in such a way as to make the results worse than useless. But it is a matter for serious thought that the Ivanhoe costs for re-grinding sand should be only 8½d. per ton treated, when the Oroya-Brownhill, Lake View, and other mills should show a figure twice as high. It cannot well be a matter of pans only, or we should have heard of similar low results from the other companies exclusively employing pans (at the Oroya and Lake View pans are used in conjunction with tube-mills). Is it that the Ivanhoe ore is softer or that the final product is not slimed as fine? The management has recently been very keen on reducing the value in their residue—and have succeeded in doing so materially—and one would therefore expect that they would not overlook the gain from crushing as fine as their neighbors. It is to be assumed, therefore, that the crushing is as fine, but the fact remains that the cost of grinding sand at the Ivanhoe is less than half that of neighboring companies of the same group that employ wet crushing. There is an obvious need of that careful comparative test between pans and tube-mills for which I have so often asked and which I hoped would be undertaken by Klug or

Denny. Indeed, it was understood that the latter was making such tests, but nothing seems to have come of them.

Meanwhile I am informed that such a test has been made at Broken Hill with a pan, a wet-grinding ball-mill, a disc-grinder, and a tube-mill, and that the tube has proved itself the best of the lot, but in the absence of details, we can only take this statement as the expression of individual opinion. Even if we assume tube-mills to be the best slime-makers there is still the question of stage-grinding, and it is by no means certain that the preliminary crushing of the coarser particles could not be effected more economically in pans.

The best pan figures have been given above. The following tube-mill data will prove of interest. African costs have been reduced from 8½d. (per ton ground) down to 5½d., of which 1.89d. is for power, 0.7d. for pebbles, and 0.93d. for liners. Mr. Henry Leupold states that silex linings are half the cost and last two and a half times as long as iron liners. The Consolidated Gold Fields, made careful tests on manganese steel liners as against silex with the result that the former were shown to be more expensive and the grinding nothing like as good, owing to the slipping of the pebbles caused by the polished surface of the manganese steel. At El Oro, a 20-ft. mill with 7½ tons of pebbles grinds 125 tons per day of sand for 60 h.p. to such a fineness that 90% will pass 100-mesh and 50% a 200-mesh screen. At Waihi, an 18-ft. mill grinds 77 tons of 20-mesh sand per day, so that 93% will pass 150-mesh, for 37½ h.p. The Waihi is notorious for the hardness of its ore, which ball-mills failed to grind satisfactorily. Their tube-mills do not appear to be run to their utmost capacity, but their figures are interesting as an example in practice of direct tube-mill work, that is, without any return of material treated. The above figures, of over two tons slimed per horse-power, at first sight show a less duty than that mentioned by me a year ago, of four tons per horse-power in Africa, but it must be remembered that the African crushing was much coarser, they having at that time adopted a standard of under two per cent retained on 60-mesh.

But apart from the question of comparative cost of work done, tube-mills have shown their distinct economical advantage as well as their capacity for re-grinding. Reference was made last year to the huge additional net profit of £65,000 per year accruing to the El Oro Company from this source. At Waihi three 18-ft.

tube-mills and 90 stamps have increased the output from 2.89 tons per stamp per day to four tons, and have lessened the value of the residue by 50%. They have improved the extraction by amalgamation from 5 to 7%, and effected a 75% saving in screening. They find also that since the adoption of tube-mills the slime is more permeable and the filter-press charges take less time for treatment, so that the same presses can now deal with 30% more slime than before. The new Barry linings, made on the spot, seem to be most effective and to result in a great saving of cost over the imported silex. One set of linings is stated to last for six months, and a new set is always kept ready at hand and can be replaced in two days. Mr. H. P. Barry makes a great point of having his linings as rough as possible, so as to give a grip to the pebbles, and has modified his doors to effect this also.

At Johannesburg Mr. W. R. Dowling emphasizes the necessity for the complete separation of the sand to be re-ground from the slime in the mill-pulp, and for keeping the proportion of water to sand at as low a figure as possible. He shows at the Robinson Deep a 0.48 dwt. lower residue (formerly 1.26 dwt., now after tube-mill treatment 0.78 dwt.), and an increased capacity of 2,300 tons per month. The Eckstein mines in addition to the increased output resulting from the tube-mills, show an additional profit of from 1s. to 1s. 6d. per ton (1s. at the Robinson, 1s. 4d. at the Ferreira) resulting from the increased extraction, which has lessened the value of the residue by about half a pennyweight. There are now 58 tube-mills at work on the Rand—and more on order—at 24 different mines, and these 58 tubes at the 24 mines only have increased the amalgamation returns of the whole of the Rand by three per cent.

At present the Knight's Deep appears to have the highest monthly output per stamp—for complete battery—of 7.68 tons per stamp, since reduced by the substitution of different screening to 6.68 tons. In Western Australia the Oroya-Brownhill crushes 7¼ tons per stamp through a 10-mesh screen, and the Great Fingall (1,150-lb. stamps) exhibits an average duty of seven tons per 24 hours through a 12-mesh screen, and the Sons of Gwalia at Leonora (1,000-lb. stamps) treats 6.68 tons per head per day through 20-mesh screening.

It is estimated that, in addition to the gains already mentioned as resulting in Africa from the use of tube-mills, there is a saving of 30% in equipment arising from the adoption of them in place of obtaining the same output by the addition of the increased number of stamps necessary.*

Amalgamation.—The increased extraction by amalgamation resulting from the use of tube-mills, has been referred to already. At Waihi, amalgamation is carried out on plates set in a large building apart from the mill, sand and slime being treated separately. There is much to be said for this arrangement. Companies circulating cyanide solutions through the mill find a scouring of their copper plates by the cyanide solutions. Denny, at the Meyer & Charlton, proposed to minimize this action if possible by adopting shorter plates, but probably the better method would be to exclude copper plates from the amalgamation apparatus. At the Lake View (Kalgoorlie), for instance, plates are discarded and amalgamation takes place in pans; in that mill 51% of the value of the concentrate is recovered by amalgamation and 46% by subsequent cyanidation.

Roasting.—Thomas Edwards has increased the capacity of his recent furnaces by a parallel system of rabbles, which is stated to promote inter-rabbling. G. C. Klug has adopted this for the Holthoff-Wethey furnaces at the Perseverance, and claims high results. Merton has also modified his furnace by making it a five-hearth roaster, and also of greater hearth-area. The great Boulder Proprietary has a roasting cost of 2s. 3d. per ton (employing Merton furnaces) reducing an ore going 4 to 5% sulphur down to 0.07% sulphur, as sulphide; with this exception the record of the South Kalgurli type of Merton furnaces does not yet appear to have been improved on, either for cost (2s. 6d. per ton roasted), output (32 tons per diem for small-size furnace, taking 1½ h.p.), or efficiency (3.1% sulphur down to 0.01% sulphur as sulphide; one ton of green wood roasting 11 tons of ore). With regard to the suggestion that I made in this review a year ago of adding, say, two pounds of lead acetate per agitator charge (40 to 60 tons) of roasted ore, Richard Hamilton of the Great Boulder finds that the use of lead salts appears to result in an increased consumption of zinc in the precipitation boxes, as against which he benefits by a saving of roasting fuel, an increased tonnage through the mill for the same labor, a reduction in the value of the

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**South African Mines*, October 27, 1906.

residue, and a general feeling of confidence in the ability to set foul roasts all right quickly in spite of fluctuations in natural draft.

Concentration.—There appears to be no great advance made this year in the concentration of gold ores. The difficulty remains that an attempt to concentrate out refractory particles from an ore usually leaves a tailing practically as refractory as the original ore.

It was hoped that fine sliming and the Wilfley slime-table might overcome this difficulty, but the latter machine is by no means perfect and maintenance is *a* most formidable item. So far the old canvas tables or frames appear to give the best results, but the cost of washing down is very heavy indeed—amounting in certain cases to 30s. or 40s. per ton of concentrate produced. A continuous rough rubber or canvas-belt table, somewhat on the Luhrig or Buss system, might solve the difficulty, though the capacity per unit would necessarily be small; or the old treatment table could have a traveling system of washing pipes worked by a water balance, and thus avoid the expense of so much boy-labor.

The flotation processes are being mainly applied to zinc-lead concentrate at Broken Hill, rather than gold ores. De Bavay and Simpson appear to have made an interesting discovery as to the principles involved in the various flotation processes, which they claim depend on the surface tension of sulphides as contrasted with the gangue, the greasy sulphides when wetted retaining an envelope of air that is expanded by heat, or by vacuum, or aided by the air entangled in oil, or gelatinous silica, or by flowing the mineral pulp in very thin layers—practically all surface—over inclined tables.

Treatment of Difficult Ores.—Cupriferous tailing with a copper content not exceeding 0.5% is now successfully treated by leaching out the copper with dilute sulphuric acid and precipitating on scrap iron, as described by W. S. Brown.† With ores containing more copper—but not rich enough to smelt—a preliminary roasting of the ore has been found to improve matters. The addition of ammonia to the cyanide solution has been previously suggested and carried out in practice.

Auriferous antimonial concentrate, containing say 20% antimony and arsenic, has been successfully treated by an ordi-

†*Proceedings* Institution of Mining & Metallurgy, 1906.

nary careful roast of the ore previously mixed with from 2 to 5% charcoal or coal, followed by a hot acid wash of dilute hydrochloric acid (obtainable by exposing old chlorine solutions to direct sunlight), and then lixiviating with cyanide or chlorine. A variation successfully employed is to roast as above, add salt at the end of the roast, and chlorinate. Both methods have yielded 90% extractions, but have the draw-back of requiring very careful roasting.

Slime Treatment.—The brothers Denny have made something of a sensation at Johannesburg by the methods they have applied at their new plants. In these they have adopted the West Australian method of circulating dilute cyanide solutions through the mill and filter-pressing their slime in Dehne hydraulic-closed filter-presses. They claim a recovery of over 94% at a treatment cost, including cyanide, filter-pressing, and disposal of residue (5d.), of 1s. 10d. per ton. As this carrying into practice of their proposed treatment scheme was not effected without many prophecies of failure, and as the cost of the installation necessary appears to be less than half that of the method of treatment locally in vogue, it is natural that their work should have received much attention, and as their figures do not seem to be seriously contested, the Denny brothers certainly look like scoring heavily again as the result of their enterprise and foresight, just as they did previously by the introduction on the Rand of tube-mills.

But perhaps the two greatest successes in slime treatment—as being a real advance on established filter-press practice—are the respective methods evolved by Ridgway at the Boulder and by Barry at Waihi. Anyone reading Mr. R. Gilman Brown's article* on the Moore filter, must have noticed the very sloppy, badly arranged frames shown in the illustrations. We can understand, perhaps, from these illustrations, one of the reasons for the small success attending the Moore filter in America. At Waihi, Barry has a frame¶ which is much more effective than that shown in the description referred to above, and he has successfully treated many thousands of tons by his method of open-framed atmospheric filtration; but the drawback to this—as to the Moore basket-filter—is the necessity of having men constantly in attendance to clean the frames.

*Mining and Scientific Press, September 8, 1906.
¶*The Mining Journal*, September 30, 1906.

Charles Butters seems to have been alive to the defects of the Moore frame, and has got out one that he appears to be working with considerable success at Virginia City, as does also F. L. Bosqui at Tonopah. Butters, however, has apparently allied himself to the Cassel enclosed type of press, which looks similar to that referred to in a previous paper as being experimented with at Johannesburg unsuccessfully.

The Ridgway machine has now been in use for a year at the Great Boulder, and has treated some thousands of tons of slime quite automatically and without any continuous supervision, and Hamilton, who has been testing it alongside his filter-press installation, is so satisfied with the results that he is laying down a plant to treat 500 tons per day — the largest slime plant in the southern hemisphere. The principle of the machine is a number of horizontal plates revolving round a central vertical post, which is really a tube, and to which pipes communicating with each plate are attached. Each plate has an under-filtering surface, and through a portion of its revolution it dips into the slime-pulp. A cake is formed by the application of a vacuum, and the plate in the course of its journey around its axis finds itself next in a water bath, where it remains sufficiently long for thorough washing. The cake is then dislodged automatically into the residue deposit system. This machine is certainly cheaper than the filter-presses, both in first cost and in operation, and an equipment to do the duty of an African decantation plant should not only yield higher recoveries at a lower working cost, but should be installed for less than half the expense of the existing system.

Of the other methods referred to last year but little has been heard. C. W. Merrill appears to be still at work on his hydraulic emptying filter-press, but details of successful work have not yet reached me. Philip Argall has a fixed frame and movable tank, but appears too busy at Cripple Creek to be able to give much attention to his slime-filter for the present.

In Western Australia one of the groups had a method of upward percolation of the solutions through the slime-pulp during agitation, but this method does not appear to have been attended with any great success. Generally speaking, it looks as if some direct method of automatic atmospheric filtration would displace both decantation and Dehne filter-pressing, although at the outset it will probably require a higher degree of intelligence for its effective and successful working.

Re-treatment of Tailing.—Not much has been said of the Stark process during the year. It seems to have been most profitably applied to the Crown Reef dump. Is it that there is any special characteristic feature of this dump that makes it suitable for the Stark process?

Mercuric Cyanide.—A reference to a test by Butters, on El Oro ore, of a mercury salt added to the cyanide solution, recalls the investigation made by the Cassel company (the owners of the caynide patents) in 1895 into the use of this salt. They obtained improved results, on some ores, of 3½% greater extraction of the gold and 3% greater extraction of the silver, with a lower consumption of cyanide, but the tests were not continued, as the improved extractions seemed insufficiently encouraging in view of the additional expense of the added salt; but Butters' tests have renewed interest in the subject, and we hope for fuller data.

Costs.—Tube-milling, fine grinding, milling, and roasting costs have been given above. The following tabulated statement of West Australian costs will show the progress still being made in that region by comparison with those in preceding papers, as well with those obtaining elsewhere:

Rock-Breaking.

Mill.	Pence.
Lake View	1.71
Ivanhoe	1.87
South Kalgurli (dry)	3.39

Milling.

Lake View	1s. 9d.
Ivanhoe	1s. 9d.
South Kalgurli (dry)	2s. 1½d.

Concentrating.

Lake View	6s. 7d. per ton concentrated 9d. per ton milled
Ivanhoe	8s. 0d. per ton concentrated 10d. per ton milled

Roasting.

Lake View	3s. 10d.	per ton	roasted	(concentrate	only)
Ivanhoe	5s. 5d.	"	"	"	"
South Kalgurli	2s. 6d.	"	"	(all the ore)	
Great Boulder Proprietary	2s. 4d.	"	"		

Fine Grinding Sand.

Lake View	1s. 10d.	per ton	ground
Ivanhoe	8d.	"	"
South Kalgurli	1s. 3d.	"	"

Cyaniding by Agitation.

Lake View	3s. 1d.	(includes 1s. 1d. for KCy and 1s. 2d. for BrCy)
Ivanhoe	4s. 1d.	(includes 11d. KCy, 2s. 6d. BrCy, and royalty 2d.)
South Kalgurli	1s. 1d.	(includes 7d. KCy.)

From the above it appears that the agitation treatment, less cyanide and bromo-cyanide, but including power, lime, labor, and supplies, costs roughly 7d. per ton (Ivanhoe 6d.).

Cyaniding by Percolation.

Ivanhoe	2s. 2d.	(including 9d. for KCy and lime)
Great Fingall	11½d.	(including 6d. for KCy and lime)

Filter-Pressing.

Lake View	1s. 7d.
Ivanhoe	1s. 6d.
South Kalgurli	1s. 6d. per ton filter-pressed.

The cost of filter-cloths at the Lake View Consols has been under ½d. per ton pressed for every month this year, save once when it was under ¾d.

Total Treatment Costs.

Ivanhoe	9s. 0d.
South Kalgurli	11s. 3d.
Great Boulder Proprietary	11s. 6d.
Great Fingall	6s. 11d.
Sons of Gwalia	5s. 2d. per ton treated.

General.—The battle of the processes at Kalgoorlie is now over. It is admitted that the all-roasting process gives the most profitable extraction, but the good fight made by the wet-crushing bromo-cyanide party, and notably by the Ivanhoe, has been of the greatest service to the industry, and the Ivanhoe's costs are such as to reflect great credit on the management and the staff; but the fine showing made by the South Kalgurli, the Great Boulder, the Kalgurli, and other companies, appears now to have convinced even the former advocates of bromo-cyanide.

TUBE-MILL LINING

(January 5, 1907)

The Editor:

Sir—I was very much interested in Mr. Drucker's experiences with tube-mill linings in Korea, as described in your issue of November 17, and thought a few notes from our experience of the past two years at Bodie, California, might add a point or two.

The tailing-flow from the stamp-mill of the Standard Con. Mining Co., passes directly through classifiers. The tube-mill is used to re-grind the underflow, or coarser material from the classifiers; with which is also fed dry tailing from the old slum ponds. The tube-mill used is an Allis-Chalmers pattern, about 22 ft. long by 5 ft. diam., as illustrated by the accompanying photograph.

The first lining used in the mill consisted of special steel plates about 10 by 12 in. area by one inch thick, with a base made so that the entire lining was intended to be interlocking. These plates had a life of four or five months, but as is often the case with castings subjected to grinding action, some of them would wear unevenly and occasionally one would drop out of place. This necessitated stopping the mill and replacing the loosened plates.

The next lining used was in the form of pine blocks, cut from 4 by 6 in. sawed timber into 6-in. lengths, trimmed slightly wedge-shaped (to allow for curvature of shell), and set on end. To place a new set of these pine-block liners, including removal of old blocks, and shoveling out and in of the load of pebbles, required usually about 36 hr. (The tube has only one man-hole, the cover of which is shown in the photograph.) A set of these liners lasted from 10 days to two weeks. Aside from its short life, this lining had the disadvantage (by reason of the vegetable oils contained) of making a soapy mixture with the alkalies present in the cyanide solution (crushing being done in solution). This saponification caused excessive foaming in the batteries and on the plates of the stamp-mill (for after going through the zinc-boxes the solution was re-standardized and returned to the battery storage vat), and also in the tube-mill, launders, and vats in the cyanide plant. After a few sets of these pine liners had been tried, a similar set was made out of blocks of mountain mahogany, a dense, hard wood from the Sierra Nevada mountains. This gave a few days longer life, and the saponification was decreased.

Next we tried a quartz lining similar to the one Mr. Drucker describes, except that there were no steel rails on the inside of the shell. The quartz consisted of selected pieces of chalcedonic vein-stuff, very hard and close-grained, from the mine. This lining was carefully put in with portland cement, and then allowed to set for about 10 days. In less than half an hour after starting up, the pounding of the pebbles caused the lining to break. This 'hasty finish' was, at the time, imputed to the quality of the cement used and to the fact that it was not allowed longer to set. Mr. Drucker's experience, however, shows that it was not altogether due to such causes. We next tried silex lining; but while this did not take quite as long to place and for its cement to set, as did the quartz lining, it did not give as long a life as was expected. The silex lining, also, acted similarly to the first steel-plate liners used, giving uneven wear, with occasional dropping out of blocks.

Then, it was decided to try wrought-iron bars, 8 in. wide by 1 in. thick, a quantity of which were on hand, having been used as connecting straps or plates at the joints in the old Cornish pump in the Lent shaft of the same company. It was first suggested to bend into circles to fit the inner surface of the tube-mill shell; but as we had no machine-shop rolls, and the bending would have to be done by hand in the blacksmith shop, that idea was dropped. The bars were cut in 15 and 7-ft. lengths (15 ft. being the longest that could be handled through the man hole of the tube-mill) These were put in, alternating (so as to break joint), lengthwise of the tube, and bolted with counter-sunk bolts that passed through the shell. The nuts and washers on these bolts are shown by the photograph. The bolt-holes in each strap are 'staggered;' and it was not necessary to space them closely, because of the length of the pieces. The liners were all drilled to the same measurements; and, in putting in the first set, holes were drilled in the tube-mill shell to correspond, subsequent sets not requiring further drilling of the shell.

As stated above, these straps or bars were of wrought iron. They give a life of between three and four months, and are easy to change—the entire operation of shoveling the load of pebbles, unbolting old, and bolting new liners, requiring only between 20 and 24 hr. The width of the straps, 8 in., does not allow them to fit closely to the curve of the shell, but that is an advantage, as it has a tendency to turn the pebbles over on themselves, instead of sliding, which action wears out the liners rapidly. This mill has been grinding from 50 to 75 tons daily. The portion of the stamp-mill product requiring re-grinding is principally a tough, hard, chalcedonic quartz.

WALTER W. BRADLEY.

Berkeley, December 10.

CYANIDE CLEAN-UP

By James E. Thomas

(January 12, 1907)

*In designing a milling plant it has always been the practice to erect a separate room for cleaning and pressing amalgam, and for treating any rich product, such as die sand, etc., and I think it is still more important that a separate room or floor should be set aside for the corresponding operations in a cyanide plant, as the liability of losses occurring in handling gold slime is much greater than in handling amalgam.

I therefore propose bringing forward a few suggestions with the view of evoking discussion on the subject — too long neglected — so that the engineers who design these plants may have a fairly definite idea of what is really required.

The clean-up floor should have a drainage separate from the rest of the extractor-house floor, so that any washings shall not be contaminated with oil and grease from the pumps, etc., and may be run into a sump from which the residue, after settlement, may be easily collected and included in the next clean-up. Means of transport of the gold-bearing zinc, etc., from the extractor-boxes to the acid-tubs should be somewhat better than those afforded by the proverbial Kaffir and a bath or bucket. Much time and also labor is wasted by that method. An overhead crawl with tipping buckets could, I imagine, be easily installed in most plants and would soon pay for itself by the saving of time and labor, not to mention the decrease in liability of having gold slime accidentally spilt on the extractor-house floor, whence it is apt to be carried by the traffic to the outside of the building and lost.

The acid-treatment plant is often totally inadequate for the proper dissolving of the zinc and insufficient allowance made for the washing of the gold slime before filter-pressing. I find it better to have two acid-treatment tubs of a certain capacity than to have only one of the same total capacity, as, when two are in use, one may be fed while the other is suffering from a threatened attack of 'boiling over,' while, when only one is installed, nothing can be done until the attack has succumbed to the 'cold water cure.'

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*Abstracted from *The Journal* of the Chemical, Metallurgical & Mining Society of South Africa, October, 1906.

A stoppage at the acid-tubs throws back the whole clean-up, and the time thus lost cannot be made up.

When filter-pressing is begun I have always found that it may be much more quickly accomplished when the gold slime is drawn from the bottom of the washing vat than when the suction from the filter-press pump is slung over the side of the vat. As the gold slime is very heavy, it is advisable to use a specially short length of hose for the filter-press pump suction. If, when running down the charges from the acid-treatment tubs, everything is put through a 64-mesh screen and the contents of the washing vat kept in agitation while the press is being filled, there is no danger of the hose over the suction drain-pipe becoming choked.

The use of filter papers over the cloths of the press is also to be recommended, as filling is thereby facilitated and the life of the cloths is prolonged, as no scrubbing is required to clean them.

The filter-presses supplied have invariably, in my experience, the openings to the hollow frames left rough cored and with a quantity of fused molding sand blocking the passages. By having these openings filed out so that the combined area, in a full press, is equal to or greater than that of the filling channel and with a slight increase of area toward the inside of the frames, there will be less liability of their choking, and the press may be filled in about 30% less time than when they are left rough.

For washing the cakes while in the press it is advisable to use hot water, not only for its better washing effect, but also because the cakes can be got out dry, even if there is only a layer of gold slime on the papers or cloths. A 400-gal. tank placed over the flue of the calcining furnace, if the latter is fairly close at hand, so as to utilize the heat from the escaping gases, will generally be found capable of giving enough hot water.

It is essential that a vat large enough to take all the washes from the washing vat and the filter-press be installed, so that the washes may receive further treatment before being run to waste.

Gold has been found to be present in solution in the acid washes. This is apparently due to the presence of hydrocyanic acid. The more completely the acid treatment is carried out, that is, when no zinc is left undissolved, the more danger there is of gold being redissolved. It is therefore advisable to have the washings slightly acid and then sprinkle zinc fume over the surface, meanwhile keeping the solution agitated. This should be done each time a charge from the acid-tubs is transferred for washing. After the clean-up is over it is advisable to have the contents of the large vat for the washings assayed, after good settlement, and, if necessary, more zinc fume can be added before the solution is run to waste. By this means the gold in acid washes run to waste may be reduced to 0.07 dwt. per ton, whereas the ordinary acid washes will probably carry 1.5 dwt., or more, per ton. Copper sulphate, finely divided iron, and charcoal were also used for this purpose in the laboratory experiments, but, as zinc fume gave the most satisfactory results, it was tried on a working scale with such success that it has been used on all the mines of the Consolidated Gold Fields group ever since a supply could be procured. It is prepared by the distillation of zinc *in vacuo* and sifted and is packed in air-tight cases, as it oxidizes readily on exposure to the atmosphere and is afterward, of course, useless for this purpose. From five to seven pounds are sufficient to treat washes amounting to about 100 tons.

For taking off the acid washes from the washing vat it is advisable to use a decanter. If the arm of the decanter be painted white, the operator can see at a glance whether the solution is clear enough to draw off or not. The paint will last through a clean-up and can be easily renewed. The washing-vat may have a false bottom of cement sloping toward the outlet to enable it to be more easily cleaned out. The cement should be painted over with P. & B. acid-proof paint after each clean-up. If this is done it will show no signs of wear. When the paint is being applied it is as well to keep an eye on the person doing the job, as I have had a Kaffir overcome with the fumes from the paint when at work on the bottom of a vat 10 ft. deep. He quickly recovered on being taken to the fresh air.

Stirring gear for the acid-dissolving tubs should be installed, but the motion should be reversible, as it is often found expedient to reverse the direction of stirring in order to help the dissolving of the zinc and to free any which may have collected round the upright carrying the paddles.

THE MOORE AND BUTTERS FILTERS

(January 12, 1907)

The Editor:

Sir—The article by Mr. Nutter on the Moore filter in your issue of December 15 is both interesting and instructive. There are, however, several serious errors in it which do an injustice to the Butters filter, and which should be corrected immediately. In discussing a plant to treat 350 tons of slime, as described by Mr. Nutter, a comparison with the plant erected (at Millers, in Nevada) for the Tonopah company, will be in order. This Butters slime-filter consists of 216 leaves, and would handle about 300 tons of the Liberty Bell slime, and 250 leaves would easily handle the mill output of 350 tons per day. Such a plant would require instead of eight 7-in. pumps as estimated by Mr. Nutter, only one pump that would transfer the pulp or water in 10 minutes. It is very difficult to calculate centrifugal pump capacities working on a material of such variable consistence as slime, but judging from previous experience with different slimes a 9-in. pump would be amply large; and as 48-in. pumps are regularly used to transfer sand in dredging, it is hardly in order to call a 9-in. pump "large," or to consider it or its valves difficult of operation.

The light crane mentioned by Mr. Nutter would not be necessary, as a half-ton chain-block, with crawl, would handle the Butters leaves when such handling became necessary, which, by the way, would be rare.

As regards the quantity of slime to be pumped, it is certainly not fair to require the handling of the large quantity needed to fill Mr. Nutter's filter-vats, which are not designed as is the Butters filter-box, to contain barely enough slime to cover the leaves. Neither is it a fair proposition to require that the slime be pumped against the maximum head of 19 ft., when a properly designed plant, even when built on level ground, has a maximum lift of 10 ft. and a minimum of 0 ft. In other words, one properly designed box for 250 Butters leaves will contain approximately 7,000 cu. ft. instead of the 24,000 cu. ft. capacity of the eight filter-vats that Mr. Nutter considers necessary. The use of such a box would make available for water and slime storage the vats now used for filtering and washing, thus obviating the necessity of installing the two storage tanks he mentions.

Instead of the time items being as given by Mr. Nutter in the cycle of operations, the following in the opinion of the writer, would more nearly approximate actual running conditions:

Operations.	Moore.	Butters.	Butters gravity.
Loading	1 hr.	1 hr.	1 hr.
Transferring (filters) .	10 min. (pulp)	10 min.	2 min.
Milling with water . .	Nil	10 "	5 "
Washing	1 hr.	1 hr.	1 hr.
Discharging	10 min.	10 min.	10 min.
Transferring (filters)	10 " (pulp)	10 "	2 "
Total	2 hr. 30 min.	2 hr. 40 min.	2 hr. 19 min.

Without knowing the conditions as they exist at the Liberty Bell that would influence the cost of a filter installation, such as present arrangement of vats and tanks, power required by the Moore filter, ground contours, etc., I am unable to carry this comparison to its logical conclusion, wherefore it is incomplete.

A statement from Mr. Nutter as to why it would not be better to install a properly designed filter-box; why a gravity plant could not be placed; and why one 9-in. pump would not be sufficient, would seem to be in order.

MARK R. LAMB.

Mexico City, December 22.

A SIPHON DEVICE FOR REMOVING FLOATING MATERIAL

By Edward S. Wiard

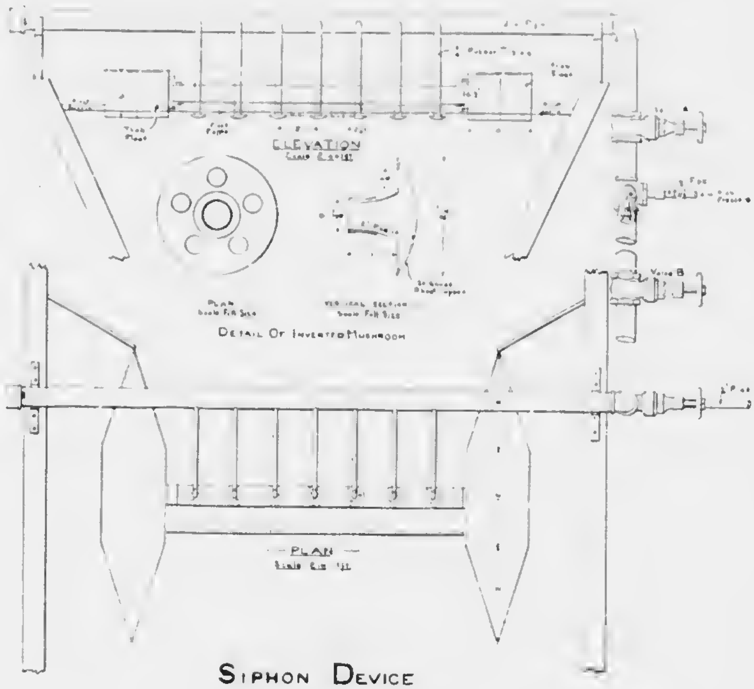
(February 2, 1907)

The device described and illustrated in this article was used by the writer in a mill in northern Idaho to remove particles of floating galena from the surface of the water in a long V-shaped tank, distributing thickened pulp to a number of Wilfley tables and vanners. Owing to the manner of mining, the milling ore contained an extraordinary amount of wood, causing more than the usual trouble of stopping up plugs and clogging the riffles of the Wilfley tables. An examination of the floating material showed it to consist largely of fine bits of wood to which had adhered scaly pieces of galena; the adhesion of the two being possibly assisted by a film of grease on the wood. The galena was rather richer in silver than the regular run of concentrate; tests with silver-leaching salts and an examination with the microscope showing the increase over the normal proportion to be due to tetrahedrite. This mineral was in minor amount in the ore as it came to the mill, the predominant silver mineral being argentite. A little galena was lost by greasy flotation. The undressed ore contained large amounts of spathic iron and the tank-water was slightly acid. Apparently, the top layers of the water in the tank were perfectly clear but the insertion of a baffle-board would cause a black scum to dam up behind it, constantly being augmented, and spreading over the surface of the water. The galena could readily be separated from the wood by agitation; after shaking a portion of water and scum in a beaker, the wood would rise to the top and the galena settle to the bottom.

The water from the siphon was used to wash down the concentrate formed by the vanners. Clean water was scarce and, it being necessary to make a concentrate high in lead, it was not possible to use the overflow water of the tank for sluicing purposes. The overflow of the tank contained a noteworthy quantity of fine matter, which was further settled on the lower floor.

The device, herewith illustrated, is suggestive of a means of reducing the volume of water flowing through a tank. The slime and sand treated in the mill under discussion were rather free from clayey matter and the surface water but a few feet from the feed end of the tank was very nearly clear. The device floated in the water about two-thirds the way from the feed end to the overflow end, but it would have been possible to have placed it much closer to the feed end of the tank and obtain water free from low-grade slime. For the purpose of merely removing comparatively clean water, a number of the devices could have been used.

The two float-tanks should properly be made of thin sheet aluminium. My floats were made of No. 30 sheet iron heavily coated with bitumen. At the top the sides are bent over a heavy wire, bent to the shape of the tank, to give stiffness. The rubber tubing used was ¾ in. outside diameter, with walls $^3/_{32}$ in. thick. The tubing should be thin-walled, so as not to interfere with the free vertical movements of the floats. The bracket between the floats, supporting the tank-end of the tubes and the inverted mushrooms, is made of light strips of wood stiffened by four little angle-irons in the corners. These can be quickly made and bored and countersunk for wood screws by the mill-blacksmith. Care should be taken to make the upper row of holes in the bracket vertically above the corresponding lower ones. These holes should be of a little less diameter than the tubing; the tubing will then be firmly held and yet adjustable. The inverted mushrooms or strainers are made of No. 24 copper plate. With the form of strainer used, the pull due to the siphon action drew the top layer of water downward and in from the sides firmly and gently. With a simple tube in the water a current would be created from the bottom of the tank that would stir up the low-grade slime below the upper layers of water The nipples on the 1½-in. pipe are short pieces of pipe screwed into the sides. The tanks are prevented from moving from their positions by strings fastened from their points to the nipple pipe; they are not shown in the drawing. The little strips of wood from the points of the floats to the sides of the tank are in the nature of baffles to cause the floating material to converge toward the mushrooms and prevent its passage between the floats and the sides of the tank. A similar strip floats behind the mushrooms to prevent floating material from passing by them.



SIPHON DEVICE
for
REMOVING SLIME.

Fig. 25.

The siphon device is started by opening the valve from the high-pressure supply; after it is started, the valve is closed and regulation is effected by the valve *A*. To avoid a climb to the tank, a second valve (*B*) was placed on the vanner floor for starting or stopping the siphon.

THE BUTTERS FILTER

By Mark R. Lamb

(February 2, 1907

The accompanying illustration shows a Butters filter with the necessary tanks and pumps. The filter is extremely simple and consists merely of two sheets of canvas sewn to a core of cocoa-matting. These resultant leaves or cells are supported by a wooden strip at the top and by a frame of half-inch pipe at the sides and bottom. The cells are connected by short nipples and hose to the vacuum-drum. When the filter-box is filled with pulp, which has been previously agitated with solution, the vacuum draws the solution from the slime, leaving the latter on the leaves in the form of a layer. This layer is made from 0.75 to 1.5 in. thick, depending on the class of slime. During the time the slime-cake is being formed the filter-box is kept full of pulp, the additions replacing the solution withdrawn through the cells and keeping the slime agitated.

The layer forms continuously and, as would be expected from the nature of the operation, its thickness varies from top to bottom, depending upon the permeability of the stuff deposited. In other words, the slime resulting from all-sliming, which contains fine sand, forms a thicker layer on the lower portion of the leaf than on the upper. This thicker part is more permeable and is washed in exactly the same time as the thinner but denser upper part.

After the layer of slime has attained the desired thickness, the pulp remaining in the box is drawn off to the pulp-sump, and the box is filled with water. This water is drawn through the slime until the latter is thoroughly freed from solution and metal in solution. The vacuum is then broken and clear water introduced under slight pressure into the interior of the leaves. This dislodges the slime which is discharged from the hopper bottom through a gate-valve. Where it is desirable, a large portion of the water remaining in the box can be withdrawn before the slime is discharged, and if water is very expensive the slime can be dislodged with compressed air and trammed from the filter-box. The small centrifugal pump returns the pregnant solution to zinc or fume or electrolytic precipitation boxes. The large centrifugal pump returns the excess of unfiltered slime to its storage vat. All valve-stems are extended to the operating floor, which permits one man to treat almost any tonnage. The canvas does not seem to deteriorate, one set having been in service over two years. The lime used in neutralizing acidity appears to protect the fibre.

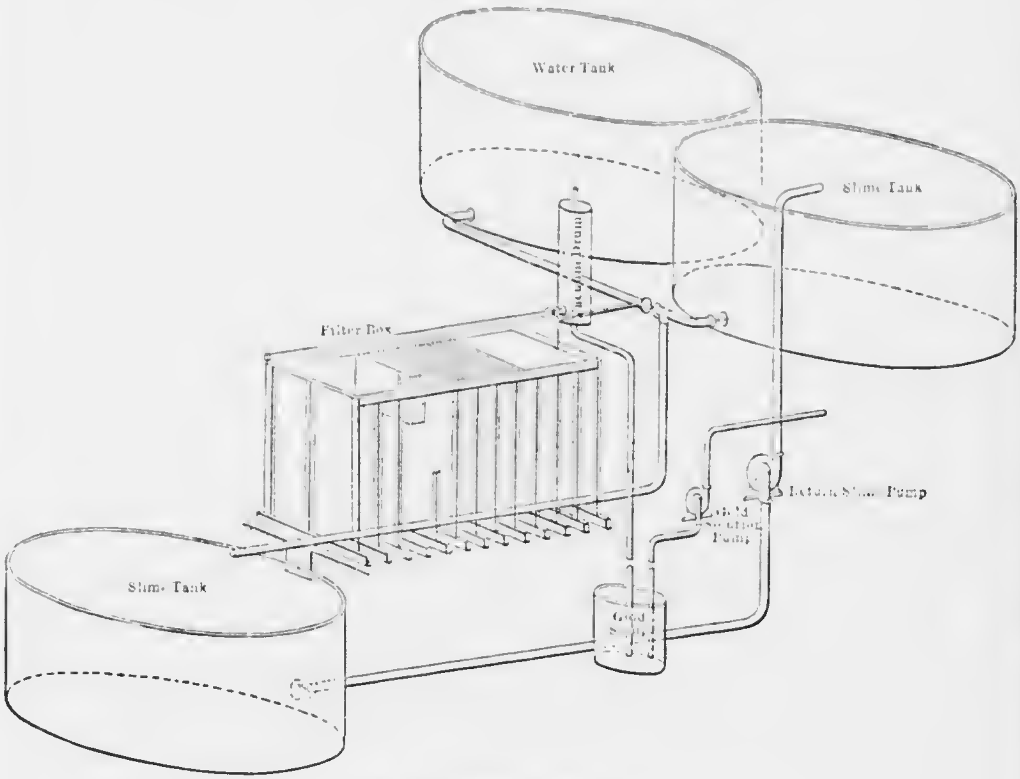


Fig. 26. Butters Filter.

When this calcium carbonate accumulates sufficiently to delay operations, it is removed by subjecting the leaves to a bath of 2% HCl.

The filter-box may be made either of wood or iron, as may seem advisable, depending on the locality, costs, and freight. Where the site does not permit of filling and emptying the box by gravity, the pulp is circulated and transferred by the pump. The valves in such a system are all placed near the pump and controlled from the operating platform above mentioned. The wash attained by the filter is notably perfect and is accomplished by the use of a minimum of water.

Slime is now being treated, resulting from the milling of Tonopah ore, which it would not be possible to treat by decantation, and for which a filter-press installation would be prohibitively expensive, as well in first cost as in labor and power. This filter combines the best features of the Moore, Butters, and Cassel patents and will be the means of extracting precious metal from slime not amenable to treatment by any other method, as well as from sand now treated by percolation, but which will be ground much finer and treated by agitation and filtration.

THE RIDGWAY FILTER

(February 9, 1907)

*This invention is designed particularly for treating slime containing gold in solution. A trial machine was built on the Great Boulder mine, at Kalgoorlie, and has been constantly in operation since January, 1906.

This machine consists of 12 flat cast-iron filtering frames (T), which are in the form of sectors of a circle. The frames are corrugated on their under surfaces, and to these corrugated surfaces screens are attached; ordinary filter-cloth is fixed over the screens. The frames are suspended horizontally from radially arranged levers (X), the inner ends of which are connected to a vertical centre column or spindle, provided with internal compartments. Each filtering frame is also connected by three radial pipes and rubber hoses to different compartments in the central column.

The outer edges of the filtering frames form the periphery of a circle 12 ft. in diameter; immediately outside this periphery is placed a circular rail track (Z), and to the outer end of the levers (X) are attached wheels or rollers (T^1), of 4 in. diam. The wheels support the levers and run on the circular track (Z), which is carried on the outer edge of an annular trough under the filtering frames. The trough is divided into three compartments—one for slime-pulp, one for wash-solution, and one (without a bottom) for residue discharge. The general level of the track (Z) is such that the frame has its lower surface only immersed; the track, however, has elevated portions so arranged that when the central columns and frames are rotated, the wheels, in running up the elevation, lift the filtering-frames out of one compartment, and, on descending, lower the frame into another.

The lower part of the central column has two passages, into which the solutions are drawn, by a vacuum, through the radial pipes that connect with the filtering frames—the vacuum being produced by a pump connected with the openings (G^1 and L) near the bottom of the column—air-tight connections between rotating and stationary parts being secured by means of glands and stuffing-boxes.

*From the *Monthly Journal* of the Chamber of Mines of Western Australia, November, 1906.

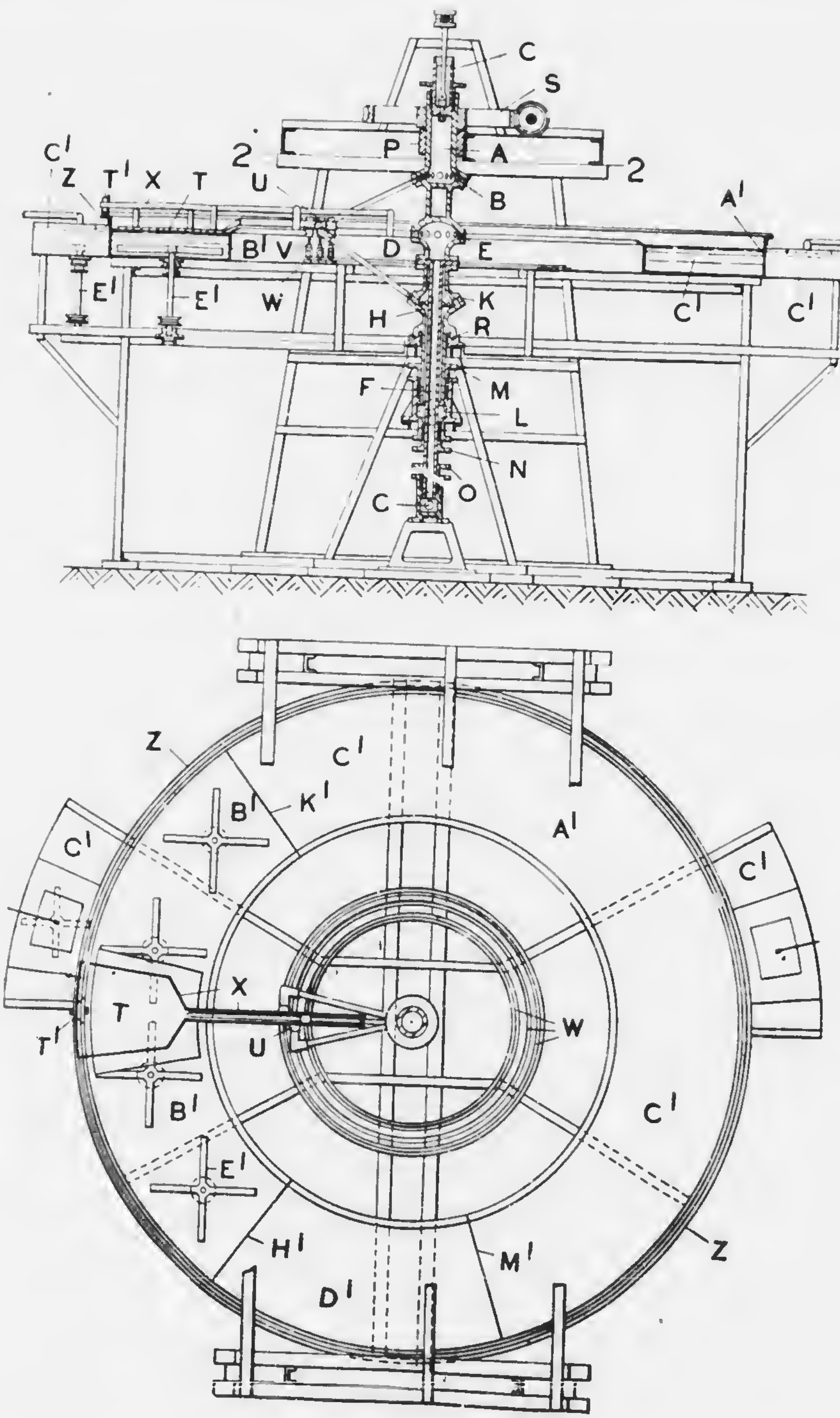


Fig. 27. The Ridgway Filter.

A nest of three valves (U) is placed in the pipes between the column and each filter-frame; each of the three valves is automatically operated while the machine is in motion by small rollers (V) passing over elevations on one or other of the three tracks shown at (W); the gold solution is thus delivered into one compartment of the central column through one of the valves, and the wash solution into the other compartment through another valve, thus providing for the two solutions being kept separate. A third pipe connects the third valve (at U) with the top or compressed air section of the column (at B), and while the frame is passing over the discharge chute this valve is automatically opened by its roller and a puff of compressed air enters the filter-frame and displaces the residue adhering to the cloth; the residue falls through the discharge chute (D^1) into a dump-truck. Each frame acts independently, having its own roller, set of valves, and connections with the central column; and each frame and its adjuncts form a complete self-acting unit. The accompanying drawing shows a single-plunger air-compressor (at C), self-contained and driven by the machine itself; the compressed air can, in many instances, be supplied conveniently from other sources.

The machine requires about $\frac{1}{2}$ h.p., and is driven (in the direction of the hands of a watch) with a 6-in. belt on a 42-in. pulley keyed to the top of the central column (at S), and makes one revolution per minute. The vacuum is on the filter-plate the whole time it is immersed whether in the pulp or in the solution wash. Each filter-frame is 13 seconds in the pulp (from B^1 to B^1), seven seconds on the elevated portion while passing from the slime to the wash compartment, 30 seconds passing through the wash (C^1 to C^1), seven seconds lifting and drying, and three seconds blowing residue off (at D^1). During the period in which the frame is passing from the pulp to the wash, the vacuum is on for one second only while the frame is rising, and one second only while it is descending into the wash, thus giving the pulp adhering to the underside of the frame an air-leaching of two seconds' duration. This period of leaching may be varied to suit the particular nature of the slime under treatment or the thickness to which it has formed on the cloth, which is usually about $\frac{3}{8}$ of an inch. The supply of pulp, wash-solution, or water into the annular troughs, (B^1 and C^1), is regulated by means of float-valves in the receiving compartments (G^1 and G^1); the pulp is kept from settling by the revolution of small agitators driven by a rope passing round pulleys attached to their spindles, which project through glands and stuffing-boxes in the bottom of the receiver (at E^1 and E^1), etc. By means of the float-valve in the pulp-receiver (G^1), the pulp supply is taken automatically from the pipes through which the Dehne presses are charged; thus, in order to compare the efficiency of extraction by the machine with that of the presses, a portion of the general pulp is taken as each Dehne press is filled.

From 0.8 to one ton of wash is used per ton of dry slime, and the residues contain from 28 to 33% moisture, and from nil to a trace of KCN and soluble gold. The tonnage treated in 24 hours and percentage of moisture contained in the residues vary according to the variations in the pulp. With clean quartz slime of a consistence of from 42 to 44% solid, and a fineness of about 92% through a screen having 200 holes per linear inch, and a vacuum of 20 in., this machine will treat 50 tons of dry slime per 24 hours, leaving a residual moisture of 30 per cent.

On a clay slime the output will decrease to 25 tons per 24 hours, and the residual moisture will increase 35%. A trial was made on the old battery slime; the original assay value was 7 dwt., the gold in residue 18 gr., moisture in residue 35%, soluble gold in residue nil; this was treated at the rate of 25 tons per 24 hours.

Each frame has four square feet of filtering area and about 0.8 square yard of cloth is used to cover each frame. The life of a cloth on slime from the Great Boulder roasted ore is about 14 days, or 700 tons of slime for 9.6 sq. yd. of cloth.

The vacuum pump has a piston sweep of 80 cu. ft. per min., and, with a 20-in. vacuum, requires 3½ h.p. The machine itself, without the self-acting air-compressor, requires about ½ h.p., and allowing 1 h.p. for the agitators in the pulp-trough and pulp-receiver, and for friction in the counter-shaft, a total of 5 h.p. is required to drive the equipment described.

As the machine is absolutely automatic in its entire operation, no additional labor is employed in running it. Taking into consideration that this is the first machine made of this pattern, the repairs and renewals have not been abnormal.

The working costs during the last three months have been:

	Pence.	
Filter-cloth	0 3	per ton
Horse-power	2 4	"
Repairs and renewals	1.0	"
Total	3 7	"

It has been impossible to segregate the labor cost on this machine (other than those included in repairs and renewals), as during the whole run any slight attention necessary, other than greasing, has been performed by the shift-boss on duty.

The method of operating the machine is as follows: The agitators in the pulp compartment of the annular trough are first set in motion, being driven independently of the machine; the pulp-compartment is filled to normal level with agitated pulp, and the wash-compartment with wash-solution. The vacuum pump is started, and the machine is set in motion by applying a tension pulley to the belt on the pulley attached to the top of the centre column of the machine, thus rotating the centre column, which imparts the motion to the filter-frames by the pipes and levers, causing the rollers (T^1) supporting the outer end of the levers from which the filter-frames are suspended, to travel on their track (Z). As the action of all the frames and their connections is similar, one only need be described. Assuming that a frame has just passed over the discharge chute and is on the elevated portion of the track, it then descends an incline on the track to the normal level, and the cloth on the underside of the frame is immersed in the pulp; at the same time, the gold solution valve (at U) is opened by a roller (at V) running up an incline on its track (W), and similarly to the action of the elevating rollers (at T^1) on the track (at Z), thus opening direct connection between the filter-frame and the vacuum which is maintained in the centre column; the vacuum draws the liquid through the cloth and leaves the solids adhering to it. The roller (T^1) on the outer end of the lever, while continuing on the level track, keeps the cloth submerged till another incline is reached, up which it then ascends, lifting the frame out of the pulp-trough; on descending the incline again to normal level, the frame is lowered into the wash-solution. When the frame ascended from the pulp, the gold-solution valve closed, and as it descended into the wash solution the wash-valve opened, causing the vacuum to draw the wash through the adhering pulp, which is usually about three eighths of an inch thick. The frame continues in the wash-trough during the travel of half the circle, and is then raised by similar means to those already described, but the wash-valve is not closed till the discharge chute is reached, thus allowing the vacuum to remove, by air-drying, some of the moisture from the cake of slime formed on the underside of the cloth. When the discharge chute is reached, the wash-valve closes, and the air-valve in connection with the upper portion (*B*) of central column is opened, admitting a puff of compressed air, which dislodges the cake from the cloth and precipitates it into the discharge chute (D^1), under which a dump truck is placed to receive residues. The compressed air required at the time of the discharge can be supplied in any usual way or by the self-contained arrangement shown in the drawing at *C*.

THE BUTTERS FILTER

(February 16, 1907)

The Editor:

Sir:—In a recent number of the PRESS*, Mr. Mark R. Lamb, of the Butters Co., has criticised a comparison I made† of the Moore and Butters filters as applied to conditions at the Liberty Bell mill. Mr. Lamb claims that my comparison was unjust and unfair to the Butters process, and goes on to say that instead of the one I made, a comparison with the Butters installation for the Tonopah Mining Co. at Millers, Nevada, would be in order.

Since reading Mr. Lamb's article I have obtained additional information covering the points raised, and in the light of that information I am forced to doubt Mr. Lamb's data and question his conclusions. Mr. Lamb does not state specifically that all of his data are from the plant at Millers, but, as that plant is one of the latest of the Butters installations, it presumably represents the best Butters practice, and it would seem fair to examine Mr. Lamb's statements and conclusions in the light of facts pertaining to that plant.

Mr. R. Chester Turner, superintendent for the Tonopah company, and Mr. A. R. Parsons, mill superintendent, have kindly furnished me information about the Millers plant, which is given below in parallel column with Mr. Lamb's statements:

	Lamb.	Turner & Parsons.
Capacity of filtering vats	Not given	9,130 cu. ft.
Size of filters	5 by 10 ft.‡	4 ft. 9 in. by 9 ft. 9 in.
Number of filters	216	192
Time required to transfer pulp	10 minutes	20 minutes
Time required to fill vats with water	10 "	20 "
Discharging cake	10 "	Not given
Running back water	Not given	20 "
Running in pulp	10 minutes	20 "
Maximum height of lift during transfer of pulp and water, by centrifugal pumps	10 feet	25 feet

It would have seemed in order for Mr. Lamb to have made sure of his own ground before attacking the statements I made.

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*MINING AND SCIENTIFIC PRESS, January 12, 1907, page 54.
†MINING AND SCIENTIFIC PRESS, December 15, 1906, page 714.
‡Given in personal letter to me.

In a personal letter he admits the fairness of my assumption, that, with the same slime, the loading and washing periods and the vacuum required, etc., can be figured the same for both processes. Let it be assumed then that Liberty Bell slime is to be handled by a plant built of the same units as at Millers. The treatment cycle would be as follows:

Loading	60 minutes
Running out pulp	20 "
Running in water	20 "
Washing cake	60 "
Discharging	15 "
Running out water	20 "
Running in pulp	20 "
Total	215 minutes
Total time of cycle	3 hr. 35 min.
Total number of cycles	6.7 per 24 hr.

Mr. Lamb states that 250 leaves or filters like the ones installed at Millers "would easily handle the mill output (Liberty Bell) of 350 tons per day." As stated in my previous article 45 changes of the Moore baskets are now necessary to handle that tonnage. Each basket has 66 filters six by eight feet, giving a filtering area of 6,336 sq. ft. per basket. With 45 charges, then, we would have 285,000 sq. ft. of cake formed. The 6.7 cycles of the Butters unit, under consideration, would represent 59,563 sq ft. of cake formed, as the 96 filters have an area of 8,890 sq. ft. To handle, then, the Liberty Bell tonnage would require 4.79 vats of the size at Millers, or 460 leaves instead of the 250 that Mr. Lamb gives as a sufficient number, or the 330 Moore leaves now doing the work.

Again Mr. Lamb states that one properly designed box for 250 Butters leaves will contain approximately 7,000 cu. ft. If this is so, why were not the Millers boxes properly designed? The two contain only 192 leaves and have a capacity of 9,130 cu. ft. These two boxes have an aggregate filtering surface of 17,780 sq. ft., which gives 0.51 cu. ft. of box per square foot of filter. The Moore boxes at the Liberty Bell have a capacity of 3,050 cu. ft. each, as stated in a previous article, and one basket has 6,336 sq. ft. of filter surface. This gives 0.48 cu. ft. of box per square foot of filter. It would seem from this that the Moore boxes at the Liberty Bell are rather better designed for Butters process work than are the boxes installed by Butters; and Mr. Lamb's contention that "it is certainly not fair to require the handling of the large quantity (of slime) needed to fill (the Liberty Bell) filter vats, which are not designed as in the Butters filter-box, to contain barely enough slime to cover the leaves," rather falls to the ground. These boxes are of the same general design, that is, they are rectangular boxes with hopper-bottoms, having a 60° slope.

Mr. Lamb advocates the use of one large filter-box for the Liberty Bell, instead of the units at present in use for the Moore process, and says that one 9-in. pump would probably handle the pulp in 10 minutes. For a box containing 250 Butters leaves, the capacity would be 11,800 cu. ft., if it were of the same design as the boxes at Millers, instead of the 7,000 cu. ft. given by Mr. Lamb. The amount of pulp and water to move would be about 1,900 cu. ft. less on account of the displacement of the loaded filters, giving 9,900 cu. ft. of material to handle. If we accept the statements of pump manufacturers—and the capacities of their pumps are probably not under-estimated—a 15-in. discharge pump would be necessary to move this volume of material in ten minutes, or a 12-in. pump in 20 minutes. If we put our 460 leaves in one box, a 15-in. pump would be necessary for moving the pulp in 20 min., and a 20-in. pump necessary to do it in 10 minutes.

His proposition to use a single large vat also involves the condition of maximum power consumption where power is bought on peak load.

Mr. Lamb's statement that "as 48-in. pumps are regularly used to transfer sand in dredging, it is hardly in order to call a 9-in.pump 'large' or to consider it or its valves difficult of operation" scarcely calls for comment.

It is safe to say that where a 48-in. pump is regularly used for dredging sand, it is not used in connection with a nest of 48-in. valves.

Mr. Lamb asks why a gravity plant could not be placed at the Liberty Bell. The answer is, for the same reason probably that it was not installed at Millers. It would cost more to excavate for it than it would be worth.

In a personal letter, Mr. F. L. Bosqui has pointed out wherein my former article was seriously incomplete, by not giving comparative cost data for filter-repairs and maintenance, for the Butters and Moore systems. Mr. Bosqui's criticism is entirely just. I could not give these data, as I have not got them. Such as I have on the Moore process, are vitiated by two factors, namely: At Bodie, the plant was run several weeks without re-grinding, and the heavy sand rapidly cut the filters to pieces, and raised the costs materially for the period covered by the data I have. At the Liberty Bell, the filters are being changed as rapidly as may be to a better type, in part similar to the ones at Bodie, and repairs make consequently a large item. Experience seems to indicate, though, that filter-repairs are less for the Butters than for the Moore process.

EDWARD H. NUTTER.

Telluride, January 30.

CYANIDATION AT COPALA, MEXICO

(March 16, 1907)

The Editor:

Sir—Enclosed please find diagram which is self explanatory. It was prepared for the purpose of comparing quickly

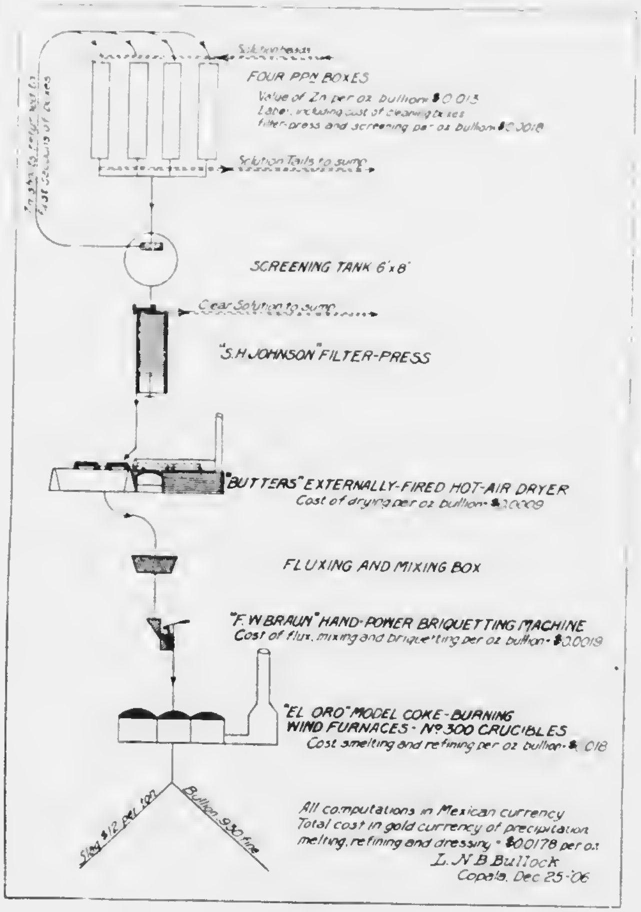


Fig. 28.

costs at the various stages and also to graphically describe the present system and compare it with others which have preceded it. You will notice that the acid treatment is entirely dispensed with, all zinc shorts being returned to the head sections of the boxes, not only affording us a marked saving in zinc consumed, but also greatly simplifying the process and saving us the cost of acid, extra filter-press, acid-pump, etc.

The precipitation boxes are of eight compartments with bottom-discharge valves. The slime is run into a screening tank through a No. 30 screen. The tank is made in two compartments, separated by a vertical division having its top six inches below the upper rim of the tank, so that when the first division is full the product overflows into the second (leaving, of course, nearly all the precipitates in the first section) and is then pumped through the filter-press. Both divisions have bottom connections to the filter. This arrangement allows the cleaning of all the boxes without interruption, for, as we very frequently get more than a cake of precipitate in three of the four boxes and as we have to clean all of them daily, without this device we would be compelled to empty the filter before we could complete the boxes. Of course, we open up the suction to the first section, when we are ready to make cake.

The precipitate, which contains from 18 to 20% moisture, then goes to the dryer and the moisture is reduced to 5%. This dryer consists of an iron cylinder 30 ft. long by 30 in. diam. having an external fire-box, which allows a uniform heating. A car 24 ft. in length and holding six portable steel trays, containing the precipitate, is run in on a track that also extends a like distance outside the shell to allow easy handling of the product. There is a vent at the far upper extremity of the shell to carry off steam, etc., and the entrance is closed with a steel-hinged door. This device has given every satisfaction, being effective and economical.

The precipitate then goes to the mixing-box, where the flux and binder are thoroughly incorporated and pressed into briquettes, which are then ready for the crucible. Only the No. 300 crucible is used, and a unit charge consists of 220 lb. precipitate with its flux. As the briquettes melt down, more are added, until the crucible has its full charge. *There is no dusting.* Four thousand three hundred ounces are refined as a charge and four charges are run in the same crucible within seven and one-half hours. Dip samples are taken.

The briquetting machine is manufactured by the F. W. Braun Co. and is really intended for the manufacture of cupels. But with a square mold and die, 2½ by 2 by 1½ in., it makes a clean, hard cake and will handle about 100 lb. per hour.

The precipitation-boxes, screening-tank, filter-press, and briquetting-machine are so placed as to allow a solid concrete floor draining to a small pump near the filter. The entire floor is thoroughly washed with a hose every evening and by a simple bye-pass on the suction this water is pumped through the press. The building is of brick, with white plaster walls extending to within four inches of the floor, where there is a cement collar. This makes it easy to detect any carelessness in handling the product.

The staff consist of one white foreman and five native boys. On account of the high charges for freighting between Mazatlan and the mines, coke costs us 69 *pesos* per ton; fluxes, zinc, crucibles, etc., are proportionately high, so that taking everything into consideration the cost per ounce is extremely low. We handle on an average about three tons of precipitate per month.

Although this method is far from new, I think there are two or three features of it that are certainly not in general use, namely:

1. Returning all zinc shorts to boxes and thus entirely dispensing with acid treatment.

2. Briquetting product with flux incorporated and smelting same without re-drying (they average 5% moisture).

3. The refining of the bullion in large crucibles, making unit charges of four and five 1,000-oz. bars.

The first has been in successful operation for ten months. By first laying, say, two inches of long zinc on the trays of the first section of the precipitation-boxes, then lightly laying the short zinc on top to a depth of 1½ in. and alternating this until the section is half filled, and finally filling entirely with carefully placed long zinc, we have attained the desired result. All the zinc is consumed and precipitation is perfect.

The briquetting partly does away with the expense of calcining, as the product is dried to only 5% moisture. This product is in every way safer and cleaner to handle and there is no spitting or dusting of the product in the crucibles. The capacity of the latter is likewise greatly increased, unless the comparatively unsafe practice of adding the dry calcined powder to same is followed, which, even when the paper bags are used, allows more or less loss by dusting, etc. In the case of briquettes we add to the crucible without any danger until it is as full as desired.

Mr. Edwin Burt, superintendent of cyaniding for El Oro Mining & Railway Co., Ltd., will bear me out in all of the above, as it was he who first put it into practice in Mexico, inventing his own briquetting-machine, a very ingenious device; the abovementioned company also now briquettes all of its product.

As for No. 3, this means less work and a very large saving in fuel.

I shall finish by saying that the practice here is very satisfactory, the equipment is complete, and the results are certainly economical. It is hoped that you will make use of whatever you see in this short sketch that may prove of value to others, and I shall be glad to read any discussion on the subject.

LAURENCE N. B. BULLOCK.

Copala, Sinaloa, Mexico, January 12.

THE BUTTERS FILTER

(March 23, 1907)

The Editor:

Sir:—The discussion in regard to the Moore and Butters filter-processes has become most interesting to me and probably also to others interested in the treatment of slime. The following data may be of use to the profession; they refer to the Combination mill at Goldfield, Nevada.

The tanks and apparatus in use are as follows: Two slime storage vats, two wash-water vats, the filter box and canvas frames, one acid-box for the washing of frames, one vacuum-pump, one four-inch Butters centrifugal slime pump, one two-inch Krogh centrifugal pump, valves, and piping.

The pulp after agitation, containing 60% of a 1.5-lb. solution of cyanide, is discharged into the slime-storage vats (13.65 ft. diam. by 12 ft. high with slightly conical bottoms) ready for the filter-box. Two of these vats are in use, each one containing a mechanical stirring apparatus with paddles, keeping the pulp of the same consistence throughout. One vat of this capacity will easily hold three charges for the filter-box (the size of which is specified further on), and the two used in conjunction will hold about eight charges, all depending on the thickness of the pulp and cake to be formed.

The storage-vats, filter-box, wash-water vats, and vacuum pump are all on the same level. The pump used in connection with the filter-box is placed about six feet below the vats, which simplifies the piping and operation of the valves. This four-inch centrifugal pump (patented by Mr. Butters) is especially adapted to the handling of pulp containing slime or fine sand, and has given little trouble to the operator or mechanic; it is the least troublesome pump in the mill.

The inside top measurements of the filter-box are 9 ft. 11¼ in. by 10 ft. 2½ in., the sides extend 5 ft. 9½ in. above the cone, and from the discharge gate to the top is just 11 ft. 7⅜ in., with a capacity of about 785 cu. ft. when not containing the frames. The frames are 116⅛ by 56¼ in., each having about 90 sq. ft. of filtering surface. There are 27 of these frames; six sets of four, and one of three. The four frames are connected at the bottom by ¾-in. rubber hose with the use of ½-in. iron nipples and Ts. Each set is connected by ¾-in. iron piping from the connection on the frames through the side of filter-box to the main (pipe on the outside) leading to the vacuum-pump. It may be mentioned here that the connecting of the frames to the vacuum-pump should be at the top, in order to be convenient to get at whenever necessary.

About 35 tons of slime are treated daily at this plant, forming a ¾-in. cake on the filters with a 22-in. vacuum and washing with the same amount of vacuum for from 70 to 80 minutes with a weak solution of cyanide (0.8 lb.), using about 10 tons of wash solution in that time.

Water is scarce in southern Nevada. If water could be used in the washing of the cakes, the time consumed would be about two-thirds that required with the use of a cyanide solution. Also if the slime could be agitated or treated in just twice as much solution as is used at present, necessitating less time of agitation and giving the same result, it would require still less time in washing the cake. The slime-treating capacity is now being enlarged for this purpose.

A record was kept of the charges, in regard to the time in forming cake, pumping of pulp in or out of box, thickness of cake, etc. The following is taken from these records when the plant was first installed:

	Minimum time. Minutes.	Maximum time Minutes.
Filling of box with pulp	16	18
22-in. vacuum, while forming ¾-in. cake	19	22
Emptying box with 5-in. vacuum on cakes	16	17
Pumping in weak-solution wash	15	16
Washing with weak solution, 15-in. vacuum	10	50
Pumping back weak solution	15	16
Filling with wash-water	15	16
Washing with wash-water, 22-in. vacuum	30	50
Sampling cake	7	10
Dropping cake with water	12	14
Running back surplus water	8	10
Sluicing	5	7
Total	198	246

Five months later when carbonate of lime became noticeable and with increased amount of slime in the pulp, the figures were:

	Minimum time. Minutes.	Maximum time. Minutes.
Filling	18	20
Vacuum	22	25
Emptying with 3-in. vacuum on cake	17	20
Pumping in wash	16	18
Washing with 22-in. vacuum	70	80
Sampling cake	7	10
Dropping cake	14	15
Running back water	8	10
Sluicing	5	7
Total	177	205

You will notice that two washes were used when the plant was first installed, but later only one. The reason for this change was the increased amount of slime; the saving of time; the exposure of the cakes the second time for so long to the air, causing them to crack; the wash-water often contained as much cyanide as did the weak wash and the second process gave better results.

At first, when the canvas was new, while pumping the surplus pulp out or wash-water into the box after the cakes had been formed, a vacuum of five inches was left on the frames, this being just enough to keep the cakes from slipping off and not enough to dry or cause cracking, but I found as the carbonate of lime began to form on the canvas (and where only the dipping of the frames in hydrochloric acid was resorted to) that five inches was too much vacuum and caused the cakes to dry and crack, but three inches proved more satisfactory.

After the washing of the cakes they are dislodged from the frames by water forced from the inside. The surplus wash-water is run back to the storage-vat by gravity and only that water ($4\frac{1}{2}$ to $5\frac{1}{2}$ tons, depending on the amount of associated washed slime) remaining in the cone of the box is lost, which is necessary in sluicing the slime away through an 8-in. quick-opening gate-valve.

Very little has been said heretofore about the carbonate of lime forming on and in the filters, which in my opinion is quite an essential obstacle to be got rid of. Although somewhat simple, it is a necessary feature in the successful operation of the process, and it cannot be accomplished by simply dipping the frames in a 2% solution of HCl and water where any large amount of lime is used in the milling of the ore, say from 7 to 20 lb. per ton. It must be accomplished by washing with the use of vacuum, or any other means that either forces or draws the dissolving 2% solution of HCl acid through each cell of the cocoa-matting for 15 to 45 min. If this process is used, the frames will only have to be subjected to the wash once in three or four months, depending on the amount of lime in use, whereas the dipping operation would have to be done at least once in two or three weeks and then it would be very unsatisfactory.

Although a small formation of lime on the filters does not seem either to interfere with the washing of the gold from the cake or to diminish the net extraction, it greatly prolongs the operation and reduces the capacity of the plant.

One cubic foot of dry Combination slime weighs 76 lb. or thereabout, while 160 lb. in water only occupies one cubic foot. It was found that one square foot from a ¾-in. cake contained 5.03 lb. dry slime and 28% moisture, so calculating from these results, in forming a ¾-in. cake you would be treating 6.11 tons of dry slime per charge. But from a larger test we found that 6.76 tons were treated in forming a ¾-in. cake, due to the greater thickness of the cake at the bottom of the frame.

Now, I think I shall be stating the truth when I say that in a plant of this size and design, when filters are kept free from $CaCO_3$, by washing for 70 min. you can treat 50 tons of slime (having the same characteristics as that produced in the Combination mill) in every 24 hours, counting all delays, with the use of a 15-h.p. induction motor operating two mechanical agitators, one vacuum pump, one 4-in. centrifugal slime-pump, and three 2-in. centrifugal pumps.

One man per shift of eight hours can easily operate the valves from a platform near the filter-box, and have plenty of time for other mill-work.

The Butters-Cassel process for filtering slime is certainly a great improvement over all other processes and as an invention it ranks with the tube-mill in importance.

E. S. Pettis.

Denver, February 28.

TUBE-MILL LINING

(March 30, 1907)

The Editor:

Sir—The experience of Mr. A. E. Drucker, in Korea, as given in your issue of November 17, 1906, is interesting. To my mind, the reasons why he obtained such very unsatisfactory results in his ingenious experiment are the following:

1. His mill is probably too light in construction, thus allowing the lining to 'work.' Any continuous bonding movement in the walls of his mill must gradually break up his lining, which is in one continuous length.

2. I think it was a mistake to make the lining in two distinct layers and to cover the tops of the rails.

In reference to my 'honeycomb' linings which he refers to as 'proposed to be used,' I can assure him there is no doubt about the matter, they have actually been in use for over a year, showing over three times the life of imported silex. Nine tube-mills are now running in Waihi with my patent linings.

By dividing up the spaces, as I do, the lining material has not the same chance of working loose as in the method tried by Mr. Drucker. He is quite right in saying that no cement will stand the terrific wear, but it only wears to a certain depth, when it becomes protected by the quartz projecting around it. If Mr. Drucker has a foundry handy I am sure he will find it worth while to try my system, as his only difficulty seems to be in holding the quartz in position and not in the wearing quality of the quartz lining.

H. P. Barry.

Waihi Gold Mining Co., Ltd., Waihi, New Zealand, February 1.

ASSAY OF CYANIDE SOLUTIONS

(March 30, 1907)

The Editor:

Sir—Will you kindly publish the following scheme for the assay of cyanide solutions; it is a modification of Crosse; it is quick and accurate, does away with the tedious evaporation, and chance of loss, and the furnace.

Measure 30 c.c. KCy (working solution), add silver nitrate solution used ordinarily in titrating, until precipitate ceases to form; the gold is all now precipitated as argenti-auric cyanide.

Allow precipitate to settle; decant clear solution, then with wash-bottle transfer precipitate into small funnel, using thin paper; wash several times to get precipitate well down into point of paper; draw out paper and cut off just above the precipitate; place this cone into a hole cut in charcoal, blow-pipe, flatten, part, anneal, and weigh.

H. W. Gendar.

Ballarat, Cal., March 5.

[Many thanks. Such hints are useful.—Editor.]

THE BUTTERS FILTER

(March 30, 1907)

The Editor:

Sir—As Mr. Nutter says, in your issue of February 16, the data given in my former criticism were not taken from the plant at Millers, since at the time of my visit to that camp, the plant was not in operation.

I am glad to get the data supplied by Mr. Nutter, and knowing the possibilities of the filter, I feel sure that the results now being attained by Mr. Parsons will shortly be even better.

My error in regard to the number of filter-leaves at Millers was due to the fact that 216 leaves were billed to the company. I presume that it has been found that less leaves will treat their slime. I should perhaps have said that, "a plant, designed according to my ideas," instead of "a properly designed plant," would be arranged for a maximum lift of 10 ft. I enclose a tracing covering this point. The *average* lift is of course much less varying from minus 10 to plus 10 ft. This is Mr. Bosqui's arrangement at the Combination mill at Goldfield, Nevada. The actual maximum lift varies from hour to hour, depending on the stage of the tide in the various tanks of pulp, solution, and water. It it therefore clear that any construction entailing a maximum lift of 25 ft. will necessitate large pumps (or more time) and more power. The tracing sent you is wrong in showing the discharge from the pump to the wash-solution tank as going over the top of the tank. This should be connected to the tank below the low-water level. It will be easily seen from this drawing that the greatest possible lift would occur in returning pulp or solution, with the storage-tanks full at the moment that the filter-box was empty. This tank is 10 ft. higher than the outlet of the filter-box, and I am unable to see any advantage to be gained by increasing this height. The same arrangement can be applied to a plant of whatsoever size.

As regards the cubic capacity of the filter-box, I must explain that this depends upon the spacing of the leaves, and this spacing depends upon the nature of the slime treated. As an example, experiments are being made upon slime here in Guanajuato of

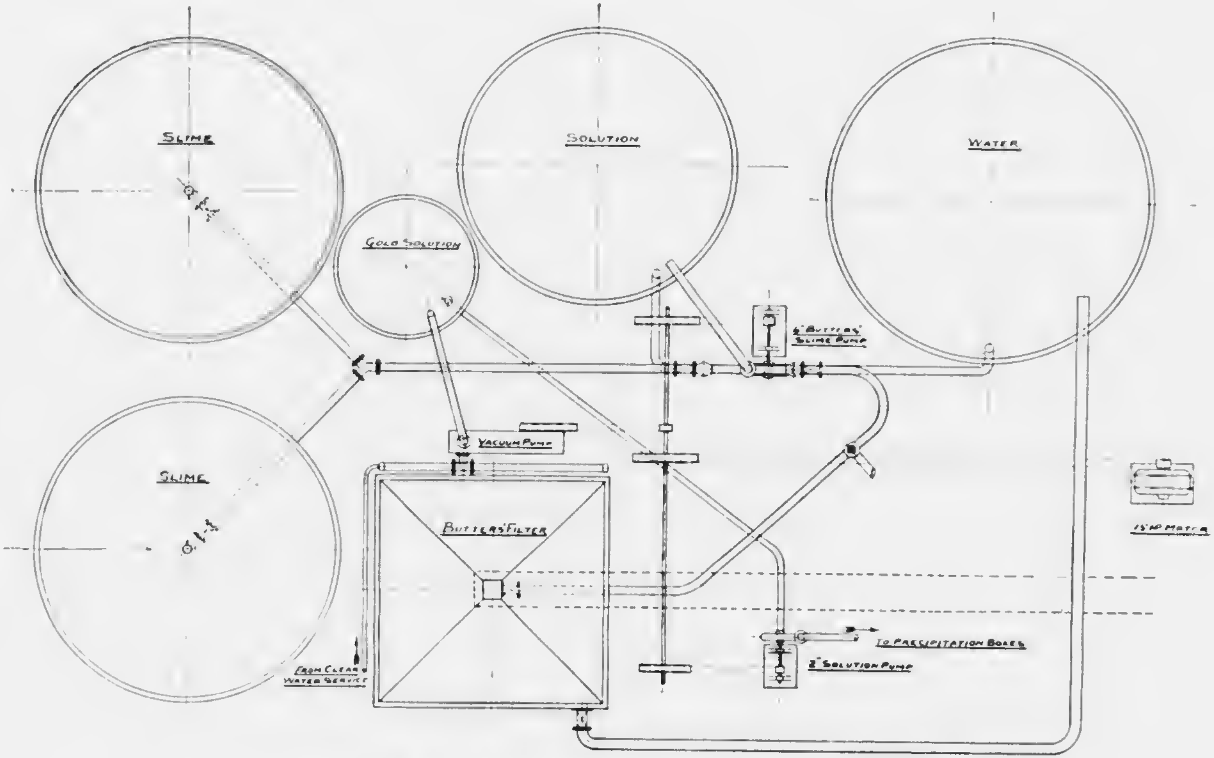


Fig. 29. Arrangement of Butters Plant. Plan.

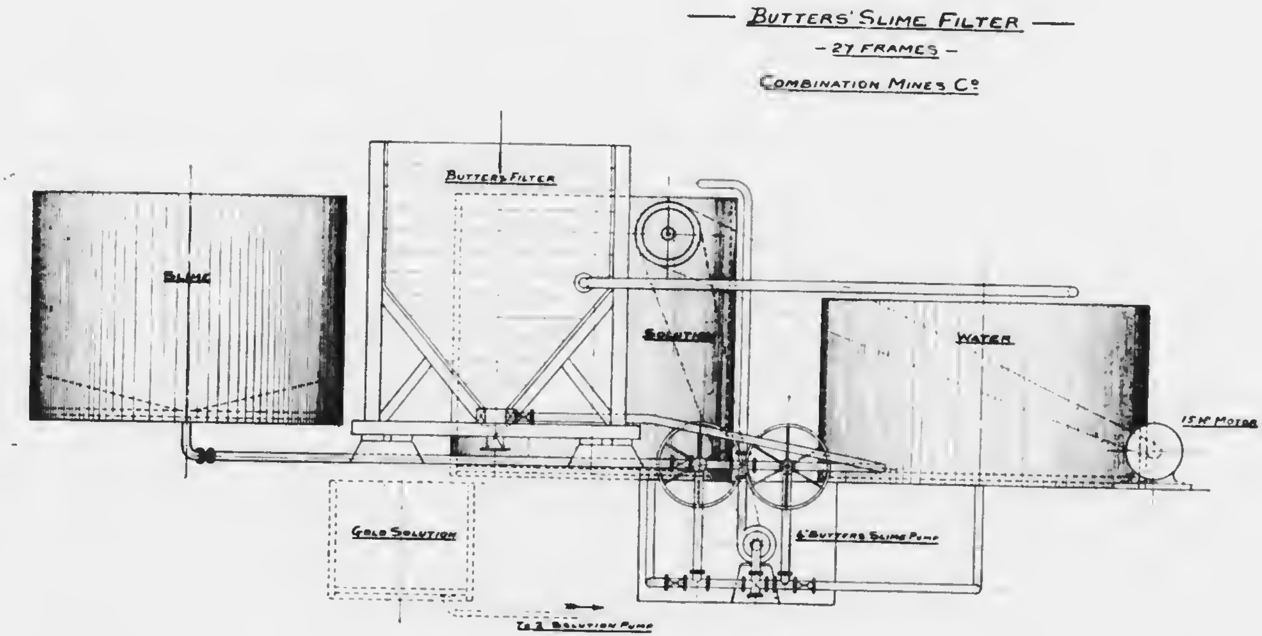

Butters' Slime Filter
- 27 Frames -
Combination Mines Co
Butters Filter
Slime
Water
Gold Solution

Fig. 30

which a cake 1¼ in. thick forms with a 22 in. vacuum in 30 min. This slime necessitates a spacing of five or more inches (instead of four). The filter-box for this slime will be 5 ft. 10 in. from the top of the filter-leaves to the hopper, and the latter will be 6 ft. 3 in. deep. I repeat that for 250 leaves for slime ordinarily produced in milling, a Butters filter-box should not exceed 7,000 cubic feet.

As regards the time and size of pump necessary for transferring pulp and water, and taking Mr. Nutter's figure of 1,900 cu. ft. as the displacement of the loaded leaves, 5,100 cu. ft. of pulp would have to be transferred. A 10-in. pump (I have no data on hand on the capacity of a 9-in. pump) would actually require "according to the manufacturer" 7.5 min. and 15 h.p. to do this.

As regards filter repairs, it is hardly putting it strongly enough (in view of the fact that Butters filter-leaves have been in operation over two years without any signs of deterioration) to merely say that filter repairs are "less" for the Butters than for the Moore process. One of the great points of superiority of the Butters method is that the leaves hang perfectly still, and are subject to no strains, are not scraped (as are the "better type at Bodie") or even touched for months at a time.

As a point of interest, I may state that the filter now being constructed for the Pinguico mill of this place, is designed on "gravity" lines, as illustrated in your issue of recent date, and not "all pumping;" and in this case (where the filter is filled and emptied by gravity, and the pulp returned from a low sump-tank) their maximum lift will not exceed 30 feet.

I must not be suspected of being convinced of the merits of the Butters filter on account of being interested in its success—rather the contrary, having become interested after conviction as to its merits.

MARK R. LAMB.

Guanajuato, Mexico, March 6.

[Having been favored by Mr. Lamb with a copy of the foregoing contribution, Mr. E. H. Nutter writes to say that "the filters at Bodie have not been, and are not, scraped in order to clean off the cake The discharge is accomplished entirely by blowing the cake off with compressed air, as stated by Mr. R. Gilman Brown in an earlier article."—Editor.]

PANS V. TUBE-MILLS

(April 6, 1907)

The Editor:

Sir—With reference to the discussions that have appeared in the MINING AND SCIENTIFIC PRESS regarding the relative merits of pans and tube-mills as sliming machines, it may be of interest to publish the results of some tests carried out at the Dolores mill, in Chihuahua, Mexico.

In test No. 1, the ore was crushed with cyanide solution by 15 stamps, weighing 900 lb., through a 2-mesh screen, the pulp passing to the two Bryan mills with 60-mesh screens; then to four pans in series, thence through two settlers, and finally through two cone-classifiers. The overflow from the classifiers went to the agitation plant, and the underflow was returned to the tube-mill for re-grinding, the re-ground product joining the stream from the settlers at the head of the cones.

In test No. 2, the pulp from the Bryans passed directly to the tube-mill, thence to the cones, the overflow as before to the agitation plant, the underflow being returned to the head of the pan series, thence through settlers to cones as before.

In both tests samples were taken every hour, over periods of twelve hours, from the heads of the tube-mill and heads of pans, and also from the discharge of same. The bulk 12-hour samples was determined from the proportion of sand and slime in each sample, the difference in slime content between heading and tailing being taken as the sliming power of each machine.

The following are the results obtained, from the average of five samples:

TEST NO. 1.

	Percentage of slime.		Percentage of slime
Tube-mill heading	8.20	Pan heading	11.02
" " tailing	42.64	" tailing	52.54
Slimed by tube mill	34.44	By pans	41.52

TEST NO. 2.

	Percentage of slime.		Percentage of slime.
Tube-mill heading	4.66	Pan heading	7.36
" " tailing	44.08	" tailing	51.88
Slimed by tube-mill	39.42	By pans	44.52

Taking an average of both tests, to bring the machines under the same conditions, gives the following result:

Percentage slimed by tube-mill	36.93
" " " pans	43.02
" in favor of pans	6.09

The pans used are of the Wheeler type, with plain flat muller and dies; they have a diameter of five feet. The speed is 65 rev. per minute. The power required to drive the four pans is 50 h.p. The tube-mill is a machine specially constructed by Allis-Chalmers, made in sections for mule transport, and I believe is one of the first, if not the first, sectional tube-mill made. It has a length inside of 16½ ft. by 3 ft. 6 in. diam., and runs at a speed of 35 revolutions. It is lined with white iron, and carries a load of 4½ tons of flint pebbles. The power required to drive it is 20 horse-power.

Although the above tests showed a small margin in favor of the pan for sliming, yet when the question of power and cost are considered the tube-mill is undoubtedly the more economical machine. The wear and tear in the case of the pans is considerably more than that of the tube-mill, and the difference in power required for driving, by itself would more than minimize any advantage the pans have in sliming capacity.

ROBERT CLARKE.

Dolores, February 17.

THE TREATMENT OF DESERT ORES

(April 28, 1907)

The Editor:

Sir—Referring to Mr. Bosqui's recent article on this subject, although in the typical oxidized gold ore of the desert regions of Nevada and California the gold is free, it is so extremely fine and flaky that it does not readily amalgamate, but only yields a comparatively small percentage, and treatment by cyanide is necessary. The present fashion of putting in stamp-mills and amalgamating arrangements followed by a cyanide annex evidently is due solely to the "compelling power of custom," and not to any proof that this is the best method of treating such ores. To me this looks like attempting to make the ore suit the process, instead of suiting the process to the ore.

The wet crushing followed by cyanidation does not allow the treatment by either amalgamation or cyanide to be done under the best conditions. It often happens, especially when inside amalgamating plates are used, that particles of amalgam become detached and are carried with the pulp to the leaching vats. The gold in this amalgam is not completely recovered in the cyanide treatment, because its solution is very slow indeed. In fact, in practice none of it is got. It can only be recovered after the tailing has been allowed to weather for a sufficient time to oxidize all the mercury to mercuric oxide—this and the gold being readily soluble.

Further, the sand and slime are received in the cyanide department in the worst possible condition for treatment, as they contain a large excess of moisture which has to be displaced by cyanide solution. This occasions a mechanical loss of cyanide and the dissolution of the gold is much slower than if the material had been dry when brought in contact with the cyanide solution.

Many of these ores also contain large amounts of hydrated silicate minerals, such as clay and talc, which have the property of becoming plastic when worked with water, yielding slime which is difficult (and requiring an expensive equipment) to treat. On the other hand in the dry-crushing direct-cyanide method, the material is got in the best condition for treatment by cyanide.

As all the interstitial spaces are filled with air, the extraction of the gold is rapid. In the drying also, the hydrated silicate minerals are de-hydrated and lose the property of becoming plastic; therefore the subsequent leaching is much more cheaply and efficiently done. Further, if the ore contains gold too coarse to be dissolved in the time of treatment allowable, the tailing can be easily passed over some amalgamating device and the gold (having been cleaned by the cyanide) will amalgamate readily.

The use of stamp-mills crushing in cyanide solution may be satisfactory in the case of hard clean ores, but with soft oxidized ores, dry-crushing is undoubtedly the best. The cost of installation of a dry-crushing direct-cyanide plant is less than that of a stamp-mill and cyanide annex and will be found to yield a higher profit in the majority of cases. As the majority of these oxidized ores give a practically complete extraction by direct cyanide treatment, what is the advantage of using both amalgamation and cyanide to recover the same value? The removal of a portion of the gold as amalgam does not reduce the cost of cyaniding the material appreciably. The consumption of cyanide due to the gold content is negligible.

In many cases these surface desert ores are porous, so that very fine crushing is not needed and a straight leaching plant can be installed at a much smaller cost than a stamp-mill and cyanide annex of the same capacity. These oxidized surface ores are called "free milling," evidently just because they show free gold on panning. This is not correct. A "free milling ore" is one that yields a large proportion of its gold to plain stamp-mill amalgamation. In no instance will these 'free' desert ores yield any large proportion of their gold to amalgamation but in the majority of cases the gold is completely soluble in cyanide.

BERTRAM HUNT.

San Francisco, April 17, 1906.

SLIME FILTERS

(May 4 1907)

The Editor:

Sir—I have read with great interest the contributions to your columns regarding recent developments in slime filtering. The rivalry of the adherents of the two most prominent systems of vacuum-filtering should tend to improve the practice, but I have watched in vain for a comparison of the results attained here with those reported of the Ridgway filter at the Great Boulder mine.

The question at issue between the Butters-Cassel and Moore adherents seems to be as to whether it is more economical to install and operate a plant designed to transfer solutions to stationary filters or to transfer the filters (and charge) to and from the solution. As there seems to be little evidence to support the claims of superiority in the matter of attendance or speed of deposit and washing the cake, in the same slime, the discussion might be said to be narrowed to the comparative cost per ton-capacity of a plant of each type capable of conducting the remaining operations, transferring and discharging, in a fixed time, with the question of comparative maintenance costs as a further determining factor. Are there not questions of more vital importance to the development of the art?

Judging from published reports on Moore and Butters-Cassel installations, it appears that 50 sq. ft. of filter-area per ton of slime (dry weight) treated in 24 hours is well below the average required. The Ridgway filter, with a total effective filter-area of 50 sq. ft., is reported to have a capacity of from 25 to 50 tons per 24 hours, varying with the character of the slime, or one to two square feet per ton of slime treated. This comparison is rather startling and suggests the necessity of a careful examination of the fundamental elements of the two plans of operation. No conceivable difference in slimes could account for such a wide divergence of results.

In the American practice a cake three-fourths to one inch thick is formed on perpendicular canvas filters. In the Ridgway machine a cake from three-sixteenths to three-eighths of an inch is formed and suspended on a horizontal filter-cloth. The vacuum applied is about the same, but while the cake is formed in 13 sec. in the Ridgway machine, from 30 to 60 min. is required in the American plant. From the nature of the operation it is evident that the rate of accumulation of slime becomes slower as the thickness of the cake increases. The first one-eighth inch is deposited in a much shorter period than the last eighth. It is improbable that the American filters accumulate from three-sixteenths to three-eighths of an inch of cake in the first 13 sec., but the long time required for transfers makes a thick cake a necessary evil.

The very rapid accumulation of the Ridgway filters must be due to elementary differences that would not appear of such importance. Is it the difference in the filtering medium or the position of the filter? It seems likely that filter-cloth is so much less easily clogged than canvas, that the main difference in results is accounted for by this structural difference alone.

The results of observations on the rate of accumulation of the progressive layers of the cake, the effect of position, horizontally suspended or vertical, the rates of accumulation on canvas and filter-cloth, and the durability of filter-cloth in horizontal and vertical positions, would be of great assistance in improving our practice. Progress is in the direction of a compact automatic plant that can be installed and maintained cheaply. The last word has not been spoken in the field of vacuum-filtering.

J. R. Blake.

Searchlight, Nevada, April 8.

ASSAY OF CYANIDE SOLUTIONS

(May 4, 1907)

The Editor:

Sir—I have noticed in the PRESS of March 30, Mr. Gendar's scheme for the assay of cyanide solutions.

I would call the attention of those interested to the following well known method, which, for rapid work and accurate results, will, I think, be found to compare favorably with any.

Measure, according to grade of solution, a convenient quantity of KCy (working solution), add concentrated sulphuric acid to excess; let stand two minutes, then add from one-half to one gram of cement copper and boil until the solution takes on a blue or greenish-blue color, pour onto filter-paper in funnel, wash twice with boiling water; dry the filter-paper containing the precipitate in air bath or otherwise; flux filter-paper and precipitate in assay-crucible, or, scorify, cupel, part, and weigh.

DOUGLAS MUIR.

Guanajuato, Mexico, April 8.

SLIME FILTERING

(May 11 1907)

The Editor:

Sir—Seeing the necessity for a continuous filter in connection with the slime treatment of silver ores, and not finding any reference in your valuable journal, I wish to call your attention and that of some inventor, with means to perfect the idea, to the filtering by centrifugal force, as in the sugar mills, with the difference of raising and discharging the cake over the rim by fixed scrapers, the cake to fall into a second centrifugal machine below the first, where it is washed and filtered with weak solution to be finally discharged with water.

Although convinced that the ever alert inventor has this idea near practical perfection, it would be perhaps a benefit to the mining industry to have it worked out without patent, and for this reason my insinuation in case nobody should have taken one on this method.

Please use this to the best of your judgment without any name.

Old Subscriber.

Mazatlan, Mexico, April 12.

CYANIDE PRACTICE AT THE WAIHI

(May 11, 1907)

The Editor:

Sir—In your issue of December 15, 1906, under the heading 'Progress in Cyanidation,' the statement is made that "at the Waihi, treating 25,000 tons per month, it requires seven men continuously to clean the boxes, using zinc shaving."

This statement is apt to convey a wrong impression, the position being that these men not only clean the boxes weekly at three separate plants cyaniding a total of 27,000 tons per month, but also do all work connected with the acid treatment and smelting (weekly) of the zinc-gold slime, amounting to about five tons dry weight per month, and producing over 100,000 ounces of bullion.

E. G. BANKS.

Waihi, New Zealand, March 15.

METALLURGICAL DEVELOPMENT AT GUANAJUATO

By T. A. Rickard

(May 18, 1907)

It is worth while to tell the story of the metallurgical development at the Sirena mill, more properly named *La Hacienda San Francisco de Pastita*. The successor to the old *patio* was a mill erected in 1899; it contained 20 stamps, each weighing 1,250 lb. The ore first passed through a 9 by 15-in. Blake crusher and was then reduced to 20-mesh by the stamps, from which it passed to six Boss rapid-grinding pans. Here it was re-ground, so that all save 5 to 10% passed an 80-mesh screen; and then it descended to 12 more pans and six settlers. From these the pulp went to five Wilfley tables. The capacity of the mill was 1,500 to 1,800 tons per month. The product was amalgam and concentrate.

The pans extracted 65% of the assay-value and the concentrate contained 12% more. This was on the oxidized ore. Although the concentrate contained 2 kilo. or 64.2 oz. silver, it barely paid to send it to market under the smelter conditions then existing in that part of Mexico. However, another factor came into play; as the lower workings were opened up, the percentage of recovery by amalgamation fell off until it was only 60%. Concurrently, the consumption of mercury and copper sulphate in the pans increased, while the concentrate became richer—five to seven kilo. silver per ton. The method was changed; concentration was made to precede pan amalgamation.

By this new arrangement, the cost of milling was reduced from 7.86 *pesos* to 4.81. The concentration was carried further, so that the product contained 10 to 11 kilo. silver and 115 grams gold per ton; yet the weight of concentrate remained at two per cent of the crude ore. The higher recovery by concentration balanced the lower yield by amalgamation, the commercial result being less satisfactory because the precious metals in the form of amalgam were worth more than when enclosed within a concentrate that had to be transported to a distant smelter. Moreover, the variation in smelter rates introduced a factor of uncertainty.

Extraction finally fell below 60%. This suggested an increase in the mill, so as to lower the fixed charges. At this period the Government tax and the expenses in connection with realization of bullion amounted to 11% of its gross value. The poor extraction and the high imposts left but a small margin of profit. A search for better metallurgical treatment was undertaken. The cyanidation tests made by Leonard Holms in 1901 did not seem to justify turning to that method at that time; subsequently, however, E. M. Hamilton made a new research on a working basis, with a five-ton plant, and he obtained encouraging results. However, nothing was done for a year.

Meanwhile the recovery by amalgamation continued to dwindle and when cyanidation was recommenced, there was a fear lest the further change in the ore with depth might affect extraction by cyanide as it had done that by mercury. In 1904, Bernard MacDonald was engaged to investigate the problem, with the idea, among others, that the Hendryx process might be applied. Complete cyanide tests were made and every kind of ore in the mine was tried. The results fully confirmed Hamilton's earlier work: even on the ore from the bottom of the Sirena mine it was demonstrated that finer grinding was required and that even the concentrate, if ground dry to pass through 200 mesh, would yield 94 to 96% with the use of a 2.5% cyanide solution—a solution unusually strong, but based upon the relative proportion of silver to be dissolved. A weaker solution would have done better, as was proved later.

In these tests a fresh solution was introduced each 24 hours into bottles that were agitated by being attached to the periphery of a slowly revolving wheel. This failed to reproduce working conditions, because it eliminated a drawback inevitable in practice —that is, foul solutions. For this reason the results were higher than was to be expected in the mill, but they warranted the belief that a plant specially designed for the treatment of the concentrate, would yield larger profits than the sale of the concentrate to the smelters. The tests demonstrated also that the silver sulphide was readily attacked by cyanide when the grinding was as fine as 200 mesh. As yet, no plant has been erected to treat the concentrate, but it is likely that this will be done.

In the meanwhile—this was in April, 1904—it was decided to erect the existing cyanide plant, which started to work in May,

1905. When concentration was made to precede amalgamation, the grinding in pans was discontinued, it being found that amalgamation was most effective in charges instead of the continuous process. In erecting the cyanide annex, the only change required was to divert the flow from the Wilfley tables to cone-classifiers. The coarse sand went to a tube-mill of Allis-Chalmers make, 5 ft. diam. and 22 ft. long. It was lined with chilled iron, which, after a three weeks' run, collapsed, so that the use of the tube stopped abruptly. It has not been employed since. Tests proved that the benefit of the re-grinding in the tube hardly warranted the extra expense of repairs and power. The first cone-classifiers went out of commission, the pulp going from the Wilfleys to a set of cone-classifiers that separate the sand from the slime. The arrangement is shown herewith, in Fig 1. *A* is a large spitzkasten, 8 ft. wide and 8 ft. deep, the classification being by gravity. The sand, plus some slime, flows through rubber goose-necks to a height two feet above the bottom into four smaller cones or spitzlutten, 4 ft. wide and 4 ft. deep, equipped with hydraulic jets The undersize from all five cones unites to flow to eight Callow steel settling-cones, 8 ft. diam and 8 ft. deep, where it is de-watered. Thence the pulp passes to three masonry vats, where lime is added to effect settling previous to decantation, at the same time destroying the acidity of the slime and bringing the positive alkalinity up to 1½ lb. lime per ton. Then this slime is pumped to the treatment-vats, the sand meanwhile going to collecting-vats from which it is taken, after draining, in cars to the treatment-vats.

This was the scheme at the commencement of cyanidation; subsequently the masonry vats, formerly employed in de-watering the pulp previous to pan amalgamation, were modified so as to serve for holding and thickening the slime. The five cone-classifiers were moved into the stamp-mill, in order not to lose grade, and in this new position they delivered direct into the masonry vats behind the old amalgamation pans, the vats built in the cyanide annex being used for the same purpose, namely, to de-water the slime. Even now it contains 70% water, and this 70% is just so much liquid that has to be displaced by the effective cyanide solution, until perfect diffusion is attained.

The ore goes from the Sirena mine to the mill in cars (Kilburn & Jacobs) of 50-cu. ft. capacity, carrying 2.4 tons each. They have a double side-dump, with gable bottom, and appear to work easily. Waste is removed by sorting in the big courtyard at the entrance of the main adit. A sorting belt is to be used at the new Soledad shaft, the waste then eliminated to be returned into the mine as filling. The belt is to be 50 ft. long, giving room for five men on each side and illuminated like a billiard table by shaded electric lights. Each man is to sit astride a wooden horse, which is high enough to give freedom of reach over the belt.

Gold can be seen in the surface ores of the Sirena mine; it accompanies the argentite. Pyrite does not appear to be indicative, nor is it a close associate, of the precious metals; it is more plentiful in the undigested country rock. In the ore of the Peregrina mine there is a little arsenical pyrite and also traces of antimonial silver minerals. At Guanajuato generally, the average yield of concentrate does not exceed two per cent, carrying 150 to 1,600 oz. silver, and from 1 to 30 oz. gold per ton, so that the problem is to treat a small quantity of high-grade material, in competition with the expensive freight charges of the railroads and the heavy treatment rates of the smelters. Further, the Government tax is one per cent less on bullion than it is on the precious metals when in the form of concentrate.

In September, 1905, the mill treated 3,887 tons of ore, containing 517 gm. of silver and 2.76 gm. of gold per ton. On leaving the concentrators the pulp assayed 302.5 gm. silver and 1.46 gm. gold. The concentrate recovered amounted to 106 tons, averaging 948.06 gm. silver and 46.92 gm. gold. The cost of crushing and concentration amounted to 1.75 *pesos* per ton. The extraction by concentration was 50.1% of the silver and 49.1% of the gold.

The practice is still in course of development and experiments are continually being made. Re-grinding does not seem required by the Sirena ore; it is stamped through a diagonal-slot screen equivalent to 40 mesh; a chuck-block is used. Of the resulting pulp, 80% goes through 100 mesh. The granular quartz, when crushed, readily liberates the silver sulphide, but the chalcedonic gangue in which the silver occurs (in a cloudy dissemination like moss agate), needs fine grinding—all of it—to pass at least 100 mesh. The concentrate carries 30% silica; the portion that passes through 200 mesh represents 15.5% by weight and as it is worth 398 *pesos* per ton, it contains 42% of the assay-value of the ore.

In watching the agitation in the slime vats, it was noticeable how the circular motion becomes accelerated until the moving mass of pulp and solution moves faster than the paddles. On starting the agitation, one can see the sinuous streaks of clear cyanide solution amid the slime, and this condition of imperfect dispersion is never wholly overcome; it is due to the resistance of slime to diffusion. I noticed this appearance (or phenomenon) in a vat that had been at work for 40 minutes. Another note;

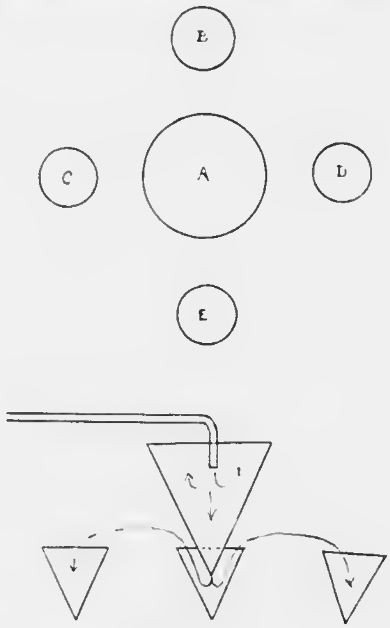


Fig. 31.

even ten minutes after the agitator is stopped the movement of water at top of the vat continues in the direction started by the paddles. Two pounds of lime are added per ton of solution in order to hasten settlement of the slime. The philosophy of this has been discussed in connection with milling at El Oro.

The loss of sodium cyanide at the Sirena mill is 4.12 lb. per ton of crude ore, while the consumption of lime is at the rate of 6 lb., worth 12 *pesos* per metric ton. Sodium cyanide costs

Fig. 32. Reduction Works of the Guanajuato Consolidated Mining Company.

$15.75 per metric ton at Marfil station, the present terminus of the Mexican Central Railroad and to this must be added 1.25 *pesos*, or, say 60 cents for transport to the works, making the total cost $16.35 per metric ton of 2,204 pounds.

There is always some re-precipitation when treating silver sulphide, by reason of the formation of potassium sulphide, but this is diminished by the addition of lead acetate, which forms a plumbous hydrate that removes the soluble sulphides by forming a lead sulphide and the potassium or sodium hydrate. In practice, the re-precipitation of silver is surpassed by the re-dissolving of it in the cyanide solution.

By passing through cone-classifiers the product escaping from the upper mill is divided into 'sand' and 'slime,' which are treated separately, or in the cyanide annex. In the 'sand' department there were 20 vats, each containing an average charge of 2,651.7 cu ft., or 89.6 tons. During the month 1,792 tons (dry) of sand was treated. The average assay-value before treatment was 297.5 gm. silver and 1.37 gm. gold; after treatment the contents were 52 gm. silver and 0.1 gm. gold. In the 'slime' department there were 82 vats, each containing 3,851.8 cu. ft. of wet slime, equivalent to 24.26 tons dry. During the month 1,989 tons were treated. The average assay-value before treatment was 275.5 gm. silver and 1.3 gm. gold. After treatment the assay became 45.5 gm. silver and 0.1 gm. gold. The extraction was 85.1% as regards the silver and 70% of the gold in the pulp treated by the cyanide annex, the total recovery by cyanidation being 41% of the assay-value of the crude ore, so that the combined extraction by cyanidation and concentration was 91.2 per cent. The total cost of cyanidation was $4.13, and the consumption of cyanide 1.9 kilo. per ton.

Note should be made of the fact that successful experiments with cyanide on silver ore had been made earlier at the mines in Sinaloa, but the results had not been heralded because they were obtained at private properties, and even in these cases the official tests of the cyanide company at Mexico City had discouraged hope. The trouble was due to the use of too weak a solution—a swing of the pendulum in cyanide practice, for in its early days the main fault was the employment of a needlessly strong solution. Another factor, that prevented success with these silver ores, was the insufficient time given for chemical action. The element of time is especially important in the case of concentrate—iron pyrite carrying gold free and silver as argentite; the millman can afford to give the time required because of the small quantity of this product and its richness.

TREATMENT OF MATTE FROM THE CYANIDE MILL

By A. E. Drucker

(May 18, 1907)

It may be of interest to some to know of a quick and complete way in which to extract the metals from a rich matte as obtained at a cyanide clean-up. It often happens that the yield from this source adds considerably to the month's bullion.

Sulphur, the matte-forming element, comes from the zinc sulphate ($ZnSO_4$) remaining with the precipitate after acid treatment. The amount of $ZnSO_4$ present will depend, of course, upon the number of water-washes used in the final acid treatment. The more the precipitate is washed, the less will be the matte formed on a gold button. I believe that in our case (treatment of concentrate) it is possible at times for extermely fine material to escape from the filters, to be carried in suspension to the zinc-boxes where it is bound to remain with the other precipitate. The amount, however, is exceedingly small. If such be the case after treatment with acid, ferrous sulphate ($FeSO_4$) will be contained in solution, while free sulphur separates out with the precipitate according to the reaction:

$$FeS_2 + H_2SO_4 = FeSO_4 + H_2S + S.$$

There is a possibility then, unless a complete roast is given to the acid-treated precipitate, of sulphur being present as a matte-forming material upon melting the final precipitate. The amount of sulphur obtained from this source to form matte will in all probability be slight. Experiments now point to the fact that the amount of matte obtained on a gold button depends principally upon two things, namely, the completeness of the final water-washes after acid treatment and the extent to which the final roast has been carried (whether merely a drying of the precipitate or a complete roast).

The matte that I am treating contains zinc, lead, iron, gold, and silver. When using the zinc-lead couple for precipitation, the presence of lead in the matte, if the gold slime is acid-treated and then smelted in a graphite pot, is practically inevitable.

Lead in the matte may also be due partly to the dissolving (by the cyanide solution) of lead salts from oxidized concentrate followed by the precipitation of lead on the zinc. It may originate from the lead acetate used in some solutions. There are two sources by which iron may enter the matte, either by iron pyrite collecting in the zinc-boxes and finally appearing in the acid-treated precipitate as $FeSO_4$, or by some iron scale gathered from the

Fig. 33. Oriental Consolidated Mill. Korea.

roasting-pan with the gold precipitate. Zinc seems to enter the matte in considerable amount, especially when the acid-treated slime contains some undissolved zinc or has been only partly roasted, so as to contain zinc sulphate.

I believe that a small amount of matte on a button is desirable, for the slag above will, as a general rule, assay less in silver, especially when the matte contains lead. No nitre is used by us to keep down the matte upon fusion of the slime in crucibles. Its damage to the pots amounts to more than the cost of treatment of the additional matte. Previously to using my present method of reduction, we certainly did not welcome the presence of any considerable amount of matte at a clean-up, since the old method of treating small quantities with nitre or scrap iron was slow and incomplete. It is not safe to allow any amount of this rich matte to accumulate (it assays about $20 per lb.) and for that reason the best plan is to treat it immediately at the end of the clean-up, melting the bullion with the gold buttons into bars.

Potassium cyanide acts as a powerful reducing and desulphurizing flux, and for this reason it was used for decomposing the matte. Our particular matte is unusually high-grade, assaying as much as $45,000 per ton. It contains zinc, lead, and iron as sulphides, besides the precious metals. At the monthly clean-up about 800 lb. precipitate is obtained, producing nearly 3,000 oz. bullion, and from the treatment of this we obtain from 30 to 40 lb. matte. The amount varies from month to month. The method of treatment for the matte is as follows:

The matte, borax, and cyanide are put separately through a small rock-breaker and crushed fine. Borax and cyanide are used as fluxes and are put with the matte into three No. 60 graphite pots in the following proportions and order: There are alternate layers of borax, matte, and cyanide throughout, until the pot is full, and finally covered with borax. The crucibles are now put into the furnace and a white heat maintained for two or three hours. As a rule, it takes a little over two hours with a good hot furnace. It can be seen when the action is complete, for the charge will subside and the bubbling cease. The action and burning of the sulphur will be violent at the end of the first hour. Also the slag will become quite thick when the action is complete, being removed with a skimmer; this is necessary before pouring the contents of the crucible into a conical mold. If an excess of cyanide be used, it will be found just above the gold button and can be broken off by a blow with a hammer. The matte will now be entirely decomposed, leaving only a light porous slag and the gold button.

This method shows an extraction of 85 to 94% of the total value of the original matte, depending upon the richness of the material treated.

CYANIDE PRACTICE AT COPALA

(June 8, 1907)

The Editor:

Sir—In regard to saving cyanide in the treatment of gold and silver ore, we have been enabled by a simple method to reduce our cyanide consumption from 4.5 to 1.5 lb. per ton of ore treated, and the results are so interesting that I herewith tender them for publication.

When we first commenced treatment in October, 1905, our protective alkalinity in the working solutions was carried at 0.04% (in terms of caustic soda), and the cyanide consumption varied from 4.3 to 4.5 lb. per ton of ore.

The ore is valuable for its silver, which occurs as a sulphide, and carries from 12 to 20 oz. of this metal per ton, together with a small quantity of gold. We find that 5.2 lb. of lime per ton is sufficient to keep the alkalinity at 0.04%. Our extraction has been good, averaging 93% of both gold and silver, and, accepting the fact that a certain amount of cyanide will of necessity be constantly tied up as a double cyanide of zinc and sodium (using sodium cyanide), we have not considered the amount abnormal, though the advisability had been discussed of installing electrolytic precipitation to obviate this loss of potential free cyanide. However, in reading over 'The Cyaniding of Gold and Silver Ores,' by Julian and Smart, I noted what was said (page 99) in regard to the decomposition of the double cyanide (Na_2ZnCy_4) by an increase in the free alkali. W. R. Feldtman is there quoted as saying: "The addition of alkali to working cyanide solutions which have become somewhat weak in alkali, brings up the strength by regenerating, that is, decomposing, the zinc cyanide, so that as a matter of fact, when the solutions are pretty strongly alkaline they contain no zinc as cyanide, but only the hydrate dissolved in alkali."

With the object of ascertaining what results could be obtained by increasing the alkalinity, I commenced adding lime, and kept gradually increasing the same until the solutions tested 0.2% alkalinity. Our working solution for slime, which we carry at 0.125% KCy, at once began to gain in strength, and kept growing gradually stronger until it showed 0.3% KCy; the alkalinity was then allowed to fall to 0.09%, when cyanide strength also fell. After many experiments with various strengths, I found 0.135% to be the least alkalinity with which I could regenerate, and consequently I have kept the alkali at that strength ever since.

As a result of this regeneration, our cyanide consumption has not exceeded 1.5 lb. per ton treated for more than five months; as a matter of fact, no cyanide was added to the slime-treatment solutions for nearly 13 weeks, and the amount used to bring the leaching-plant solutions up to strength was small; this of course was due to the large excess of zinc cyanide then existing in the system.

In regard to the testing for alkali, we first determine the free cyanide by titrating with standard silver nitrate solution to the first faint opalescence, without the use of the potassium iodide indicator; we then add a few drops more of the silver nitrate to make the opalescence well defined, and titrate with N/5 oxalic acid solution using phenolphthalein as indicator. This is the method recommended by J. E. Clennell, and as it has been found to check very closely with L. M. Green's ferrocyanide method, and also with that of Gerard Williams published in the *Journal* of the Chemical, Metallurgical & Mining Society of South Africa, it has been adopted by preference on account of its simplicity.

Finally, I would say that up to date there is no indication of increase in our cyanide consumption, and as this work has been going on for the past five months I have come to the conclusion that the regeneration has assumed constant proportion.

L. McN. B. BULLOCK.

Copala, Sinaloa, Mexico, May 4.

ASSAY OF CYANIDE SOLUTIONS

(June 8, 1907)

The Editor:

Sir—Having noticed the schemes for the 'Assay of Cyanide Solutions' in the Press of March 30, by Mr. Gendar, and of May 4, by Mr. Muir, I wish to call attention to the following well known method, which I have found quick and accurate.

To a convenient quantity of KCN (working solution) add an excess of weak copper sulphate solution. Acidify with dilute sulphuric acid. Filter the precipitate, which is white (or light-bluish green) and flocculent, and which contains all the gold and silver. Wash the precipitate twice with hot water; dry filter-paper and contents by placing them in a crucible and heating in a muffle; flux filter-paper and precipitate, fuse, cupel, part, and weigh.

To solutions weak in KCN add a small piece of KCN salt, in order to give a good precipitate of copper cyanide.

Augustus MacDonald.

Zacatecas, Mexico, May 17.

(June 8, 1907.)

Some time ago Messrs. G. A. and H. S. Denny announced, with a great flare of trumpets, the triumph of their metallurgical innovation. Several of the mines under their technical control spent thousands of pounds to install the process of circulating cyanide solutions, advocated by the Denny brothers. The following paragraph from a speech delivered at the annual meeting of the New Goch mine, by Mr. George Albu, the chairman, is rather disconcerting to the 'new metallurgy:' "In addition to other drawbacks, we have experienced considerable trouble with our new metallurgical plant. Whatever theoretical merits the system of circulating cyanide solutions may possess, the fact remains that the one check depended upon to ascertain the average value of the ore going into the mill, namely, the screen-assay, is effectually destroyed. Our managers and technical advisers have endeavored to overcome this difficulty, but apparently without any great success, and consequently we have been frequently disappointed at the end of the month in the quantity of gold won, having been misled by the unreliable nature of the screen-assays. I do not propose to enter into technical details, but the suggestion is now under consideration to abandon the use of circulating cyanide solutions and revert to what is known as the decantation process, supplemented by the filter-process."

Common caution should have prevented the introduction of the 'new metallurgy' on more than one mine, until it was proved an unqualified success. Because a scheme works well for a few weeks it is not safe to pronounce it a success. An actual working test of a year or so is necessary when such a proposition as the new metallurgy is suggested. It is fortunate for the rest of the Rand that conservative councils prevailed, and that the mines of other groups did not take up the metallurgical schemes so persistently promulgated by the Messrs. Denny.

THE FILTRATION OF SLIME BY THE BUTTERS METHOD

By E. M. Hamilton

(June 22 and 29, 1907)

As the Butters patent filter has now been in operation for nearly three years, it may fairly be considered a practical success, and the aim of the present paper is to give a detailed description of its inception, construction, and method of working, in the hope that it may prove of general interest to the mining community.

The need of a filter of some sort was first forcibly presented to the minds of Charles Butters and his associates at their works in Virginia City, Nevada. The tailing being cyanided there was originally derived from the Comstock mills, but it has been treated and re-treated several times by the pan-amalgamation process, and as it stands today in the dams it contains about 75 per cent of material that is unleachable, and which may, therefore, be designated as 'slime.' And this slime is of an exceptional character. In addition to the difficulties connected with solution of the gold and silver contents, which it is not proposed to touch upon in this paper, the mechanical condition of it is such that it gives trouble in settlement for decantation. The clarification produced by a coagulent such as lime is perfect, but the subsidence is so slow that the amount of solution recoverable in this way is not enough to make the decantation process a practical success. A 70-ton charge, with a four to one dilution, will in 24 hours not settle to less than three parts by weight of solution to one of slime, and the thickest pulp obtainable by two or three days' settlement in the deep vats will still contain about two of solution to one of slime.

A curious point about it is that even after being washed in many changes of clean water to remove all soluble salts, it will still settle in a medium of distilled water more rapidly and compactly than in any solution hitherto tried, whether acid or alkaline. When lime, however, is added to the pulp the mixture seems to coagulate into a gelatinous curd, and the slime loses to a large extent its power to separate out by gravity.

Here are displayed in a marked manner the phenomena mentioned by Julian and Smart in 'Cyaniding Gold and Silver Ores,' namely, that coagulation is not necessarily accompanied by good settlement, and that under certain circumstances excessive coagulation is positively injurious. In one instance a sample of slime, from which most of the fine silica had been removed by careful separation, was diluted in the proportion of four of water to one of slime, and sufficient lime added to neutralize the charge and leave about 0.05 per cent of free alkali in solution; this pulp was then placed in a graduated litre cylinder, where its level stood at the 875-c.c. mark. At the end of 24 hours the solids had not subsided through more than four divisions of the scale, a distance equivalent to 40 c.c. of solution.

Another fact pointed out by the same authors, in regard to the quality of the settled slime produced by different electrolytes, is also exemplified on this material, for the use of caustic potash instead of lime produces a curd of much finer grain, which settles better in every way, and finally yields a good clarification also, though the latter proceeds more slowly than the subsidence of the well-marked line of solids, and the solution has not the brilliancy of that settled with lime. Thus, if the filter method had not been adopted it is probable that caustic soda or potash would have been more useful than lime as a protective alkali.

In regard to the settlement in distilled water, if it be true that some degree of coagulation is a necessary precursor to settlement and clarification, then the slime being treated must either possess some inherent power to coagulate, or else, in spite of thorough washing, must develop sufficient soluble material by standing in water to form an electrolyte suitable for this purpose.

It may be interesting here to give a few results, out of a number of determinations, on the rate of settlement of this slime. These tests were made on an average sample such as is usually treated in the vats. The accompanying diagram will illustrate. (See Fig. 34.) The arrow-heads denote the defined surface lines of subsiding solids; the percentage figures on the right-hand side of each cylinder represent the percentage of moisture still held by the settled slime at that point. The following notes will explain:

CYLINDER NO. 1.—This sample of slime was washed in 10 changes of clear service-water, the operation extending over several days, and finishing with distilled water. When this last wash had been settled and decanted, distilled water was used to bring up the dilution to 4 to 1. The line of subsidence was well defined from the start, but the liquor retained a faint opal tint, which did not disappear throughout the test.

CYLINDERS NO. 2 and 3.—These were both neutralized with their respective alkalis for two days before beginning the test, so as to insure having an approximately constant strength of free alkali in solution during the time of settlement; in the case of the lime this strength was 0.035 per cent alkalinity, and in that of the caustic potash 0.08 per cent, both in terms of caustic potash.

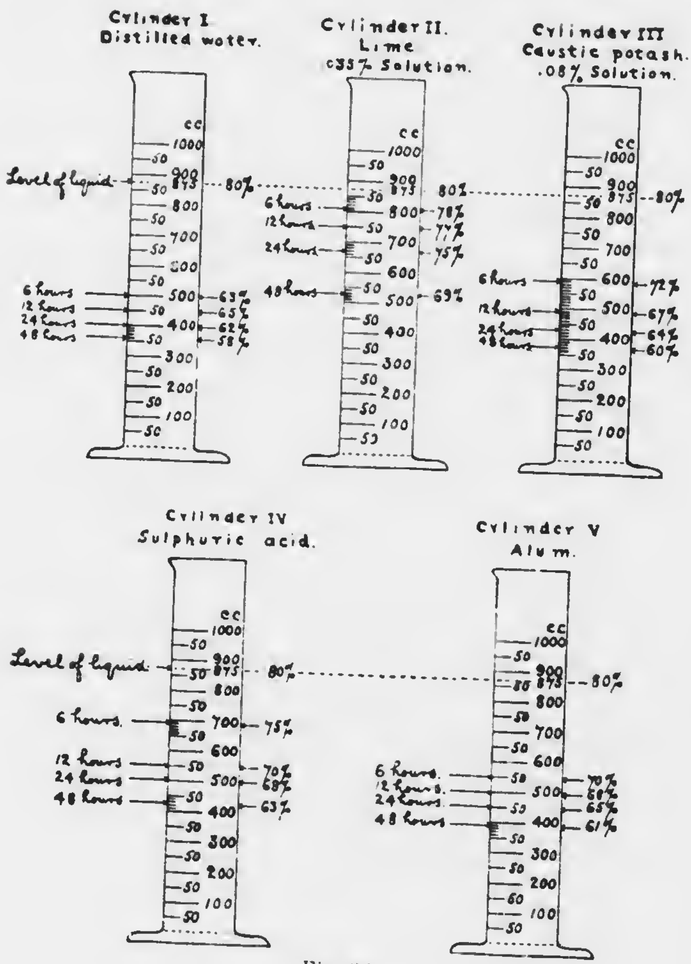


Fig. 34

Similar tests, made with considerably larger amounts of free alkali in solution, showed slightly inferior results in the case of lime, but a tendency to closer settlement with caustic potash. In the case of cylinder No. 2, the lime left a sparkle and limpidity of the supernatant solution lacking in that settled with caustic potash, though the latter might be said to be perfectly clear unless compared with No. 2, when a difference would become apparent.

CYLINDER NO. 4.—In this test the same amount of slime was taken as before, and the same proportion of water added. Sufficient pure sulphuric acid was then mixed with it to leave a little free acid in excess after dissolving all it would of the soluble material. In this medium the slime settled better than in the lime solution, but the liquid remained slightly opalescent.

CYLINDER NO. 5.—To this five grams of alum were added and allowed to dissolve, giving a result nearly as good as that obtained with caustic potash, and a solution of about the same degree of clearness, that is, clean, but lacking the absolute purity of the lime solution.

The experiments here given will be sufficient to indicate the nature of the material to be treated, which has a bearing on the results obtained by the use of the filter.

After experimenting with several forms of vacuum-filter units, both cylindrical and rectangular, the system hereinafter described was finally evolved, and very few changes have been made in it since the first appliance was installed on a working scale in the Virginia City plant.

As at present constructed the filter-leaf is made on a frame, the upper side of which is formed on a bar of wood, 1¾ in. thick and 5½ in. wide, placed on edge, and 10 ft. in length, or sufficiently long for its ends to rest on top of the containing box from side to side. (See Fig. 35.) The remaining three sides of the frame are made of ½-in. pipe, one extremity of which is flattened, turned at a right angle, and firmly bolted to the under side of the wooden bar, while the other extremity projects through the other end of the same bar, in order to receive its connection with the vacuum-chamber, being tightly clamped to the wood at the point where it passes through. The upper face of that portion of the pipe forming the bottom of the frame is pierced with 5/16-in. holes at intervals of four inches for its whole length, and through these the solution passes off from the filter into the receiver.

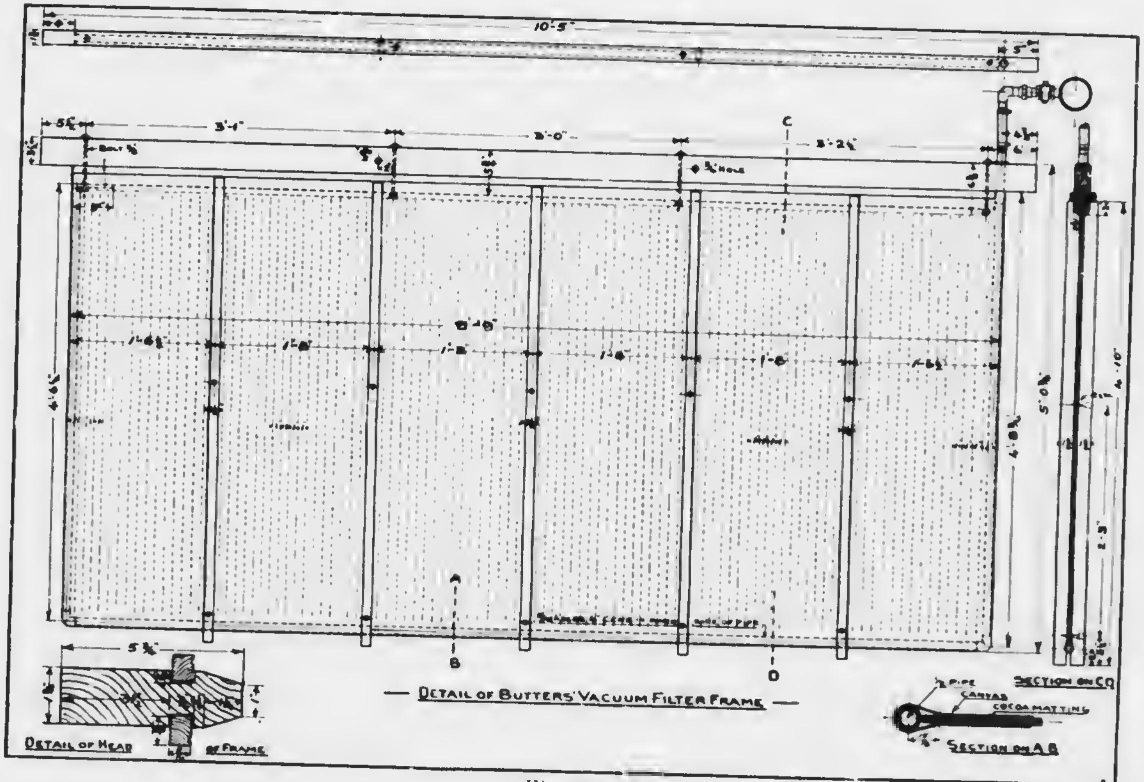

DETAIL OF BUTTERS' VACUUM FILTER FRAME
DETAIL OF HEAD OF FRAME
SECTION ON C.D.
SECTION ON A.B.
½" PIPE
CANVAS
COCOA MATTING
10'-5"
3'-1"
3'-0"
3'-2¼"
5'-0⅛"
4'-8⅛"

Fig. 35.

The filtering medium consists, first, of a piece of cocoa mat, of such a size as exactly to fill the space formed by the frame described above. On either side of this is placed a sheet of 16-oz. canvas, large enough to overlap the frame. These three thicknesses of material are then sewed together by machine with parallel rows of stitching four inches apart,* and perpendicular to the sustaining wooden bar. Next, the filtering medium so formed is fitted in the frame, the overlapping canvas edges being stitched round the outside of the piping, and the upper edge fastened to the wooden bar by being sunk in the groove shown in the drawing and a lath driven in over it, wedging it firmly in position. This lath is also made to fulfill another function; instead of being of such a thickness as to be driven flush with the surface of the wood, it is made to project outward for a distance of about half an inch, and the under side of this projection is grooved or weathered so as to deflect the drippings of solution that run down off the bar when the pulp is withdrawn, and prevent them from trickling over the surface of the newly-formed cake. This action was found to cut channels in the surface of the slime, and form the starting points of cracks. A point to be noticed in the section (of the head of the frame) is the shoulder, formed by the lower angle of the wooden top, over which the canvas has to be strained before reaching its attachment. The lack of support at this point between the canvas and the cocoa mat was found to cause a tendency to start cracks in the cake, and to avoid this the material is rendered impervious with P. & B. paint down to the line at which it comes in contact with the mat, which is about 1½ in. below the lower extremity of the wooden support. At this point there is a horizontal line of stitching holding the three thicknesses of material together. The paint is also applied all round the frame of pipe in a border about an inch wide, for the same reason, and this has proved a simple way of meeting the difficulty, and giving the cake a perfectly plane surface on which to build itself up.

After everything has been done, six or seven strips of wood are placed vertically on each side of the leaf, thus dividing it into a number of panels of equal size; these reach from top to bottom of the frame and are screwed together through the filtering material. They act as braces to stiffen the frame and also to

*The latest plan is to have these rows of stitching only one inch apart; this gives greater strength and rigidity.

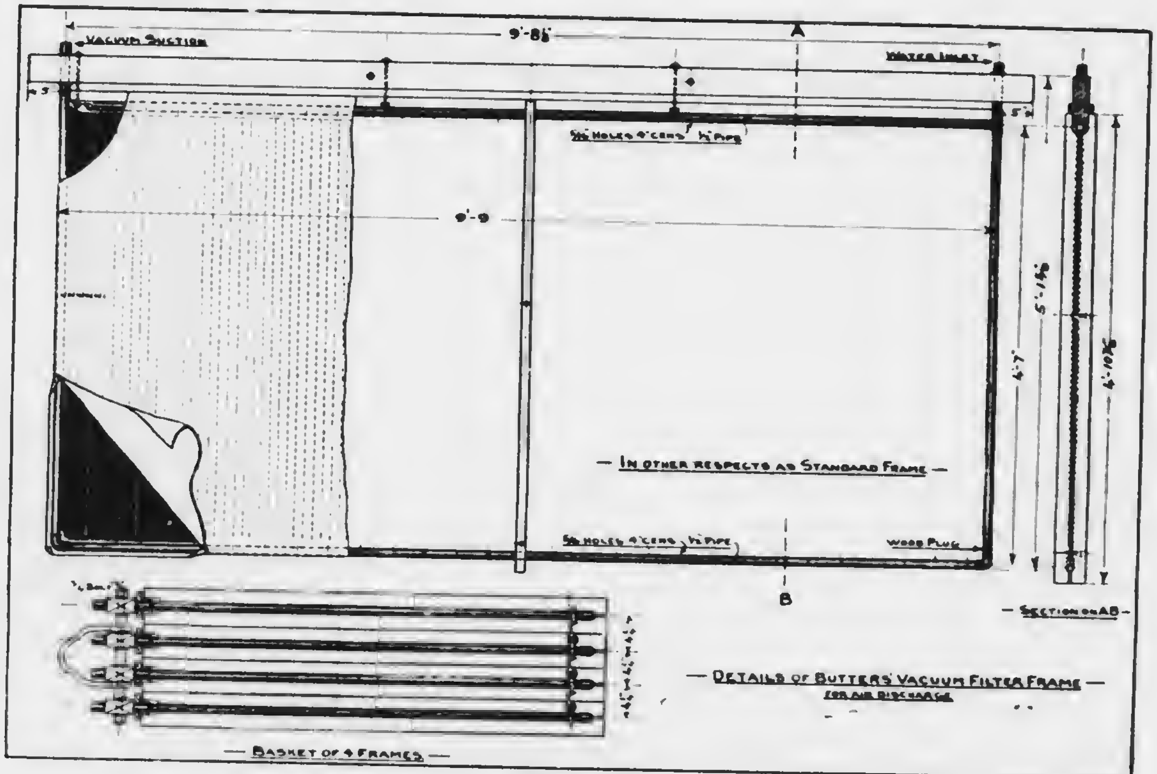
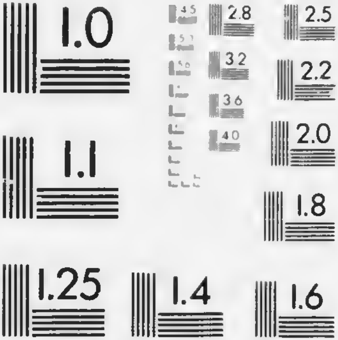

9'-8½"
VACUUM SUCTION
WATER INLET
⅝" HOLES 4" CENS. ½" PIPE
9'-9
IN OTHER RESPECTS AS STANDARD FRAME
WOOD PLUG
4'-7"
5'-1½"
4'-10⅜"
SECTION ON AB
A
B
DETAILS OF BUTTERS' VACUUM FILTER FRAME
FOR AIR DISCHARGE
BASKET OF 4 FRAMES

Fig. 36.

MICROCOPY RESOLUTION TEST CHART

(ANSI and ISO TEST CHART No. 2)



APPLIED IMAGE Inc
1653 East Main Street
Rochester, New York 14609 USA
(716) 482 - [illegible] Phone
(716) 288 - 5989 Fax

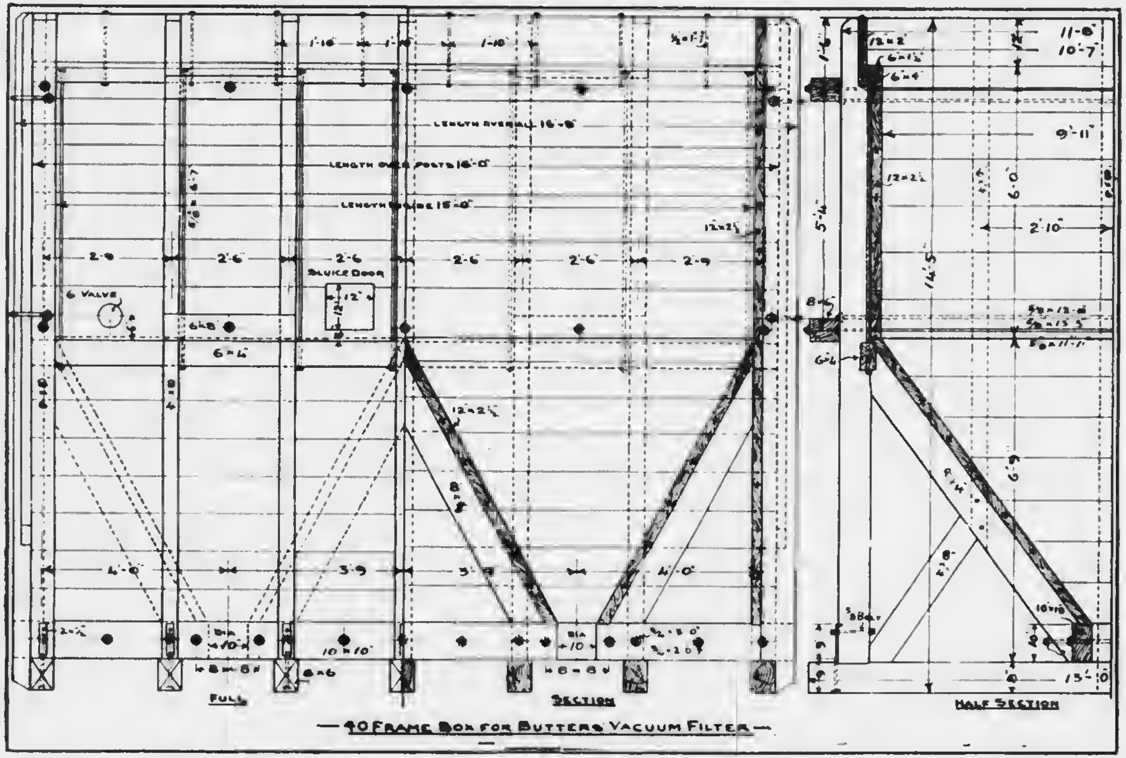
hold the canvas vertical and prevent it from sagging out to one side or the other.

For localities where water is scarce, and generally where it is desirable to send out a comparatively dry cake, the frame of the leaf is now modified to admit of throwing off the cake without previously submerging it in water as is done at Virginia City. The frame of pipe in this case is made to extend round all four sides instead of only three; the extremities pass through the wooden bar at each end instead of only at one end of it, and the pipe at the upper side of the frame is perforated like the lower one. By this arrangement, water can be injected into the leaf along the top through one of the projecting extensions of the pipe, and simultaneously compressed air can be admitted through the other pipe into the lower part of the frame. This has been found to make a perfect discharge of the cake into the empty box, whereas water alone could not be made to give a proper discharge unless the cakes were submerged.

Compressed air, alone, can be used for a dry discharge, but it is not as satisfactory as the combination of water and air simultaneously. (See Fig. 36.)

At Virginia City the containing box is an electrolytic precipitation box which was not needed for its special purpose and was adapted to the use of the new filter, being allowed to remain in its original position. It was formerly divided into 12 compartments, each having a hopper-bottom; the partitions were removed, and with them two of the sides of each hopper, so that the whole space was vacant from end to end, and after the heavy top rails had been removed the result was a box 31 ft. 8 in. long and 9 ft. 11 in. wide, the sides being vertical for a depth of five feet and then meeting the V-shaped bottom.

In place of the top rails, boards 5¾ in. wide and 1 in. thick were placed horizontally on the top edges of the box, and on the outer edge of these were fixed vertical boards 12 in. high and 1¾ in. thick, all being well bolted together and to the box, the result was to raise the sides 10 in. higher than they were when used for precipitation. The two shelves thus formed along the sides of the box serve to carry the ends of the filter-leaf bars and allow the leaves to hang free inside the box (Fig. 37.) On one side of the sloping bottom and extending down to its lowest point are four discharge-doors, each giving an aperture of 12 in. square; the doors are



— 40 FRAME BOX FOR BUTTERS' VACUUM FILTER —
FULL
SECTION
HALF SECTION
SLUICE DOOR
6 VALVE

Fig. 37.

hinged on their lower sides, and seat on a rubber insertion led into the iron frame; they are closed by hand, and then held in position by a lever. To discharge the contents of the box it is only necessary to shift the lever, and the door drops open by its own weight. At each end of the box is another door 10 in. square, through which the nozzle of a hose is manipulated for flushing out any hard slime that may remain behind in the bottom.

When designing a containing-box for the special purpose of the filter, the lines of the original box are departed from as regards the shape of the bottom, which is then composed of one or more complete hoppers of about 60° for the better discharge of the cake, and the door of each hopper is placed at its apex and may be a 12-in. quick-opening valve or any other suitable device. With such an arrangement the residue gravitates out without the aid of any washing with hoses.

At Virginia City the exhaust-pump is a converted compressor, which happened to be on hand. It was made by the Rix Compressed Air & Drill Co. of San Francisco, and is described in their catalogue as a "vertical duplex blowing engine." It has two cylinders 14 in. by 8 in. and is belt-driven, the crank-shaft making 120 rev. per min. Only one cylinder is used for the purpose of the filter, the other being connected with a separate drum and used to form a vacuum for drying out the final washes from the sand-leaching vats.

The vacuum-drum is of riveted boiler-plate, and is 3 ft. 8 in. diameter, 13 ft. 4 in. long, standing upright on its end. The pipe from the vacuum-pump enters at the top, while the pipe connecting with the filters enters 4 ft. from the bottom. At or near the bottom a drain-pipe enters, for removing the solution as it is drawn in from the filter-leaves. This pipe passes down into a small sump 22 ft. below, so that the solution will drain by gravity and keep the chamber empty. The level of the solution in the sump is, of course, always kept above the discharge to prevent the entrance of air into the drum. The average barometric pressure at the level of the works is 24.5 and the average vacuum maintained is 22.5 in., which is indicated by a column of mercury connected with the drum. As the small receiving-sump fills with solution, it is periodically pumped into a storage-vat, from which it passes to the precipitation-boxes.

Instead of the dry vacuum-pump and gravity-drainage, a 'wet' vacuum-pump may be used, which would obviate the necessity for placing a solution-sump 20 to 30 ft. below the filter-level, in cases where such an arrangement might be difficult or undesirable.

The filter-leaves are placed in the containing-box at a distance from each other of about four inches from centre to centre, and are then ready to be connected with the vacuum-drum. One space is usually left vacant midway along the box to allow access to the surface of the leaf for purposes of measuring the formation of the cake, and also of admitting an electric light to observe its condition after the charge of pulp has been withdrawn. Blocks of wood are fastened on the shelves that carry the filter-leaf bars to act as spacers between the leaves, and it is generally found advisable also to use a line of blocks or iron clamps down the centre of the row of frames to prevent the bars from sagging horizontally toward one another under the weight of the cake. The ends of the bars, too, need to be held down in their places by wooden buttons, to prevent them from floating up by the action of the bouyancy of the thick pulp.

A 4-in. pipe runs the whole length of one side of the containing-box, and has a number of small horizontal holes bored in it, one exactly over each filter-leaf; into each hole is screwed a ½-in. nipple and plug-cock, on the further side of which is a union into which screws the small end of a reducing ell; the large end of the small ell enters a short length of rubber hose, which connects with the projecting end of pipe from the filter-frame (Fig. 35). Thus, by unscrewing the union any individual leaf may be disconnected and removed from the box in a few minutes.

At each end of the 4-in. pipe is a quick-opening valve; one is connected with the vacuum-drum, and by opening it the suction is applied to the filters. The other unites with a low-pressure water-service of about 20 ft. head, and through it water is admitted between the two sheets of canvas for the purpose of detaching the residue-cake prior to discharging.

There is also a ¾-in. pipe running all around the top of the box; this has holes pierced in its under side, and is connected with a small solution-vat above; its function is to form a spray to wash down the sides of the box when the pulp is withdrawn, so as to reduce to a minimum the amount of slime introduced into the solution-vat when the subsequent wash is thrown back, and also to minimize the loss of valuable solution in cases where, as at

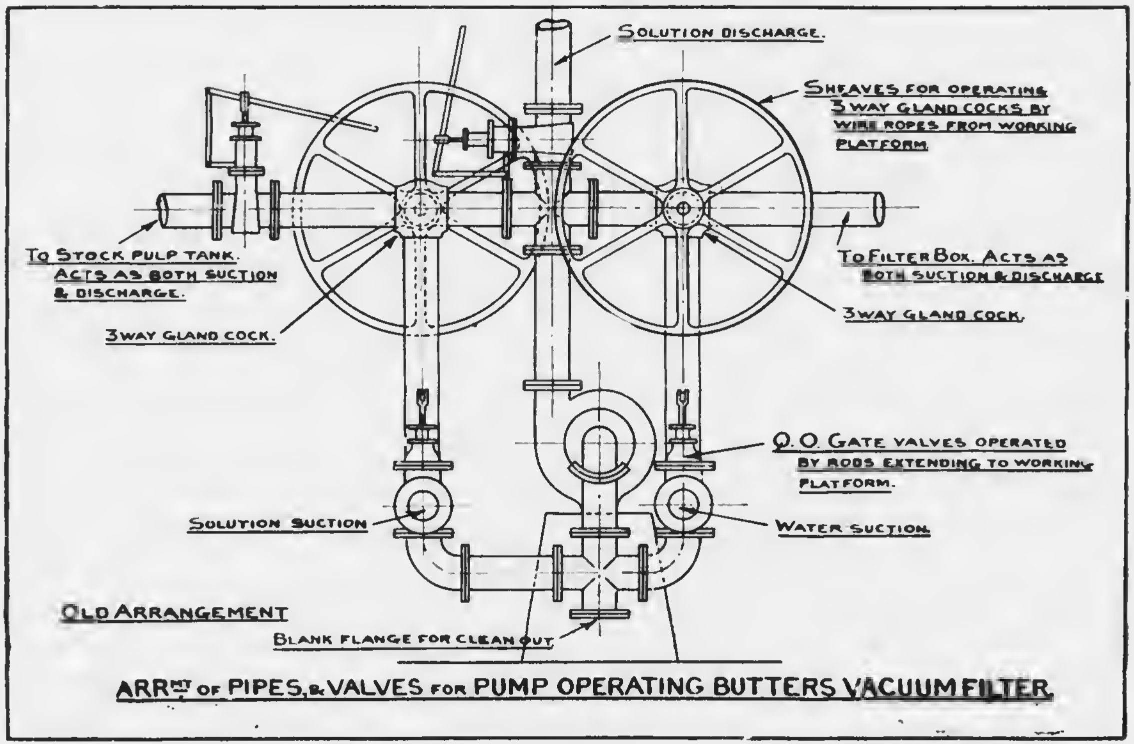

SOLUTION DISCHARGE.
SHEAVES FOR OPERATING 3 WAY GLAND COCKS BY WIRE ROPES FROM WORKING PLATFORM.
TO STOCK PULP TANK. ACTS AS BOTH SUCTION & DISCHARGE.
TO FILTER BOX. ACTS AS BOTH SUCTION & DISCHARGE
3 WAY GLAND COCK.
3 WAY GLAND COCK.
Q. O. GATE VALVES OPERATED BY RODS EXTENDING TO WORKING PLATFORM.
SOLUTION SUCTION
WATER SUCTION.
OLD ARRANGEMENT
BLANK FLANGE FOR CLEAN OUT.
ARRMT OF PIPES, & VALVES FOR PUMP OPERATING BUTTERS VACUUM FILTER.

Fig. 38.

Virginia City, a water-wash follows directly on the withdrawal of the surplus pulp and is discharged with the residue-cakes.

The charge of slime, after being treated by agitation in the ordinary way, but not having received any subsequent washes, is allowed to settle for 24 hours in the treatment vat. When the clear solution has been decanted, the charge is stirred up and run into a deep settlement-vat, of which there are three in use at the present time. Here it settles, usually for two or three days, and from here, after being decanted as closely as possible, it is taken directly into the filter.

This method of feeding the filter works well with the peculiar slime under discussion, but for the more ordinary kinds of slime and particularly when there is much fine silica present, it is found advisable to have the stock pulp-vat fitted with a mechanical stirring-gear to keep the slime from settling too hard for pumping and to give a homogeneous product to the filter.

All transfers in connection with the filtering process are performed by a single 6-in. centrifugal pump (see Fig. 38) with valves and levers so arranged that the man in charge can carry out each operation without moving from one spot. From an examination of the pipe-connections with this pump it will be seen that it will first draw pulp from the feed-vat and throw up through the bottom of the filter-box; next, it will draw from the filter and throw back through the bottom of the feed-vat; it will then draw from either solution-vat or water-tank to fill the filter-box and return the same to where it came from.

This may look complicated on paper, but it is perfectly simple for the operator, and a new hand will learn the whole routine in a couple of days. Within easy reach are, first, the clutch-lever for starting and stopping the 6-in. pump, and another for the 4-in. centrifugal used for emptying the drain solution sump; second, two small winches connected with wire-cord with the two 3-way valves, each having a dial indicator visible from his station to show exactly the position of the valve; third, four iron handles projecting through the floor at his feet, for opening and closing the straight-way valves of the system; these also are marked, so that there may be no mistake. Within three feet of where he stands the filter-box is situated, and close alongside is the vacuum-drum with its mercury-gauge, and gate for opening and closing the main vacuum-pipe; and 30 ft. away, on the same level, is the valve that admits water to the leaves for detaching the cake. Thus, the only occasions on which he need move more than a few feet away from his station are, (1) when he has to go below to manipulate the discharge-doors of the box, (2) when he has to change from one feed-vat to the next one, and (3) when it is necessary to oil or pack one of the pumps.

METHOD OF PROCEDURE.—It is assumed that the filter-box is empty and one of the deep settlement-vats has been charged to its full capacity and has had time for settlement and decantation, and that the pulp therein is composed of about one part of slime to two of solution. This vat has six 12-in. iron plugs in the bottom, connected by iron rods to a screw and wheel attachment at the top of the vat, by which they may be opened or closed from above. The seats of these valves are all connected by tees to the 6-in. pipe of the centrifugal pump.

In order to put the filter in operation, the shift-man first opens up two of the plugs in the deep vat; he then returns to his station and opens the valve between the vat and the pump, and by turning his winches sets the 3-way valves in position to draw from the feed-vat and discharge into the bottom of the filter-box. As soon as the box is full the pump is stopped and the valve to the vacuum-drum is opened, putting each filter-leaf in direct communication with the vacuum-pump, which is now in action. Very soon a subsidence in the level of the pulp is apparent, due to the withdrawal of liquor through the filters; the 6-in. pump is then started again to supply fresh pulp to the box, and this is repeated at short intervals so as to keep the level of the pulp always above the line of permeability of the leaves. The solution, which is drawn from the pulp into the vacuum-drum continually, gravitates into the small sump and is periodically pumped into the precipitation vat, as before described.

The thickness of cake allowed to accumulate on the leaves varies from ¾ to one inch and these seem to be the economical limits; a thickness of over one inch necessitates a disproportionately longer time to collect and wash the cake, because the deposition becomes slower and slower with increased thickness, and the cake is more liable to drop from its support when the pulp is withdrawn; while if less than ¾ in. be used economy is again interfered with owing to the time needed for the subsequent operations being spent over too small a tonnage of material.

The time needed to accumulate a ¾-in. cake varies with the amount of moisture held in the pulp and with the quality of the slime, and may be from 15 minutes to an hour. The progress of formation is estimated from time to time by reaching the hand below the level of the pulp and passing a short stick, such as a wooden match, through the cake and noting the measurement between the canvas surface and the finger and thumb.

As soon as the necessary point has been reached no more pulp is supplied to the box, but the 3-way valves being reversed, the pump is again started, when the suction and delivery lines reverse their functions and the pulp remaining in the box is withdrawn and thrown back into the feed-vat. Immediately after starting this reverse motion of the pulp, a small valve is partially opened; this admits air to the vacuum-drum, and is so regulated that the mercury shall not register over six to nine inches vacuum. The correct height for each kind of slime has to be ascertained by trial, the object being to have it as low as possible consistently with holding the cake in contact with the canvas. This is an important point; during the preliminary experiments it was found that when the box was emptied of pulp and the surfaces of the cakes exposed to the atmosphere, the passage of air through the pores rapidly dried the deposited slime and caused cracks, which rendered a perfect washing of the residue impossible. By lowering the vacuum to a point sufficient to hold the cake firmly in position and yet cause a minimum of air to pass through it, the difficulty was overcome and the cracking prevented.

When the box is nearly empty, the spray of solution is turned on, washing down the sides and bottom, and removing any light slime that may be adhering to them.

As soon as the last of the pulp has been removed, the main slime-valve is closed, and another one is opened to admit precipitated solution to the pump, by which means the filters with their adhering cakes are again completely covered with liquor. As soon as this has been accomplished the air-valve is closed, and the full vacuum a second time thrown on the leaves, so as to draw the wash-solution through the cake and displace the dissolved gold and silver contained therein, the level being maintained, as before, so as to keep all filtering surface covered with solution. The time required for this wash varies with the thickness of the cake and the quantity of solution that it is deemed desirable to pass through it. As a rule it is found that by drawing through an amount of liquor equal to twice the amount of moisture held in the cake a sufficient displacement is effected for practical requirements. In cases, however, where the solution accompanying the pulp is especially rich in gold and silver, a wash equal in quantity to three times that of the moisture in the cake may have to be given. When the washing process has been continued long enough, the vacuum is again lowered, and by re-setting the 3-way valves the remaining solution is returned to the vat whence it came. (See diagram of pump connections, Fig. 37.) The solution-valve is then closed and water is thrown through the common 6-in. pipe until the filter leaves are again submerged; the vacuum is put on for a few minutes to displace any solution that may be lying in the pipes, and is then entirely cut off and the valve is opened; water then enters each leaf through the vacuum connections, throwing off the cake, which drops to the bottom, and cleansing the canvas in preparation for the next charge. All that now remains to be done is to open the four discharge-doors and allow the contents, both water and hard slime, to gravitate to the waste-dam, open the end-doors, and wash out the interior thoroughly with the hose. After closing all the doors the filter is ready for a new charge.

Precipitated solution is not invariably used for displacing the valuable moisture in the cake; when the stock of solution in circulation through the plant is low, it is brought up to bulk by using water instead of solution for this purpose, in this case of course, the necessity for a transference after washing is removed, and when the requisite number of tons of water has been drawn through the cake it is at once discharged, together with all the surplus water remaining in the box.

In localities where water is scarce it is not necessary to waste all the water standing in the box at the time the cake is thrown off; before dumping the residue, this water may be decanted to the level of the slime and stored for further use.

At the Minas Prietas plant of Charles Butters & Co., compressed air alone is used for dropping the cake, and the residue is run out to the dump in cars. The use of water and air together, however, as already described, is considered an improvement.

WORKING COSTS AT VIRGINIA CITY.—The two chief items of expense are power and labor.

1. Labor.—As already stated, the filter requires only one man on each shift, or three men for 24 hours. Taking the rate of wages as $3 per 8-hour shift, the labor bill will amount to $9 for handling 150 tons of slime, or 6c. per ton.

2. Power.—The principal divisions are, the 6-in. centrifugal, the 4-in. centrifugal, the vacuum-pump, and the service-pump for supplying the tank used to furnish hose-pressure for sluicing out the residue. The water used for filling the box and for throwing off the cake is gravitated into the plant and uses no power.

The 6-in. centrifugal makes 1,440 r.p.m., and when running takes 25 h.p. During one operation of the filter this pump runs as follows:

Operation	Minutes
Filling with pulp	12
Maintaining pulp at one level	5
Emptying	12
Filling with solution	12
Maintaining constant level	5
Emptying	12
Filling with water	12
Total	70

The power used is therefore equivalent to 29 h.p. hr. Power, as supplied by the Electric Power Company, costs approximately 1c. per h.p. hr., so the cost of running the 6-in. pump is 29c. for each charge, or 1.54c. per ton.

The 4-in. centrifugal pump makes 1,600 r.p.m. and uses 10 h.p. During one operation it runs 34 min., equal to 6.5 h.p. hr., giving a cost of 5.6c. for each charge, or 0.3c. per ton of slime.

The vacuum-pump runs 2 hr. 20 min. for each charge, taking 8 h.p. or 18.6 h.p. hr., equal to 18.6c., or 1c. per ton.

The water-service pump runs about 10 hr. per day, using 3 h.p., or 30 h.p. hr., which, divided over 150 tons of slime, gives a cost of 0.2c. per ton.

Thus the total cost of power, per ton of slime, is as follows:

	Per Ton
6-in. pump	$0.0154
4-in. pump	0.0030
Vacuum-pump	0.0100
Water-service	0.0020
Total	$0.0304

3. Repairs.—The only repairs of any consequence are those to the 6-in. pump. About once a month this pump needs:

1 pair of liners	$2.00
1 disc	3.35
1 shaft	1.80
Mechanic's labor (2 hr. at 50c.)	1.00
Total	$8.15

This for 4,500 tons gives a cost of 0.2c. per ton. The running expenses will be:

	Per Ton
Labor	$0.0600
Power	0.0304
Repairs	0.0020
Treating leaves with acid	0.0150
Total	$0.1074

These figures were compiled when only one filter-box was in use; since then a second has been installed, and as one man attends to both, the cost of labor has come down to 3c. per ton, showing a total working cost of about 8c. per ton of slime.

NOTES ON WORKING.—It will be noticed from what has been said about washing, that the practice at Virginia City at this time was to use either a solution-wash or a water-wash, but not both on any given charge. The reason for this was that the particular slime being treated was such that it would not stand being twice exposed to the action of the air without cracks forming in the cake. This, however, is not the case with all slimes, and it is usually found that a wash of solution may first be given and then a further period of water-wash. Where possible the latter arrangement is preferable, because, as a rule, when using water alone it is impossible to give a sufficient quantity for effective displacement without overstocking the plant with weak solution, which finally has to be run to waste; and the giving of an adequate wash with barren solution, followed by a quantity of water sufficient to compensate for the daily losses in the stock, and thus

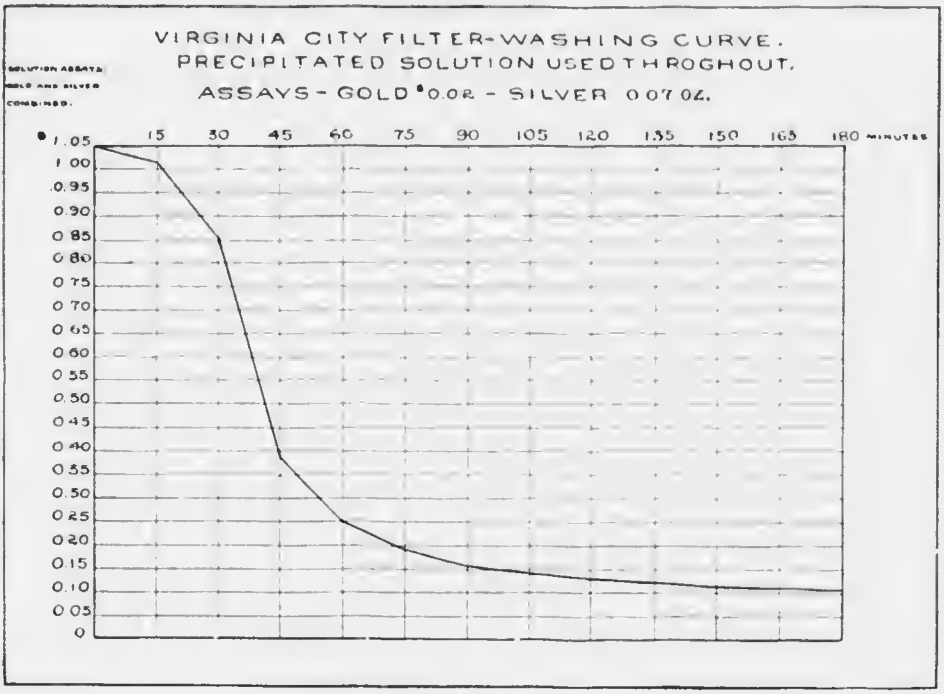


Fig. 39.

displacing as much as possible of the cyanide remaining in the cake, generally results in a better saving of cyanide than is obtained by the Virginia City practice. Another method, where it is not possible to give the double wash, is to add to the wash-solution each day just enough water to maintain the balance of stock liquor in the plant; this gives more uniform results than alternating between solution and water, and is sometimes preferable.

With regard to the time necessary for washing the cake, the important point is the quantity of solution that is drawn through it; in a given time this varies between wide limits with different classes of slime. Fig. 39 and 40 are two charts of the results of the wash at Virginia City. There is an interval of two or three weeks between them. In one of them solution alone was used, and in the other, water. The rate of percolation was 18 tons per

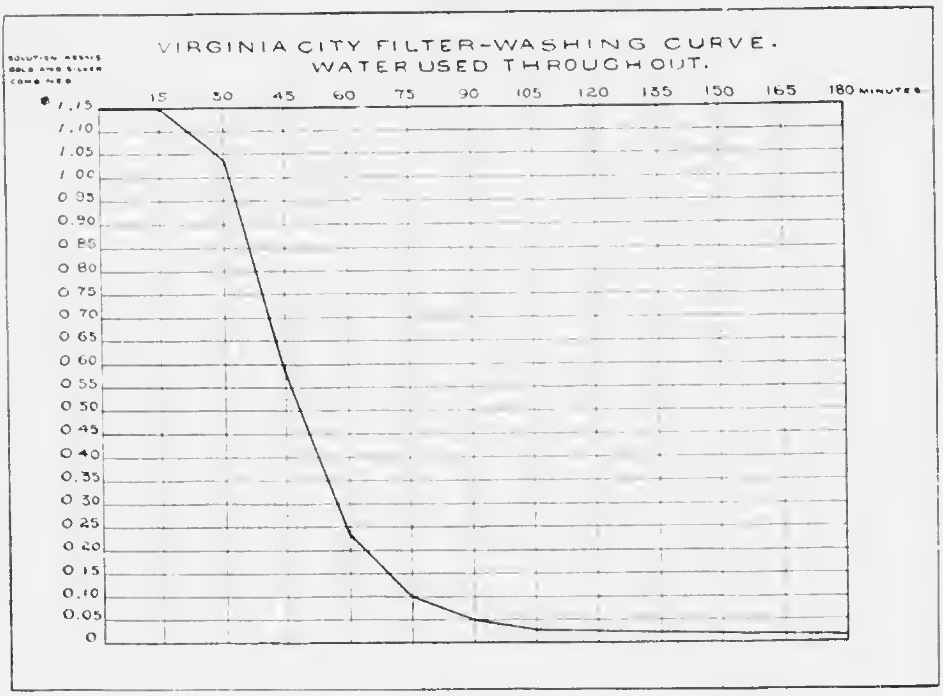


Fig. 40.

hour in each case; weight of dry slime, 19 tons; moisture in the cake, 40 per cent. The samples of solution were caught in a trap below the main vacuum-pipe, capable of holding at least 20 assay-tons, which quantity was used for each assay. Fig. 41 shows a similar chart, made at the Butters Divisadero Mines, in Salvador. Here the ore is ground in tube-mills to an agitation product of which about 85 per cent will pass a 200-mesh screen, and the whole of the pulp finally reaches the filter. In this instance the rate of

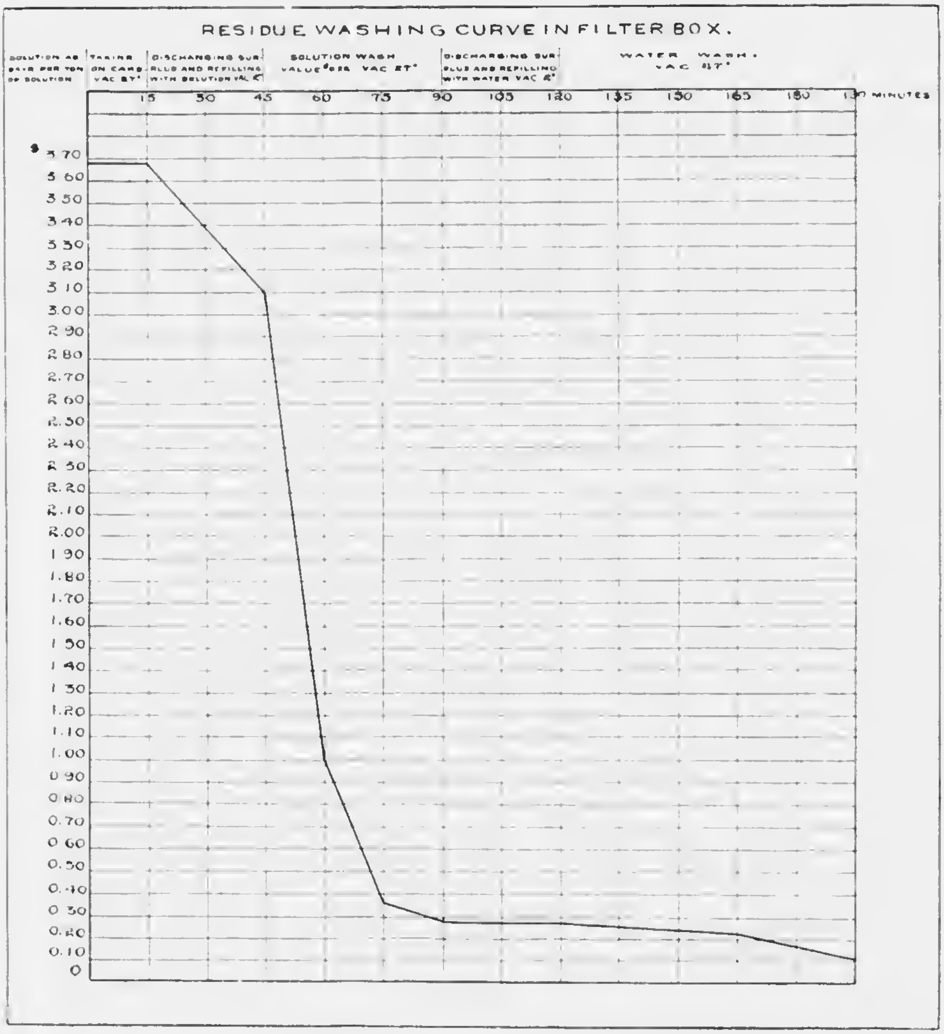

RESIDUE WASHING CURVE IN FILTER BOX.
SOLUTION WASH
WATER WASH.
15 30 45 60 75 90 105 120 135 150 165 180
MINUTES
$ 3.70
3.60
3.50
3.40
3.30
3.20
3.10
3.00
2.90
2.80
2.70
2.60
2.50
2.40
2.30
2.20
2.10
2.00
1.90
1.80
1.70
1.60
1.50
1.40
1.30
1.20
1.10
1.00
0.90
0.80
0.70
0.60
0.50
0.40
0.30
0.20
0.10
0

Fig. 41.

percolation was 28 tons per hour; quantity of dry slime, 18 tons; moisture in the cake, 28 per cent.

When considering the contents of the solution draining from the filter just before discharging, it will be obvious that this assay does not usually represent the soluble gold and silver left in the cake. Where water or barren solution is used for the wash no portion of the residual moisture will assay as high as that which has already come out, and the average of it will be much lower; for example, if the last solution from the cake should assay, let us suppose, 20c. per ton, the moisture remaining behind would probably average only 10c. or less, and as there will be only half a ton of it contained in a ton of dry slime the assay of the residue would be raised thereby only 5c. per ton, an increase that could hardly be detected by the assay methods ordinarily in use. Five cents per ton, however, is 5c., whether it can be shown by the assay or not, so that when fixing the time necessary for washing, it will be well, instead of depending on what is usually known as the 'washed residue assay,' to give a trial wash of excessive duration once or twice a month, taking samples of the issuing solution periodically, and plotting as a curve the figures so obtained; it will then be easy, if the rate of percolation is known, to set the value recovered at any given point against the cost of obtaining it, and in this way to determine the profitable limit to place on the period of washing.

The question of saving cyanide will not enter largely into this matter of wash-duration, because the cyanide can only be displaced by water, and the amount of the latter used, as already pointed out, will be determined by the daily losses of moisture from the plant, and thus cannot be varied at will except at the expense of running to waste weak cyanide solution from some other part of the plant. Several curves showing the progressive elimination of cyanide from the cake have been made, but as they were not consistent with the gold and silver curves made at the same time they are not given here, and the matter is still under investigation.

The following is a table of the time needed (with a ¾-in. cake) for the various operations, taken directly from the filter log-book at Virginia City:

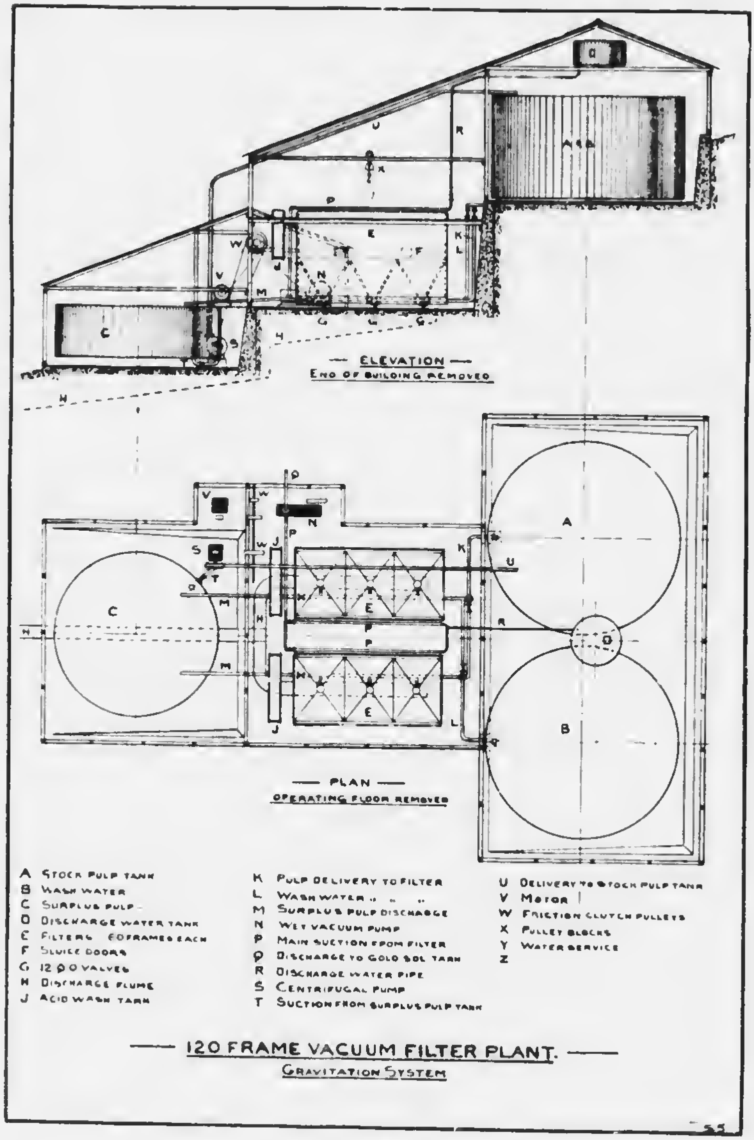
	Hr.	Min.
Filling and taking on cake	0	45
Removing surplus pulp and re-filling with solution	0	25
Drawing wash-solution through cake	1	15
Replacing solution by water, throwing off cake, discharging, sluicing out, and re-closing doors	0	35
Total time of operation	3	0

The percentage of moisture held by the cake before being detached from its support is from 35 to 40% in this case, though the average for an ordinary slime would be nearer 33 per cent.

The box contains 94 filter-leaves, and each leaf carries about 400 lb. dry slime, giving a capacity of 19 tons at each operation, or 150 tons per 24 hours. With certain kinds of slime the capacity of a filter of this size may be as much as 25 tons per charge, making 200 tons or more per 24 hours.

The consistence of the pulp fed to the filter has an important bearing on the efficiency of the latter; the thicker it is, the better it will suspend fine sandy material and form a homogeneous mass giving a uniform cake, and also the less time it will need to form the cake and therefore the greater will be the capacity of a given filter unit in a given time. Of course, there is a limit in the consistence, beyond which the thickness would cause trouble by clogging the pipe-lines, and a useful point to aim at is a specific gravity of from 1.3 to 1.4.

At the start there was one point in connection with the working of the filter that was not apparent. It was found that after a few months the permeability of the leaves began to grow less, thus necessitating a longer time for loading and also for passing the wash. After a careful examination, it was traced to a formation of calcium carbonate in the fibre of the canvas and more especially on its exterior surface. This was found to be easily removed by immersing the leaves in a dilute solution of hydrochloric acid. A rectangular wooden vat was prepared capable of holding seven or eight leaves at a time, and this was filled with a solution containing about two per cent commercial muriatic acid in water. Seven leaves are removed each day, their places being filled by



ELEVATION
END OF BUILDING REMOVED
PLAN
OPERATING FLOOR REMOVED
A STOCK PULP TANK
B WASH WATER
C SURPLUS PULP
D DISCHARGE WATER TANK
E FILTERS 60 FRAMES EACH
F SLUICE DOORS
G 12 Ø O VALVES
H DISCHARGE FLUME
J ACID WASH TANK
K PULP DELIVERY TO FILTER
L WASH WATER " " "
M SURPLUS PULP DISCHARGE
N WET VACUUM PUMP
P MAIN SUCTION FROM FILTER
Q DISCHARGE TO GOLD SOL TANK
R DISCHARGE WATER PIPE
S CENTRIFUGAL PUMP
T SUCTION FROM SURPLUS PULP TANK
U DELIVERY TO STOCK PULP TANK
V MOTOR
W FRICTION CLUTCH PULLEYS
X PULLEY BLOCKS
Y WATER SERVICE
Z
120 FRAME VACUUM FILTER PLANT.
GRAVITATION SYSTEM

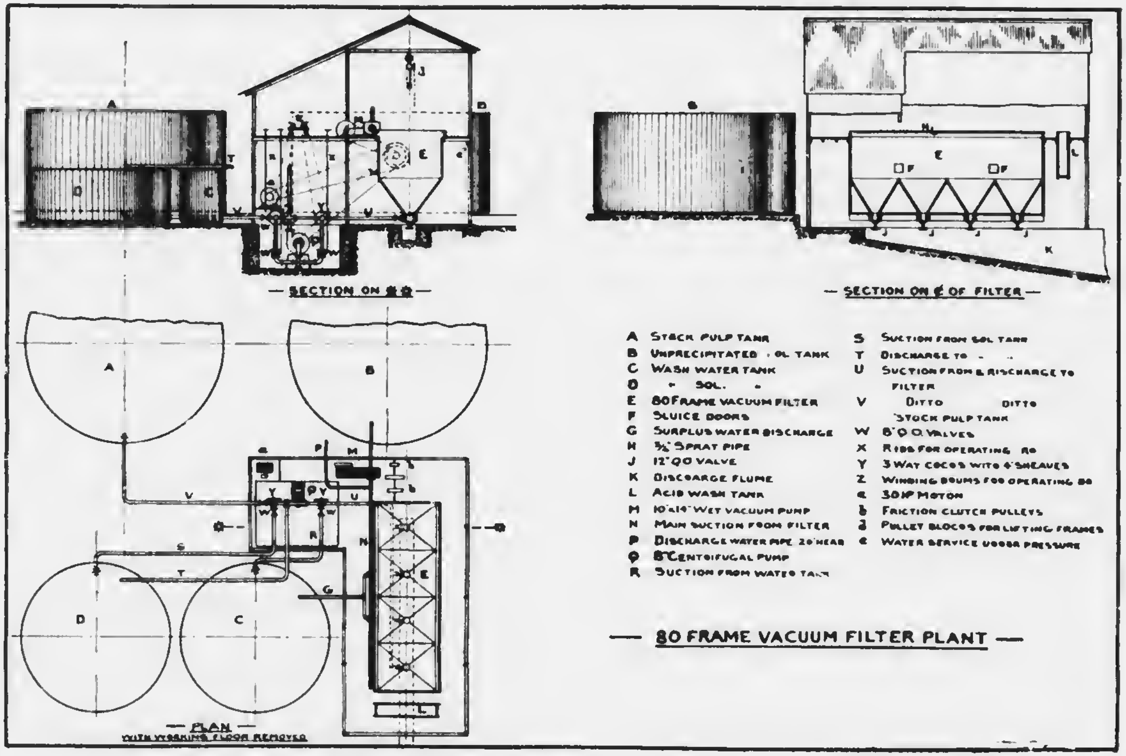
Fig. 42.

spares. They are placed in the acid bath and allowed to remain for about six hours, at the end of which time they are removed, drained, and placed in a water-tank to stop the action of the acid. In this way each leaf is treated about twice per month.

If this is not sufficient to restore the leaves to a perfect working condition, the strength of the acid may be increased to five or six per cent, and a vacuum arrangement attached to them for drawing the solution through the fibre repeatedly, until all the deposit is removed. Instead of continually adding acid to re-strengthen the bath, it is better to turn the whole liquor to waste at short intervals, because it is liable soon to get charged with salts, which, if not properly washed out from the leaves with water, may be precipitated in the fibre of the canvas when next it comes in contact with the alkaline solution.

To get the highest efficiency from this filter it is important to make the transfer of pulp and solution as rapidly as possible, and this for two reasons: (1) It materially increases the capacity of a given filter-unit and thus decreases the working costs per ton of slime in the item of labor, and (2) it lessens the liability of the cake to crack when exposed to the air, and in the case of slime that has an exceptional predisposition to crack, it renders it possible to give a double wash, of solution, followed by water, when otherwise only a single wash could have been applied.

For a rapid transfer of the pulp and solution the gravity system of filling and emptying the box gives excellent results. The general arrangement is shown in Fig. 42, though there a water-wash only is provided for. Usually a second vat at the level of C is provided to receive the surplus solution, and also a third vat for surplus water where the double wash is given and where the water is sufficiently valuable to be saved for re-use. The method of working is to fill the box with pulp by gravity through two or more 6-in. pipe-lines from the feed-vat above; then after taking on the cake to let out the surplus through the discharge gates into vat C, which has a light stirring gear to keep the slime in suspension while it is being thrown back by a 4-in. centrifugal pump to the feed-vat above. The same process is then repeated with the wash solution, and again with the water if necessary, each in turn being thrown back to the higher level by the same 4-in. pump. In this way the box may be filled in four or five minutes and emptied in three minutes, so that the cake is exposed to the air only



SECTION ON
SECTION ON ℄ OF FILTER
PLAN
WITH WORKING FLOOR REMOVED
A STOCK PULP TANK
B UNPRECIPITATED SOL. TANK
C WASH WATER TANK
D " SOL. "
E 80 FRAME VACUUM FILTER
F SLUICE DOORS
G SURPLUS WATER DISCHARGE
H ¾" SPRAY PIPE
J 12" Q.O VALVE
K DISCHARGE FLUME
L ACID WASH TANK
M 10"x14" WET VACUUM PUMP
N MAIN SUCTION FROM FILTER
P DISCHARGE WATER PIPE 20' HEAD
Q 8" CENTRIFUGAL PUMP
R SUCTION FROM WATER TANK
S SUCTION FROM SOL. TANK
T DISCHARGE TO " "
U SUCTION FROM & DISCHARGE TO FILTER
V DITTO DITTO STOCK PULP TANK
W 8" Q.O. VALVES
X RODS FOR OPERATING
Y 3 WAY COCKS WITH 4' SHEAVES
Z WINDING DRUMS FOR OPERATING
a 30 HP MOTOR
b FRICTION CLUTCH PULLEYS
d PULLEY BLOCKS FOR LIFTING FRAMES
e WATER SERVICE UNDER PRESSURE
80 FRAME VACUUM FILTER PLANT

Fig. 43.

about eight minutes for each transfer; this is a great advantage where a slime shows a tendency to crack. The arrangement is chiefly applicable where the mill-site is on the side of a hill; for level sites the pump-system is usually more suitable. By the latter, of course, results similar to those of the gravitation system may be obtained by using a sufficiently large pump and pipe-line, but where the site is favorable to gravitation this has the advantage of a rapidity of transfer without the sudden switching on and off of a heavy load at short intervals with its attendant strains on the source of power and the machinery in general, the power used being comparatively small in amount and evenly distributed over a long period of time. The method has been in use at the cyanide plant of the Butters Copala Mines, in Sinaloa, Mexico, for over 12 months and has given entire satisfaction.

The Butters patent filter is now well past the experimental stage, and its advent may be said to mark the beginning of a new era in the history of ore-slime treatment, and to open up possibilities unknown a few years ago. The time may even be not far distant when slime from gold ores will be treated by cyanide solution directly in the vacuum-filter, so that all the expensive agitation and decantation machinery now in use can be discarded.

It would perhaps be too sanguine to hope that the slimes of silver ore may also be treated in the same way in the near future, because, as far as our present experience goes, such ore requires a much longer time in contact with cyanide than would be practicable in any filter, but as soon as some means is found of accelerating the solution of silver in cyanide, then there is little doubt that agitation and decantation will give place to treatment by leaching in vacuum-filters.

OLD AND NEW METHODS AT GUANAJUATO, MEXICO

By T. A. Rickard

(June 29, 1907)

Reference has been made to the milling practice of the Guanajuato Reduction & Mines Co. in speaking of the Bustos plant. It deserves further consideration. Although the property was acquired in January, 1904, it was not until February, 1905, that it was decided to build 80 stamps, rushing the erection of five of them so as to afford experimental data during the construction period. Many tests on a large as well as a small scale, had already demonstrated that a high extraction of both silver and gold could be obtained from the ore by cyanidation, and the experimental plant became of great service in testing ores from different parts of the company's properties, as well as to suggest the detail manipulation best adapted for the main plant, then under construction. The mines are a mile from Guanajuato, and there are no streams available for disposal of the tailing; nor is there the space necessary for accumulating residue on a large scale. It became necessary to discharge the tailing into the main stream of the district one mile from the mines, and this meant the transportation of all ores through the heart of the City over an expensive railroad system or the complete separation of the stamp-mill from the cyanide plant, the latter to be placed upon the main stream.

The Homestake system, of conveying the tailing in a cast-iron sewer-pipe was adopted; it is an 8-in. water-pipe of bell and spigot type, asphalted, laid for its first few hundred feet at a grade of 3%, then flattening to 2½%. As it was desirable to settle and return for re-use as much of the water coming from the concentrators as possible, allowing the thick pulp only to pass through the pipe, and to determine to what point such thickening was possible, several hundred feet of the pipe to be employed in this work were put together and laid out upon the actual grade. Arrangements were made to circulate any given quantity of sand, crushed in the stamps of the experimental plant already installed, through this pipe-line at any desired degree of dilution. Pulp was first tried at a normal dilution of 8 of water to 1 of sand, just as it came from the tail of the concentrator tables; the water was then decanted by successive steps, thickening the pulp, and several runs were made under these varying conditions. It was desired to remove a maximum of one-half of the original water, and in the experiments the 8 to 1 pulp was reduced to 2.5 to 1, and so successfully that not only was the pipe not clogged with sand, but the pulp

Fig. 44. Pipe for Conveying Tailing.

at that thickness had such rapidity of flow that it readily carried nuts and other heavy objects without interrupting the stream. Tests made by Mr. Carlos Van Law proved that pulp which has passed through a 30-mesh screen, with water in the proportion of $7\frac{1}{2}$ to 1, will flow through a launder of square cross-section, made of rough boards, set on a grade of $1\frac{1}{2}\%$. With a V-shaped wooden launder, such pulp will flow at less grade and with less water. The area of the wet perimeter is the factor.

This problem settled, the process was outlined as follows: The ore is transported over a railroad from the mines in hopper-bottom cars, which discharge into a large bin at the crusher plant. It is then passed through a No. 5 D. Gates crusher to 2-in. size, discharging over a picking-belt for removal of waste, then re-crushed by a short-head No. 4 Gates crusher to 1-in. ring, then removed by conveyor-belts to the mill-bins, where it is distributed by cars, this (owing to cheap labor), proving more economical than the use of a system of conveyor-belts over the top of a bin with automatic trippers. The mill-bin has a capacity for five days, which, together with the large bin at the crusher plant, affords sufficient storage of ore.

The ore is then crushed under eighty 1,050-lb. stamps of Allis-Chalmers make. The mortars are set on a concrete foundation, the anvil block being cast as an integral part of the mortar. The latter has a steel liner, two inches thick, but there is 13.5 in. of metal below the liner. Between the anvil-block and the concrete a quarter-inch sheet of rubber is spread. The mortar has a broad base—36 in. wide—provided with strong ribs at the sides.

The 30-mesh pulp resulting from the stamping is concentrated over Wilfley tables, a considerable amount of middling produced being removed for re-grinding in an Abbé tube-mill and subsequent treatment over separate concentrator tables. The tailing from all the machines drops into concrete launders and passes to the cones where about half of the water is removed for use in the mill. The thickened tailing then enters an 8-in. cast-iron pipe, laid on the grade above mentioned, and goes to the classifiers in the Hacienda Flores, situated upon the main channel of the Guanajuato river. The classifiers are all of the Homestake type, two sets of cones being used, the lower one taking the bottom product of the upper. The overflow from both sets of cones goes to the slime-plant, the lower cone having an ascending current of water.

The sand coming from the bottom of the lower cones is distributed, by the Butters & Mein device, into either of two receiving-vats, each of 350 tons capacity. These were planned to serve an ultimate capacity of 500 tons daily, with the slime separated.

The receiving-vats are alternately filled and drained, the discharge being made through bottom gates onto ascending conveyor-belts, which pass over the centre of the line of eight leaching-vats, each of the same size as the receiving-vats. In these a 14-day treatment is given with 0.5% cyanide solution, the sand being then washed and ultimately discharged on conveyor-belts (running under the vats), which deliver either into the river during the rainy season, or to elevated storage-piles during the dry season, to be sluiced during the succeeding rainy season.

The slime from the classifying cones is treated by agitation with mechanical stirring and large centrifugal pumps, whic draw from the bottoms of the vats and discharge over the tops, the total time of treatment being four days. After the final wash the slime is pumped into settling-vats 30 ft. high, where a further decantation occurs before the slime is discharged, with a very small percentage of moisture, into the river.

The entire plant, both the stamp-mill and cyanide annex, is designed so that it can be doubled, when the stamp-mill will take a back-to-back form, 80 stamps with their concentrators being on either side of the bins. The classifying-cones and the receiving-vats of the cyanide annex are of sufficient size already for a 500-ton plant, it being necessary only to add another line of eight leaching-vats and the corresponding slime-vats to bring the cyanide plant to the larger capacity mentioned.

For roofing, corrugated galvanized iron on a steel frame is preferred. As there is no load of snow to fear, it is possible to use a light roof-truss. The native tile is cheaper, but it requires a heavier construction, and does not afford as complete protection. The Northerner will remark the lavishness of the masonry about the mines and mills in Mexico. It is the cheapest kind of construction and the native wood is usually poor; it will twist and untwist as the dry and wet seasons succeed each other, making it an unsatisfactory structural material.

At the time of my visit there were 120 stamps in operation in the Guanajuato district, and there were 205 tons being treated daily by cyanidation and 25 to 30 tons by *patio*. Thus does the new drive out the old. In 1887 there were 34 *patios* at work; now there are only two.

In regard to cost in the *patio* process, I have the following data from the Hacienda de San Julio at Pachuca:

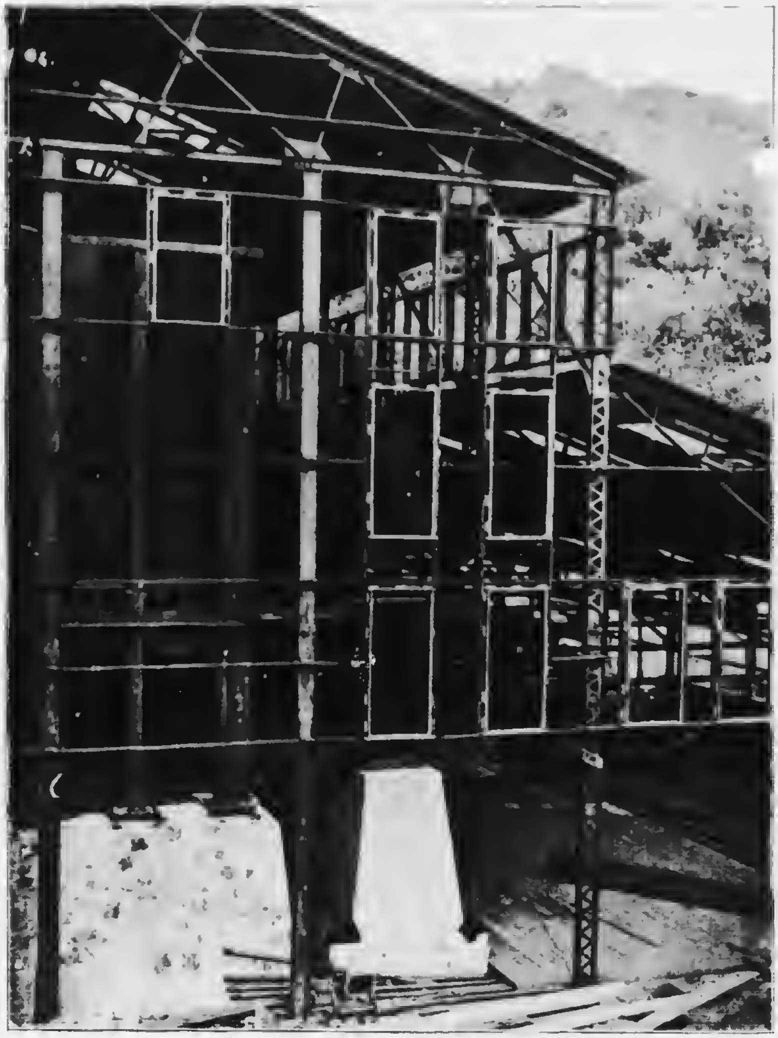


Fig. 45 Bustos Mill, in Course of Construction.

	Pesos
Crushing, to pass 60-mesh in Chilean mills, with overflow discharge	3.60
Maintenance	0.50
Salt, 5%, or 50 kg. per ton	1.75
Copper sulphate. Loss, 0.5%	1.25
Mercury. Loss, 1¼ kg. for each kilogram of silver	4.38
Transport from the mine	1.08
Total	12.56

The ore is bought on the dump, therefore the cost of transport is included. Salt costs 35 *pesos* per ton; copper sulphate 250 *pesos* per ton, and mercury 3.41 *pesos* per kilogram. The average losses for the year were 1.18 to 1.50 kilogram. Per kilogram of silver extracted; the loss of silver being 6.13 to 10.92% on the clean ore, and 15.83 to 29.13% on the galena ore. The *patio* process waits on the completion of the chemical reactions, and it is therefore continued until extraction ceases. Time is not considered, in winter it requires 20% longer by reason of the lower temperature of the air.

At Parral, in Chihuahua, with the Russell process, using hyposulphite, the cost of lixiviation and roasting is 11 *pesos*, but the recovery is not as high as it is with amalgamation in the *patio*, where the cost is 11 to 14 *pesos*, varying according to the manganese content of the ore.

At Pachuca, by Mexican methods, the cost of mining and sorting ore amounts to 15 *pesos* per ton; the transport to the *patio* and the treatment there makes a further cost of 14 *pesos*, not including losses or the expense of marketing the product. By stamp-milling, pan amalgamation, and concentration, the cost at Guanajuato was 8 *pesos*; and now by stamp-milling and cyanidation the cost is 5.85 *pesos*. That of mining and development is 3.50 to 4.50 *pesos*, so that the total present cost is about 10 *pesos*, or $5 per ton.

At the time of my visit there were about 200 men, women, and children in the Anglo-American colony at Guanajuato, the American element predominating. Of the 125 men employed, 75 to 80 were technical men, of good training. This made a strong piece of mental machinery for industrial development.

CYANIDE PRACTICE AT THE HOMESTAKE MILLS

BY F. L. BOSQUI

(July 6, 1907.)

Apropos of recent descriptions of filtering processes, including the Butters, Moore, and Ridgway systems, some account of present Homestake practice, compiled from notes taken on a recent visit to the property may be of interest. By way of preface, and to refresh the reader's mind as to the main facts, the Homestake

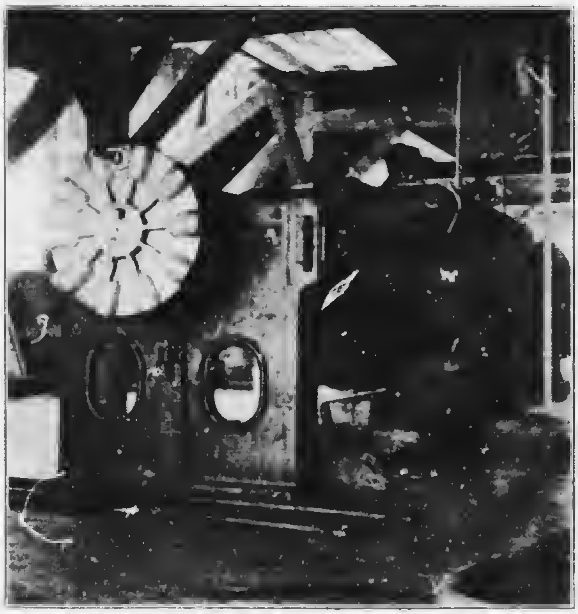


Fig 46 Precipitation-Presses.

plants at Lead, in the Black Hills of South Dakota, consist of six stamp-mills, containing an aggregate of 1,000 stamps and crushing about 4,000 tons per day. The most interesting feature of the mill-practice is the amalgamation. Each battery is provided with four full-sized plates, in series, each plate being 54 by 144 in. and $\frac{1}{8}$ in. thick. The first is plain copper, the last three are silver-plated. This addition of three silver plates, giving to each 10 stamps of the Amicus (240-head) mill, a total plate-area of 600 sq. ft., and to the other mills an average of 360 sq. ft. per 10 stamps, is a comparatively recent innovation, and has proved an excellent one, increasing the recovery by amalgamation approximately $200,000 per year

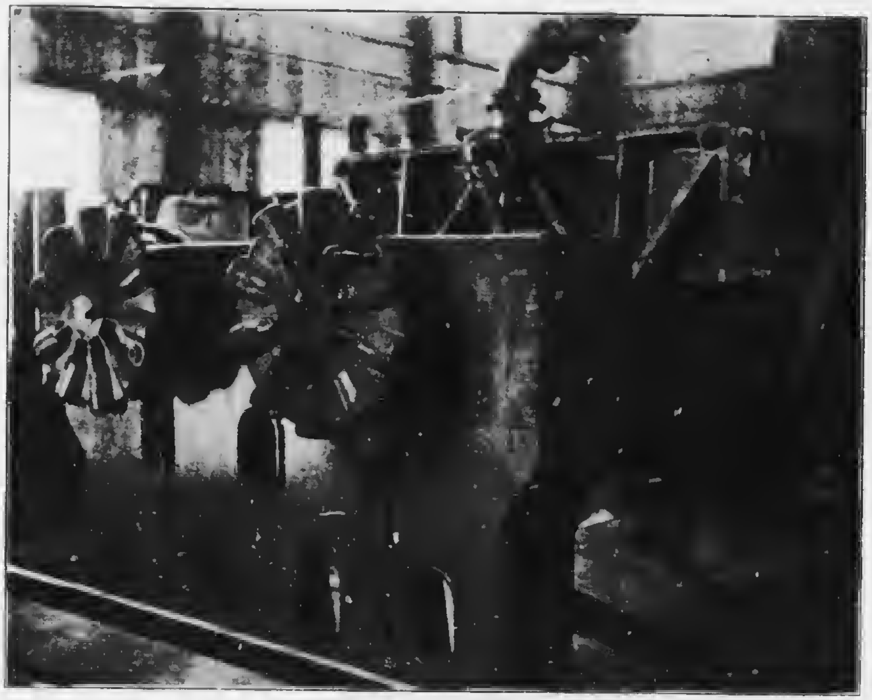


Fig. 47. Screw End of the Slime-Press.

above the annual recovery when only one 12-ft. plate was used. The saving by amalgamation is between 70 and 75 per cent.

The leachable portion of the tailing from the stamp mills (640 stamps), consisting mostly of ore from the deeper levels, is treated, after slime separation, at the Lead plant, known as Cyanide No. 1. This plant has a capacity of about 1,800 tons per day. The oxidized surface ore is crushed in the mills at the north end of the property (360 stamps) and the sand treated in the annex known as Cyanide No. 2, at Gayville, two miles distant from Lead. This smaller plant handles about 800 tons per day. Both plants were designed and installed under the direction of C. W. Merrill, who

Fig 48. Discharge from Two Presses.

devised the process and who recently installed the filter-press slime-plant to be described herewith. The practice at Cyanide No. 1 has been well described by Mr. Merrill himself* and needs no further comment.

*Trans. Amer. Inst. Min. Eng., New York meeting, 1903.

The systen. of slime separation in use at these plants has been criticized by some of the advocates of South African methods as less effective than the old pointed-box system. But at the Homestake, the choice of cone-classification has been emphatically jus-

Fig 49. The Great Open-Cut of the Homestake.

tified by the results. An elaborate sizing was not necessary and was not attempted. Exigencies of first cost, space, labor, etc., precluded double treatment: and to make the enterprise profitable, it was essential that all the sand be treated as a separate product, and in one operation. To those familiar with the difficulty of obtaining, without re-handling the pulp, a uniform charge of hydraulically distributed sand that will leach rapidly, the appended figures will seem incredible.

The leachable sand has the following average texture:

Coarse (remaining on 100 mesh) 30 per cent.

Middling (100 to 200 mesh) 32 per cent.

Fine (passing 200 mesh) 38 per cent.

Through this product the solution percolates at the rate of from three to four inches per hour.

When we consider that all the separated slime passes through 200 mesh, and that this product is in so fine a state of division that 90% of its value is extracted in six hours by filter-pressing, the merits of the Homestake classification system require no further emphasis. The cost of treatment at Cyanide No. 1, which Mr. Merrill gave in 1903 as 35c., has since been reduced to 26c. per ton. At Cyanide No. 2, the product treated is much lower in grade, averaging only 85c. per ton. This plant was the more recently constructed of the two, and is the more perfect, not only in its arrangement, but in its facilities for maintaining a low cost of treatment. A summary of the operating cost for six months is given below.

COST RECORD AT HOMESTAKE CYANIDE MILL NO. 2. FOR LAST SIX MONTHS OF 1906.

	July.	Aug.	Sept.	Oct.	Nov.	Dec.
Total tons of sand treated	24,913	25,644	24,496	24,352	24,190	25,398
Total operating cost	$4,165	$4,609	$3,941	$4,380	$4,122	$3,274
Cost per ton, classification	$0.018	$0.017	$0.017	$0.017	$0.016	$0.023
" " " treatment	0.094	0.093	0.079	0.086	0.090	0.080
" " " precipitation	0.026	0.021	0.022	0.021	0.019	0.021
" " " power	0.021	0.041	0.031	0.048	0.035	0.037
" " " assaying, refining, etc.	0.008	0.007	0.012	0.008	0.010	0.007
Total cost per ton treated	$0.167	$0.179	$0.161	$0.180	$0.170	$0.168

Total tons treated for the six months	148,993
Total operative and treatment cost	$25,493.64
Average cost per ton treated	$0.171

AVERAGE SIZING OF SAND.

Per cent.		Mesh.
37.55	coarser than	100
23.00	between	100 and 200
39.45	finer than	200

I have mentioned the slime separation as a most interesting feature of these plants. Another novel feature is the periodic introduction of air into the pulp for the purpose of providing oxygen to the subsulphide of iron (pyrrhotite)—one of the most troublesome constituents of the ore—and so maintaining the dissolving power of the solution, which would otherwise be robbed of its oxygen by the iron. Both of these big plants are remarkable for the order and neatness exhibited in every department. Operations have been reduced to so simple a system that they seem to be working automatically; yet if one glances behind the scene, he

Fig. 50 Triplex Solution Pump with Motor Double-Wound to Give Full and Half Speeds. Shows Effluent Lixiviant from Slime-Presses.

finds an able corps of assistants continually studying special problems, looking to the reduction of costs and increased efficiency.

From the two leaching plants, about 1,600 tons of slime have been run to waste, of an average value of between 40c. and $1.20 per ton. In working out a method for treating this product, Mr. Merrill considered the various slime processes now in favor, but discarded them all as unsuited to Homestake conditions. In treating an 80c. slime, a system requiring the two operations of agitation and filtration in separate vessels was not to be thought of; and the only device in which treatment and filtration could be performed in one operation was the filter-press; but the cost of operating the old type of filter-press was prohibitive, and this condition led to the development of what is known as the Merrill press, which essentially differs from the old type in that it can be discharged without drawing the plates and frames apart. Several good descriptions of the Merrill press and process have appeared

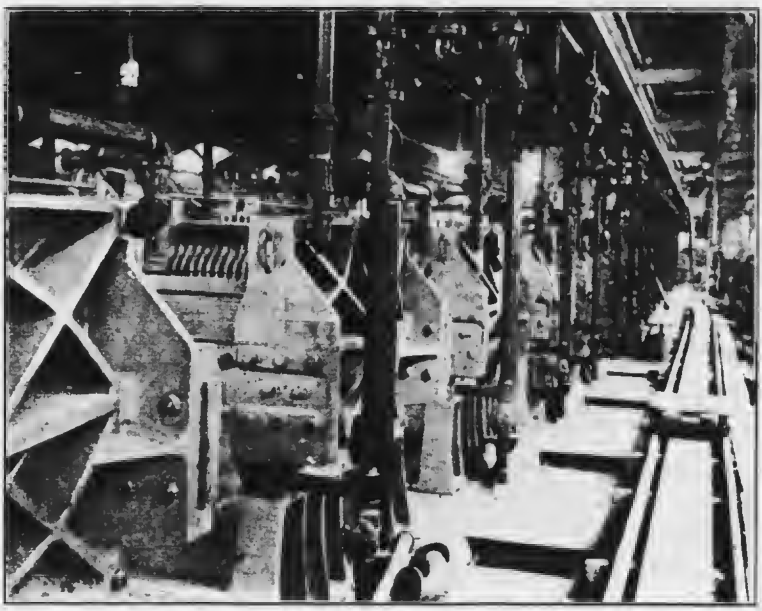


Fig. 51. Front View of Battery of 20-ton Slime Presses.

in the mining periodicals and I shall not attempt here more than a brief account of the plant.

At the beginning, a 10-ton press with 4 by 6-ft. frames was installed for experimental purposes and was in use continuously for 18 months. In one run 131 charges were treated, on which the following data are available:

Average value of slime before treatment, 91c.; after treatment, 10c. Extraction by assays, per ton treated, 90 per cent or 81c.

per ton. Recovered in precipitate, per ton treated, 91 per cent or 83c. per ton. The treatment was simply lixiviation in the press, without agitation. Amount of solution used to leach one ton of slime, 0.73 tons; amount of water used for sluicing, four tons to one ton of slime; thickness of cake, 4 inches

The large plant recently started, is situated at Deadwood. The slime, after partial dewatering, in cone-bottom clarifying vats, which reduces the proportion of water to solid, to the ratio of three to one, is conveyed in two pipe-lines to the slime-plant, as follows: A 12-in. pipe carries it from Cyanide No. 1, a distance of 3¾ miles, at a minimum grade of 1½%; a 10-in. pipe carries it from Cyanide No. 2, a distance of two miles, at a grade of 1½%. The plant is built on a steep hillside and consists of five levels. The uppermost contains the apparatus for slaking lime and feeding it automatically into the stream of slime; the next department contains two large cone-bottom accumulation-vats; the next floor is occupied by the solution storage-vats and precipitation-presses; the next by the slime-presses; and the lowest by the precipitation-vats and dumps.

Two small vats are provided for slaking lime. Their contents are drawn as required to a screen-covered box where the undissolved lumps are separated. This box overflows into an agitator, from which the milk-of-lime is continuously discharged into the main slime-stream at the rate of 5 lb. lime per ton of slime. The two sludge storage-vats are 26 ft. diam. and 24 ft. deep with a 47° conical bottom. The discharge from the sludge-vats passes directly to the filter-presses under a pressure of about 30 lb. through a 10-in. main, which extends through the whole length of the press-building. Between each pair of presses this main branches into a longitudinal 10-in. distributing pipe, which in turn sends two 4-in. branches to each press. These small branches communicate with a continuous channel in the press at the centre of the top, 4 in. diam., by means of which the slime passes to the filter-chambers. Other channels are provided as follows: One at each upper corner, 2½ in. diam., for the entrance and exit of air; one at each lower corner, of same diameter, for entrance and exit of solution; and one large continuous channel extending along the centre of the bottom, by means of which the spent slime is sluiced out. This channel is 6 in. diam., and along its top is suspended a 3-in. pipe extending the length of the press. This pipe is provided with 92 nozzles, 1 in. long and 5/32 in. diam., each of which delivers a stream into one of the 4-in. chambers, under a head of about 50 lb. By a special mechanism, a reciprocating motion is imparted to this pipe, causing it to revolve through an arc of 210°, so that each small nozzle plays against the compact slime cake, removing the cake completely and cleansing the compartment in about 45 minutes. The discharged slime

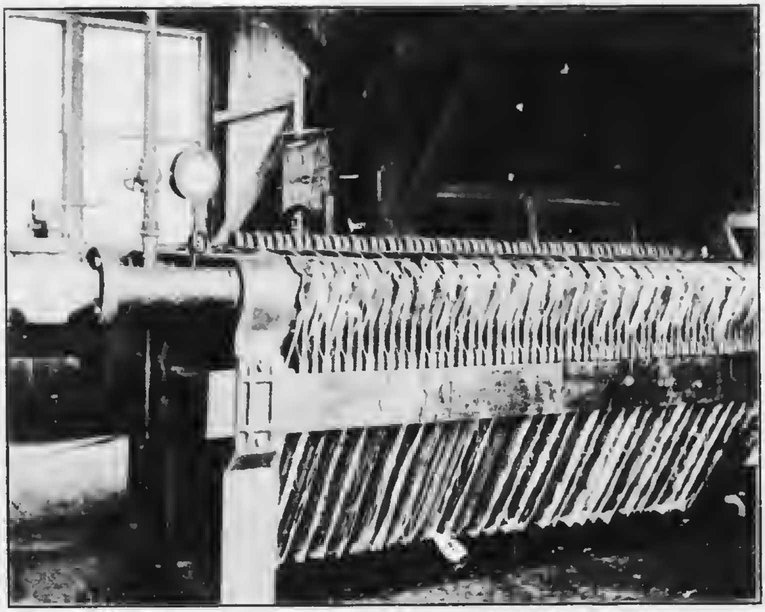


Fig. 52. Merrill Zinc-Dust Precipitation Press.

leaves the press through the crescent-shaped space between the 4-in. pipe and the 6-in. channel, the latter being sealed during the operations of filling and leaching.

The operations within the press consist of leaching with solutions of 0.1 and 0.04 per cent strength, and the aeration of the cakes so essential to good extraction in the Homestake ore. The whole operation, exclusive of filling and emptying, occupies about six hours. At the time of my visit (January, 1907), five of the 24 presses were in operation and working most satisfactorily; the remaining presses were partly set up and were being put into use as fast as the plates arrived from the factory. Under date of Feburary 22, Mr. Merrill writes: "We are now treating at the rate of about 12,000 tons per month and will increase this by several thousand tons per month until our full capacity of 50,000 tons is attained. Cost data are not obtainable yet, but will not exceed 25c. per ton for all items."

Mr. Merrill submits the following data for January, 1907;

Tons treated	7,670
Assay-value ....$	0.85
Recovery in precipitate............	0.68
Carried over in unprecipitated solution	0.09
Total recovery (equivalent to 90 per cent).. $	0.77

At the present time (May 1), eleven of these presses are working continuously, and with results confirming those of the experimental plant given above.

I have already stated* what I conceive to be the limitations as well as the merits of Mr Merrill's process. In all fairness it may be said that each of the filtering systems now in vogue possesses distinct merits of its own. The supreme test of the metallurgist in these days of many inventions is to select the right process for his purposes, that is, the one uniting the greatest number of advantages when applied to his special problem. Of Mr. Merrill's system it may be said finally and emphatically, that it is an established success, and will consequently have a wide range of usefulness. As for the installation itself, which will cost, when completed, a half-million dollars, I question whether in the whole field of cyanidation, we shall find a more ingenious and original achievement, or one more conspicuously uniting metallurgical efficiency with perfection of mechanical detail.

*Mining and Scientific Press, December 15, 1906.

VACUUM SLIME FILTERS

(July 13 1907)

The Editor:

Sir—In view of the articles that have appeared from time to time in the columns of the MINING AND SCIENTIFIC PRESS as to the merits of the Moore and Butters systems of vacuum-filters, having had some experience with both, I would like to add a few remarks to the general exchange of ideas through your columns.

I do not think there is much difference in either system as far as cost of operation and maintenance is concerned, but I do think there is considerable difference when it comes to the maintenance of the best conditions for the highest efficiency in the vacuum type of filter. It makes no difference which of the two filters mentioned is used for making the cake, as, with a given vacuum and pulp conditions, either of them make a cake equally fast. Therefore, it comes to a question of displacing the pregnant solution contained in the finished cake by barren solution or water.

The best conditions for perfect displacement are obtained by having a homogeneous mixture of the pulp in the filter-box throughout the time of making the cake and by maintaining uniform conditions until the end of displacement, when the deposited slime is ready to be discharged. By uniform conditions, I mean a constant vacuum, constant hydrostatic pressure, and as little exposure to the air as possible. Under these conditions I find that the displacement is almost perfect, no appreciable decrease in strength of cyanide being noticeable until nearing the finish of displacement, when the decrease is rapid down to the strength of the barren solution with which the washing is being done, which shows a displacement of the pregnant solution contained in the cake, with very little dilution.

Dilution, when displacing with water or barren solution, is to be avoided, as it means a larger amount of solution to be passed through the zinc-boxes, and, in the event of water being used, it also means a larger quantity of waste solution.

The methods of operation of both the Moore and Butters filters have been given at some length in your columns at various times, but for the purpose of comparision I shall briefly outline them again.

In both systems the storage, or equalizing-vat, is placed to one side of the filter-box, pulp being drawn from it to keep the leaves of the filter submerged during the formation of the cake.

In the Moore system the whole nest of leaves (which are attached to a frame) is removed from the filter-box by a traveling crane and delivered into a displacing-vat, which is immediately alongside the filter-vat, the vacuum being maintained during the moving of the loaded nest of leaves to the displacing-vats. The unit is composed of three filtering-vats, two for displacing, and one in the centre for loading.

When the crane has removed the filters from the loading-vat and deposited them in the displacing-vat, it is disconnected and taken to the other nest of filters, which are in the other displacing-vat, and when the slime-cake on the filters is discharged they are brought over into the loading-vat, and the making of the cake commences without loss of time.

In the Butters system the filters are stationary in the filter-vat, and, when the cake is made the pulp is pumped back into the storage-vat, the vat is washed out with solution and is then filled with water and displacement commences. When finished, the cake is discharged, the surplus water is run back to storage, and the discharged slime is sluiced to waste. The vat is again sluiced out to remove any particles of the discharged cake adhering to the sides, and again pumped full of slime for the next cycle of operations.

A comparison of the time consumed for a complete cycle is as follows :

MOORE SYSTEM AT LIBERTY BELL MILL, TELLURIDE, COLO., JANUARY, 1906.

	Hr.	Min
Making cake		45
Transferring from loading to displacing-vat ...		5
Displacing or washing cake..................	1	0
Sampling		5
Discharging................................		15
Transferring from displacing to loading-vat. ..		5
Total time per cycle..........................	2	15

BUTTERS SYSTEM AT COMBINATION MILL, GOLDFIELD, NEV., MARCH, 1907

Filling filter-box from storage		25
Making cake		30
Pumping pulp from filter-vat back to storage		22
Filling filter-vat with waste solution or water		20
Displacing or washing cake	1	0
Sampling		5
Discharging		15
Running surplus wash solution to storage		6
Sluicing out discharged slime residue		8
Total time per cycle	3	11

In comparing the above tables for time consumed by the various steps in each system, it will be noticed that the transfer from the loading-vat to the displacing-vat in the Moore system occupies five minutes. During this short period of time the vacuum is fully maintained; therefore, there is no change in this particular part of the operation, the exposure to the atmosphere is short and the change from the hydrostatic pressure of the pulp on the finished cake to that of the water on the same in the displacing-vat is rapid.

In the Butters system the transfer of the remaining pulp back to storage after the cake is made occupies 22 minutes; the pumping of the water for displacing (after all the pulp is out) occupies 20 minutes, making a total of 42 minutes from the finishing of the cake before displacement can be commenced. This long period necessitates the vacuum being dropped from 21 to 5 inches.

The difference in total time consumed per cycle, other than in making the cake and displacing it, is seen to be 30 minutes with the Moore system, and 1 hr. 41 min. with the Butters, the saving in favor of the Moore system being 1 hr. 11 min. per cycle, but the saving of time is not the only consideration.

The length of time occupied between finishing the making of the cake and the commencement of displacing it in the Butters system necessitates dropping the vacuum down to five inches, or barely sufficient to hold on the cake, otherwise the cake would be so badly air-cracked that displacement would be impossible.

(A ¾-in. cake under 18 in. vacuum will crack beyond redemption in seven minutes.)

When the vacuum is lowered, after finishing the cake, much of the moist exterior portion of the cake drops off when the hydrostatic pressure is removed, it being a difficult matter to adjust the vacuum at the best point for any particular cake; for if too high, the cake will be badly cracked; and if too low, the cake has a tendency to slough off.

In the 12 min. consumed in pumping the pulp back to storage and the water into the filter-box for displacement, the extreme top of the cake is exposed for the full period of time to the atmosphere and the low vacuum that is maintained, the pulp recedes slowly down the cake during the pumping out of the pulp, and the cake is slowly submerged during the entrance of the water. Therefore, there is a period of time (about 26 min.) in which the extreme upper half of the cake is exposed to the air longer than the lower half, and with sufficient vacuum to hold the cake on the filter-leaf there is also sufficient to cause displacement as the water slowly rises with the result that the pregnant solution in the lower half of the cake is thoroughly displaced long before that in the upper half and a large amount of dilution occurs, as the large quantity of weak barren solution necessary for displacement goes to prove, at the Combination plant it being four and one-half times the amount of moisture contained in the cake before displacement.

It will be seen that in the time schedule of the cycle of operations the Butters system will have to be considerably changed to allow of the best working conditions. And I consider it preferable to use one vat for loading and a separate vat for displacing, as then the small loss of time necessary for washing out the filter-box when changing from pulp to water and the reverse are avoided, and it also avoids the constant small losses that are bound to occur by this method through mixing of small amounts of pulp and pregnant solution with waste solution when they go through the same system of pumps and piping, and the richer the solution the greater the loss.

The wide difference in the capacity of the two plants makes it difficult to give a just comparison of the cost of operations, the Liberty Bell plant at Telluride having a capacity of 450 tons per day and the Combination Mines plant at Goldfield 193 tons per day as at present equipped.

The labor employed at the Liberty Bell at the time of which I write was two men on each shift, one of whom acted as shift-boss and had general supervision of settlers, agitators, and Moore filters, making all necessary titrations throughout these departments. At the Combination plant, one man on each shift attends to the filter-plant besides making other titrations in the leaching-plant, so that with the small unit as installed, that is, two filter-boxes of 28 and 30 leaves, respectively, one man per shift could possibly handle the full capacity of the plant or 193 tons per day.

The power consumption at the Liberty Bell is two 10-h.p. motors at the transferring cranes and one 40-h.p. motor to operate the vacuum and other auxiliary pumps connected with the filtering system, the same motor also operating a Goulds triplex pump that returns all solution used in milling the ore to the top of the mill, a lift of about 80 ft., requiring about 7 h.p. constantly, the total power consumption being 53 h.p. to charge against the filter equipment. The power consumption at the Combination plant when operating two Butters pumps, which the recent new addition makes necessary (the plant being two units), is 20 horse-power.

The Combination plant then, with its two units, one of 28 leaves and the other of 30 leaves, can handle 193 tons of dry slime per day when working at a maximum capacity, with labor of one man per shift and a consumption of 20 h.p. The Liberty Bell plant, with its four sets of leaves consisting of 67 each, will handle 450 tons per day when working at maximum capacity, with the labor of two men per shift and the consumption of 53 h.p. Therefore, as far as direct comparison of the actual operating costs of the two plants goes, there is not much difference. But two important items that effect the capacity of the Moore plant at the Liberty Bell, and that so far have not been taken into consideration, are the sticky nature of the ore handled and a difference of 3,000 ft. greater altitude.

In taking up the question of maintenance it will be noticed that the total number of leaves in the Liberty Bell plant is 268 against 58 leaves in the Combination plant, hence, allowing for an equal life in each case, the renewal cost for this item on the Moore plant will be almost five times as great as on the Butters plant at the Combination, but the rule previously mentioned still holding good, that a given pulp and vacuum condition will produce the same cake in either system, the Moore plant at the Liberty Bell if working on Combination ore would have a capacity of 805 tons per day against 193 tons per day of the present Combination plant. In the moving parts of the crane in the Moore system, the main item for expenses for repairs is the renewal of the lifting cables, which require renewing about once a year if properly adjusted when put on. The repairs that are necessitated by the cranes are offset by the repairs to the centrifugal pumps in the Butters system. With the Moore system, where heavy loads of slime are moved from one vat to another, there is always the possibility of accidents, for instance, a cable breaking and causing a bad wreck, running up the repair bills rapidly, but with properly designed apparatus the probability of such an occurrence is very remote.

It is supposed that the moving of the filters greatly lessens their life by straining the canvas; such is not the case, as one cannot feel the least jar when the crane starts to lift, the vertical velocity being only seven to eight feet per minute while hoisting the loaded baskets and the same when lowering into the displacing-vat, and there is apparently no more difference in the cake or filters after moving than if they had not been removed from the loading-vat. The suspended load on each filter-leaf is the same in either case, whether the filter with its load of slime is removed from the vat full of pulp, or the pulp removed from around the filter, the only difference being in the length of time the filter is required to sustain the load of slime, unaided by hydrostatic pressure of either pulp or water, which maximum length of time is seen to be five minutes in the Moore system to 44 in the Butters.

In a comparison of costs of installation, a 58-leaf plant such as is used in the Combination mill will be taken and the main items of necessary material for each one tabulated.

BUTTERS SYSTEM.	MOORE SYSTEM.
2 1 by 10 ft. filter-boxes.	3 11 by 10 ft. filter-boxes.
1 14 by 12 ft. pulp storage-vat.	4 14 by 12 ft. pulp storage-vat.
1 14 by 12 ft. water " "	1 12 by 10 in. vacuum pump.
1 12 by 10 in. vacuum-pump.	1 10-h.p. motor.
2 4-in. Butters centrifugal pumps.	1 20-ton crane equipped with
1 20-h.p. motor.	1 10-h.p. motor.
58 filter-leaves.	1 4-in. centrifugal pump.
	58 filter-leaves.

It will be seen that the Moore system requires three filter-boxes against only two needed in the Butters, only one storage-vat being needed, and that for pulp in the first instance, as against two in the second; and a 20-h.p. motor against two 10-h.p. The main difference in the cost of installation is in the crane, which will add 35% to the total cost of the plant. The above equipments comprise an equal filtering-area in each case. The Moore system will handle 22 charges in 12 hours, and the Butters system 10 charges in 12 hr. 44 minutes.

To summarize: The operating costs of either system are about equal, as are also the maintenance and repairs.

The installation cost for the Moore system is 35% greater than the Butters system for an equal filtering-area.

The capacity is 50% greater in the Moore system than in the Butters system for an equal filtering-area.

The efficiency of the Moore system for the recovery of pregnant solution with the least amount of loss and dilution, is higher than the Butters system for reasons previously given.

There is no doubt but that given ideal topographical conditions, the Butters system can be a great deal better arranged than here at the Combination plant.

Where sufficient elevation is available to allow of ample pulp and water storage above filter-boxes, and also the same ample storage below (to again allow for the storage of pulp and water), a great saving of time is obtainable by allowing the pulp and water to flow in and out of the filter-box rapidly. This would necessarily greatly increase the cost of installation. Under such conditions the amount of pulp and water to be moved back up hill after each cycle of operations would be performed over a long period of time and not obstruct the filtering operation, instead of a short period during which filtering operations must cease, as at the Combination plant.

But even under these conditions, which have materially lessened the time consumed and also improved the conditions for displacement by removing the pulp from around the filters rapidly and getting them quickly submerged in the displacing medium, we have not eliminated the obvious opportunities for loss, that is, by alternately running pregnant solution and water through the same pumps and pipes and into the same receptacles, as a large percentage of the water that in this way becomes mixed with the small amounts of pulp and pregnant solution has to be allowed to run to waste with the discharged cakes of slime. Therefore, the less opportunity there is for any such mixing, whether accidental or unavoidable, the better. Hence my reason for saying that a loading-vat in which the cake is made should be used for that purpose alone. And the same reasoning holds good as regards pipe-lines and pumps for moving pulp containing pregnant solution.

Since commencing this contribution, Mr. E. H. Nutter, of the Liberty Bell mill at Telluride, has given us a further article on the Moore filters in use there, slightly changing the figures as shown in the table I have given for time consumed per cycle, which change no doubt comes from over a year's further experience with the plant. I am hoping that he will furnish us with some valuable figures from recent operations there, and that Mr. Mark R. Lamb will do the same with reference to the more recent installations of the Butters system in Mexico, for the benefit of the readers of the MINING AND SCIENTIFIC PRESS.

A. G. KIRBY.

Goldfield, Nevada, June 15.

CONVEYING TAILING THROUGH PIPE

(July 20, 1907)

The Editor:

Sir—In reply to your request for further data concerning the pipe-line for conveying tailing, mentioned in your article appearing in the issue of June 29, I beg to say that this line is 5,410 ft. long, of 8-in. cast-iron bell and spigot pipe, ⅝-in. thick, and joints calked with hemp rope loosely driven into place after having been tarred. It is laid for its first 800 ft. on a grade of 3½%, after which it has a uniform grade throughout of 2¼%. The pipe-line was most carefully asphalted on the specifications for asphalting pipe contained in the proceedings of the American Institute of Mining Engineers two years ago, and was made by the American Cast Iron Pipe & Foundry Co. at Anniston, Alabama, no special price being paid for said asphalting, which is part of their regular practice, apparently.

The line of pipe passes through a crowded city and has in its construction many curves, the majority of which are on a 11-ft. radius, the curves being made by short sections of curved pipe in 3-ft. and 6-ft. lengths, ordered for that purpose. At the head of this line are two de-watering cones of 20 ft. diam. with 45° sides; these remove about one-half of the water from an eight to one pulp before introduction into the pump.

The pipe was put into service at the middle of March, 1906, and has been in continuous use up to the present date, having carried approximately 100,000 tons of dry pulp during that time. Careful measurements of the interior diameter of the pipe, made at the expiration of 13 months of service, show that it has undergone no appreciable wear. The total cost of maintenance and up-keep during the 13 months has been 3.57 *pesos*, spent for paint on one of the viaduct towers. There has been absolutely no other item of expense since its installation.

The pulp is that resulting from crushing through a mill of 80 stamps with a steel wire screen of 26 mesh with 28 wire followed by tube-milling of a portion of the coarse product such that after removal of 38% of the pulp as slime, 52% of the balance nominally passes a 120-mesh screen. At times, however, when the tube-mill has been out of service for replacement of linings, we have run two weeks at a time on straight 26-mesh battery-pulp. On two or three occasions, by reason of neglect on the part of the Mexican in charge of the de-watering cones, the thickness of the pulp has been considerably increased over the normal four to one proportion and on such occasions a slight deposit of coarse sand in the pipe has occurred, which always gives notice by a whistling noise at a blow-hole intentionally placed in the pipe about 1,000 ft. from its head. This notice always occurs in plenty of time for the error in the thickness of the pulp to be corrected; on two occasions this occurred on night-shift and the line filled approximately half-full with coarse sand, but an hour's time was sufficient to clear the line (by the introduction of more water at the head) with no expense beyond the momentary delay. As soon as riffles of sand are formed so as to cause an obstruction in the flow, the pipe develops hydrostatic head above the point of obstruction sufficient to force the sand through. Delays from this source have not caused more than four hours' loss of time in the entire 13 months, and even then they were due to carelessness at the cones, as mentioned above. Normally, month after month, there was not the slightest tendency of the pulp to deposit any sand whatever in the line and when, due to electric shut-downs or any other stoppage at the mill, pulp ceases to flow into the head of the pipe, the line keeps itself entirely clear without the introduction of water.

At the present capacity of 250 tons per day the pulp runs only $1\frac{3}{4}$ in. deep on the bottom of the 8-in. pipe, running at such speed that the distance between the ends of the line is traversed in about 12 minutes. The additional 80 stamps to be started in September will double the amount of pulp, but the pipe can carry 1,000 tons as easily as the present 250.

The wear is so slight as to be invisible up to date, but in any case, the pulp running only in the lower part of the pipe would allow of the turning of the entire line five times before it was entirely worn out, and we figure that if the line were to become worn out every two years it would only mean two cents per ton carried. The present appearance is that it will last 50 years.

C. W. VAN LAW.

Guanajuato, July 2.

CYANIDATION IN THE TRANSVAAL

(July 20, 1907)

The Editor:

Sir—I must take exception to your Johannesburg correspondent's remarks in your issue of June 8. Possibly he does not understand the matter he criticizes, otherwise it is altogether difficult to understand why he should so strongly depreciate the work of the Denny brothers because the local people are apparently unable to estimate accurately the assay-value of their ore.

That the "new metallurgy" consisted merely of a method of assaying is news to me. What the Messrs. Denny announced was that:

(a) The old method of treating coarse sand by a long percolation was wrong in theory and in practice and that in spite of opposition the mines would abandon it in favor of fine grinding, for which purpose they introduced tube-mills.

(b) By the use of tube-mills they expected to obtain not only a higher extraction from the resulting finer grinding, but also a higher output.

(c) A continuous method of slime treatment by having solution occur during the flow or travel of the pulp instead of in special agitators after collection and settlement was cheaper, both in first cost of plant and in working, than the prevalent decantation method.

In order further to increase their extraction they employed filter-presses instead of decantation.

All their points the Messrs. Denny appear to have proved to the hilt. Indeed, the use of tube-mills, starting from the one imported by the Dennys, has now spread all over the Rand; and I note no fault whatever is found with the extraction obtained by the "new metallurgy," and that in the same speech Mr. Albu apparently states his satisfaction with the work of the filter-presses. His one cause of dissatisfaction seems to be his difficulty in obtaining reliable preliminary assays, but this surely should not be insuperable, and is an entirely subordinate matter to the obtaining of the highest proportion of gold at the minimum cost

I write this because the remarks of your correspondent are calculated to convey a wrong impression, and not because I am entirely in accord with the line taken by the Dennys. They were cautious in their preliminary work on tube-mills; their verdict was amply justified by results. They did a good deal of preliminary work with filter-presses; there, too, their decision seems justified by results. Whether they displayed the same caution in building two new plants depending on solution of the gold by what we may term 'mortar-box contact' would be a much more debatable subject except for the fact that it appears they provided additional plant for the purpose, but the use of which was not found in practice to be necessary, and I have been advised—not from any source connected with the Messrs. Denny—that Mr. Albu has strongly objected to the expenditure of the money on this precautionary measure, though the vats have been converted to other use.

It looks, therefore, not merely as if Messrs. Denny have proved their case to the hilt, but that indeed this same system has been adopted in practically all the latest American plants and in a number of those in other territories. To my mind a serious matter against the new system is the difficulty of copper-plate amalgamation. In Australia, pans are successfully used for the purpose, and I have no doubt that other suitable methods will be devised in plenty—possibly by the use of some special alloy for amalgamating plates—now that the need of special effort in this direction is known; but I cannot believe that a method of extracting all the gold in the slime at practically no expense for plant (as far as the solution of gold is concerned) will be lightly abandoned by the industry (I do not refer here to South African practice only) merely because of a faulty or ineffective method of obtaining preliminary assay-values. The saving in cost of equipment alone is a factor compelling most serious attention.

In conclusion, I am not prepared to admit that the Messrs. Denny have yet made made out their case in favor of all-sliming. If the words "finer grinding" are substituted for "all-sliming" there would be general agreement with them, but personally I am of opinion that it is still the most satisfactory and economical method in general practice—with exceptions—to percolate in vats, without expense for power or handling, all the sand suitable for percolation in vats, the remaining pulp receiving the suitable treatment necessary. Apart from the less cost of thus creating the fine sand, we have as yet no reliable data to prove that the extraction obtained from the slimed sand, as sand, apart from slimed concentrate or slime, is so much higher as to justify the extra cost of sliming and of handling the slime. There may be ores in which total sliming is preferable, but where the gold is carried in the concentrate or in contact with some softer or more fragile material it is surely cheaper to treat the fine particles of sand without subjecting them to the partially unrecovered expense of further grinding.

ANOTHER CORRESPONDENT.

London, June 26

CYANIDATION IN THE TRANSVAAL

(July 27 1907)

The Editor:

Sir—We have read in your issue of June 8 some statements made by your Johannesburg correspondent in connection with our metallurgical innovations on the Rand which have considerably astonished us, and we trust that as you have circulated through your journal statements which are calculated to do us some harm that you will grant us the use of your columns to rehabilitate ourselves in the eyes of your readers.

Your correspondent begins by stating that "several of the mines under our control spent thousands of pounds to install the process of circulating cyanide solutions advocated by the Denny brothers," and follows this with a quotation from the address of the Chairman of the New Goch Co. (on which mine we installed one of our plants), and an observation of his own that the Chairman's statement referred to "is rather disconcerting to the 'new metallurgy.'"

Before proceeding to deal with the Chairman's remarks on the subject of our process, we would like to point out that not a single pound was spent by us in the installation of the process for circulating cyanide solutions. We purposely built our plants so that in the event of our wishing to try the effect of circulating solutions they were as readily adapted for that as for our ordinary treatment without circulating cyanide solution, which we had installed at the Van Ryn mine, and which, we may add, is the system to which it is proposed now to revert at the New Goch. It would have been fair to us, more especially as we are not on the Witwatersrand, to reply to your correspondent's criticism, if he had stated that after all, the only portion of the plant that the Chairman of the New Goch found fault with was the system of circulating cyanide solution, and that the main principles of our plant, embodying tube-milling, automatic slime-treatment, and filter-pressing, remained as highly successful elements of the 'new metallurgy,' and constituting, of course as the technical man knows, practically the whole of the 'new metallurgy.'

Coming now to the distinct feature of our system, which has been pronounced by the Chairman of the New Goch to have given rise to difficulties, we would observe that the test of the efficiency of a plant is high extraction coupled with low working cost. Neither of these claims are questioned by the Chairman, who clearly states that it is the intention to abandon the circulating cyanide solution method, merely because of the unreliable nature of the screen-assays, which had given rise to erroneous estimates of the gold to be expected at the monthly clean-up.

Adverting first to the efficiency of the plant, we may say, without any fear of contradiction, that the residue from the plant will not average more than from four to six grains, when in the former plant the average was from 18 to 24 gr. This result cannot be said to be unsatisfactory.

The assertion of the Chairman that the screen-assays have proved unreliable is one we can neither contradict nor confirm. In our complete paper, read before the South African Association of Engineers, in June, 1906, we admitted the difficulties inherent in the practice of screen-sampling, when the sample itself is subjected to the solvent effects of weak cyanide solution; but we also pointed out very emphatically that the ordinary system of screen-sampling, and especially the usual method employed on the Rand, were open to even graver charges of contamination, which charges were not rebutted in the ensuing discussion.

As a matter of fact, we might mention that the screen-assay at the New Goch during several years of our experience was as unreliable as it could possibly be, and we attributed it then, and now, to the fact that the ore of the mine contains unusually coarse particles of free gold, and therefore we do not believe that the abandonment of the circulating solution will overcome the difficulty, while we are certain that the working costs will be higher and the extractions lower. We make the last statement quite advisedly, because we ran the plant at the Meyer & Charlton mine (a plant the exact counterpart of the New Goch plant), first without cyanide solution, and carefully noted the results; and subsequently with cyanide solution in circulation, thus getting actual working figures for comparison. The important differences are given in our paper previously referred to.

What is the more inexplicable part of this change-over on the New Goch mine, is that with an exactly similar plant at the

Meyer & Charlton, extraordinary success has been attained; which is attested by the fact that the manager of the mine, when showing the members of a scientific society over the plant at the end of last year, stated that the extraction up to that date, nearly a year after the plant had started running, was 95%. We may mention that with the former plant, our extractions averaged between 80 and 85% only, so that the new plant, according to the official statement of the manager, is responsible for 10% higher extraction, at an increased working cost of one shilling per ton; the net result being from 4 to 5s. per ton profit in favor of the new plant. Our estimate on which the plant was built was a net improvement of 4s. per ton in the profits.

It is interesting to observe that the Chairman of the New Goch, who is also Chairman of the Meyer & Charlton, stated in his annual address in 1907, when dealing with the operations for the past year, that "the larger profits resulting (from the year's work) have more than justified the expenditure occurred in the additions to, and improvement of, the plant." He further stated that "the company had produced more gold and earned a larger profit than in any previous year," the actual figures being with the old plant for 1905 a gross profit of £99,784, and with the new for the year 1906 a gross profit of £136,967, or an increase of over 36 per cent.

The question now arises why, with the Meyer & Charlton plant, such magnificent results have been secured, and why at the New Goch they have been so disappointing as to give rise to the criticisms and damning statements of your correspondent.

We are in position to say that during the time we were in control of the plants, we made complete clean-ups of the new Meyer & Charlton plant, first, to satisfy ourselves that the gold called for, as between the screen assays and the residue-assays, was in the plant, and subsequently, at the instance of the Chairman of the company, on more than one occasion, to satisfy his doubt on the matter. The clean-up was in each case a complete vindication of our methods, as we had in each case a plus extraction. In the face of these results, and of the actual record of the Meyer & Charlton with its new plant, for 1906, we think that you must agree that your correspondent's strictures, and his concluding observation that "it is fortunate for the rest of the Rand that conservative counsels prevailed, and that the mines of other groups did not take up the metalliferous schemes so persistently promulgated by the Messrs Denny," are unfair. It is rather unfortunate for your correspondent that this final statement should, in any case, so inaccurately represent the actual situation.

We were solely responsible for the introduction of tube-mills, and the leading, if not the only, advocates for fine-grinding, and secondary amalgamation. Our principles have been accepted and a plant to carry them out has been installed on every up-to-date mine on the fields. We are again solely responsible for the introduction of filter-presses to the Rand. Slowly, but surely, these appliances are being installed, and no less than five big mines are at the moment putting them in. We are solely responsible for the introduction to the Rand of a new form of mortar-box, in which the water-feed is delivered through the back of the box, at about the level of the die. We learn from the consulting metallurgist to the leading group on the field, that the same method has been adopted by them with the greatest success.

In conclusion, we would say that when the whole cause of dissatisfaction with our plant is boiled down it merely refers to the fact that on a particular mine, disappointments in the gold yield have been experienced, which on that particular mine is no new thing. We assert without any fear of contradiction, that the only other mine on which we installed a plant, where the dissappointments indicated have never existed, a complete clean-up of the plant has always shown rather more gold than the screen-assay called for. The reversion to the so-called decantation system, about which the Chairman of the New Goch and your correspondent have both made so much, does not invalidate one single principle of our scheme, nor cause the slightest alteration to our plants, other than the substitution of water for weak cyanide solution, and a dam for receiving the slime and solution from our specially designed slime-plants, which latter, as a matter of fact, we provided before the plant was started.

The whole of the details of this scheme we worked out, and successfully practised at the Van Ryn mine, with an equipment of 160 stamps. We are sorry that an explanation of the true position in connection with our plant has necessitated so much space, but we believe it to be due to our friends and ourselves to state the position fully and clearly. DENNY BROS.,

London, June 27. Per G. A. DENNY.

TUBE-MILLS AT GUANAJUATO

(August 17, 1907)

The Editor:

Sir—As to our experience with our tubes: Apparently it has been markedly different from that of a good many people, and we really do not understand the reason why, as we have had no difficulty whatsoever, aside from routine replacements in the eight or nine months we have been running them. These mills, as you know, are the Abbé mills, 4 ft. 6 in. by 20 ft., and are handling about 80 tons of pulp per day.

A screen test made on pulp going to the tube-mill shows:

	Mesh.	Per Cent
Remaining on	40	11.2
" "	50	11.2
" "	60	8.9
" "	80	16.6
" "	100	16.3
" "	120	26.1
Passing	120	9.7

Moisture 60 per cent.

The screen test on discharge-pulp shows:

	Mesh.	Per Cent
Remaining on	40	0.5
" "	50	1.7
" "	60	2.9
" "	80	6.0
" "	100	16.2
" "	120	21.8
Passing	120	51.2

Mechanically the mills have given us no trouble since their installation, with the exception that the first pinion shipped with the mills was shrouded and the distance between the shrouds was not sufficient to allow for any end motion whatsoever of the mill as a whole, and the gear ultimately rode the shroud of the pinion, smashing both the gear and the pinion, but doing no further damage; since replacing with a pinion wider between the shrouds, no difficulty whatever has been had.

To limit the end motion upon this type of mill, two guide-rollers with vertical axis bear upon the sides of the supporting tire next the gear end, and a good deal of trouble has been had by earlier users of the mill, we understand, on account of the tendency of the mill to run endwise, being so severe that these guide-rollers were quickly worn out; the Abbé Company suggested to us, before our first mill was installed, that it was possible to cant the friction-rollers upon which the tire moves (and which constitute the support of the mill) in such a way as to cause the main tube to go in either direction longitudinally, and that by delicate adjustment of this canting, it should be possible to confine the mill entirely to one position without its bearing unduly hard on either of the guide rollers. We experimented with this and within the first day or two after the mill was started we found it possible by this adjustment to cause the main tube to 'float' entirely free on its supporting rollers without touching either of the guide-rollers at all, and the mill ran for three months at one time without touching either guide-roller, and normally it hardly touches them once a day.

The silex lining of the mills has given good satisfaction, the first lining running eight months before it had to be replaced, though at the end of about six months, three or four defective blocks were replaced, with a two days' shut-down.

A great deal depends apparently upon the truth of the supporting tires and the homogeneity of the material composing it, because if in these tires the slightest irregularity occurs, either as an original defect or as the result of operation, bumping would rapidly ensue, which would cause the destruction of the mill and perhaps its foundations. So true is this that we found on starting the first mill, that a film of $^1/_{16}$ of an inch of sand, which had accumulated on certain oily spots on the tire, caused violent bumping, until the mill was stopped and the ring tire carefully cleaned off. We keep these tires at all times carefully cleaned and a little oil is put on the surface of the tire once or twice a day. Since the tires were first cleaned, there has not been the faintest bump or vibration evident about either mill, the mills running as quietly and smoothly as possible. The mills are kept filled slightly above centre with pebbles, the pebble wear being approximately $\frac{3}{4}$ lb. per ton of ore treated; silex linings wear eight months; power to start the mill, about 60 h.p., which immediately drops to 43 h.p. as soon as at running speed.

Trusting that the above data may be of interest to some of your readers.

C. W. VAN LAW.

Guanajuato, Mexico, July 14.

CONVEYING TAILING IN LAUNDERS

(September 14, 1907)

The Editor:

Sir—I have read with much interest the description appearing in your issue of June 29, on conveying tailing in cast-iron pipe at Guanajuato, also the further statements on the same subject by Mr. Van Law in the issue of July 20. Having in the last two years made about 7,000 experiments on the carrying capacity of launders, the subject is of course interesting to me. In my experiments on tailings which were clean pure quartz and quartzite (probably much more angular and sharp than the material at Guanajuato) I find that on a grade of 1.5% in a rectangular launder on material of the fineness mentioned in your article one pound of water will only carry off 0.022 lb. quartz, when the launder has the most economical width for the quantity of material carried.

My experiments also show that with the most economical width of launder, the same material with water in the proportion of 7½ to 1 would require a grade of nearly 5%. Only on absolute slime did I obtain better results in a V-shaped launder than in a rectangular one. My experiments were checked, some twice and some three times.

Why my results are so much lower than those mentioned, I would like to ascertain. I know Mr. Van Law personally and know the figures he has given can be nothing but correct, and the difference must be wholly in the nature of the two materials handled (this refers to the issue of June 29).

With your permission, and begging the pardon of my brother engineers, I can positively state that the "wetted perimeter" has nothing whatsoever to do with the carrying capacity (as far as solids go) of a launder. Personally, I was formerly under the impression that it had, but my experiments have conclusively proved that we have all been wrong in this assumption. For example, a 1-in. launder using 25 lb. water per min. will carry exactly one-half of the tailing of a 2-in. launder using 50 lb. water, and one-tenth of a 10-in. launder using 250 lb. water per min., all, of course, on the same grade; that is, the carrying capacity in sand, etc., is proportional to the width. This I can prove by figures from about 500 actual experiments. In the next statement I shall probably get myself into trouble, but Mr. Van Law's experience will bear me out.

Under the right conditions, material can be transported in a pipe. On a grade, we could not possibly transport it in an open launder, but the initial grade should be greater than the grade at the end (the sooner to develop pressure) which in an open launder would be bad practice. But, as a rule, we can only utilize a pipe on very fine material, and when handling comparatively large quantities. To make myself clear, if we attempt to transport in a pipe ½-in. material at the rate of 10 lb. per min. on a certain grade, with the least quantity of water, the theoretical size of pipe required would be so small that in a short time the pipe would clog, but to transport 100-mesh material through the same pipe would be easy. I know full well that pipes used in mills in place of launders, often give much trouble and it is not my purpose to recommend pipe in place of launders, except for carrying off great quantities of comparatively fine tailing. There is more to Mr. Van Law's expression, which I will quote, than most of us (probably even himself included) would think. "As soon as riffles of sand are formed so as to cause an obstruction in the flow, the pipe develops hydrostatic head above the point of obstruction sufficient to force the sand through." That is, if the pipe is too large to give the velocity of water that will keep the material in suspension (or carry it on the grade used) then nature itself, by the filling in of sand, reduces the size of the pipe down to the point where the velocity will be high enough to carry the material off, depositing it ahead again where velocity is slower, until this point in turn reaches the carrying stage, and so on. This action also shows to some extent in a launder, but the effect is not the same as in a pipe, because the launder can keep on filling up and water still flows on top, without increasing its velocity at that point. The carrying of tailing in a pipe and in a launder are two entirely different problems, and do not at all follow the same laws. In the pipe we depend mostly on the head developed, but in the launder on the rolling and sliding action down an inclined plane, except in the case of exceedingly fine material, where the hydraulic condition or wetted perimeter approximates the right proportion of a launder. I hope in another year to finish my investigation in regard to the problem of launder capacity and to present it in much better form than these ramblings.

G. A. OVERSTROM.

Salt Lake City, August 3'.

CONVEYING TAILING IN LAUNDERS

(October 12, 1907)

The Editor:

Sir—The courteous query for more light in your issue of September 14, and signed by Mr. G. A. Overstrom, deserves full data in answer.

The conditions are not quite as Mr. Overstrom takes them. The ore crushed is an exceeding hard tough quartz, with very small admixture of calcite, and no other gangue; it is therefore very sharp and granular. The coarser portion of the pulp which results from battery crushing through 26 mesh 28 wire steel screens is tube-milled, and the resulting screen-test after tube-milling is as follows:

PULP WHEN USING TUBE-MILLS

	Mesh.	Per Cent.
Remaining on	40	3.1
" "	50	4.4
" "	60	2.8
" "	80	4.1
" "	100	4.3
" "	120	10.0
" "	150	3.1
Passing	150	68.2

The line has for several weeks at a time, however, been run without the tube-mills running, therefore taking the straight battery product, screen test of which is as follows:

STRAIGHT BATTERY PULP

	Mesh.	Per Cent.
Remaining on	40	5.1
" "	50	5.2
" "	60	5.2
" "	80	5.7
" "	100	5.6
" "	120	11.8
" "	150	1.7
Passing	150	59.7

The normal proportion of water that these pulps contain in the pipe-line is about five to one by weight, and while there is no doubt that the hydrostatic head formed behind riffles of sand in the pipe when such occur greatly assists in preventing the complete clogging of the pipe, this action has only occurred a few times in the eighteen months that the pipe has been running, and normally has no influence whatsoever. Under normal conditions, embracing all but a few hours since the line was installed, there are no riffles whatsoever in the line and the upper three-quarters of the pipe is entirely clear, the pulp flowing in a smooth rapid stream without ripples or disturbance in the lower quarter of the pipe, precisely as in an open launder. I may say that since adding the second 80 stamps, the line which now carries pulp from 160 stamps is running slightly less than half full.

Mr. Overstrom's difficulty in reconciling the performance with many detailed experiments is shared by many. During the time of installation of the line the number of expert opinions passed upon whether it would run or not and based upon experiences with launders, was very large and with one solitary exception, flatly unfavorable. We had, however, demonstrated experimentally that pulp much thicker than that now being handled and somewhat coarser, would run freely in an identical pipe-line 300 ft. long on a 2½% grade—running so freely, in fact, that we felt justified in going to a 2¼% grade to better fit the contour of the country.

The limitations of the line seem clearly marked. With the tube-mills running, no tendency to clog has ever been discovered. With the tube-mills out and on straight battery pulp if the water is reduced to less than five to one, riffles of sand commence to form at a point about 500 ft. from the upper end of the line, giving notice there by a whistling noise, due to the air disturbance in the pipe, which issues through a blow-hole placed at that point. The restoration of the proper dilution at the thickening cone at once corrects the fault.

The economic importance of being able to transfer tailing over great distance at such small grades and at practically no cost (as evidenced in our expense of P3.57 as the total charges against the carrying of 100,000 tons of ore a mile in distance) is considerable, hence the intrusion upon your space with the foregoing detail.

The wear upon the line to date is not measurable after eighteen months' service.

C. W. Van Law.

Guanajuato, September 28.

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