

History, Development, and  
Modern Applications of the  
Cyanide Process

by Victor M. Cone

*June, 1906*

Submitted to the School of Engineering of the  
University of Kansas in partial fulfillment of the  
requirements for a course in Mining Engineering

# Mining Thesis

Victor M. Cone

MINING THESIS

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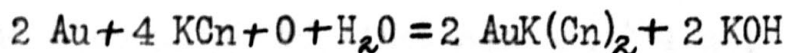
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## HISTORY OF THE CYANIDE PROCESS.

The fact of gold being soluble in cyanide of potassium solution has been known for over a hundred years at least. As early as 1805, Hagen stated that gold is dissolved not only by free chlorine and aqua regia, but by a potassium-cyanide solution. Within the next few years mention was made of this and analogous facts in many texts on chemistry, also in scientific journals. It was not, however, until 1840 that any commercial application of the discovery was attempted, but at this time Dr. Wright, of Birmingham, England, used gold-cyanide solution for electro-plating. Wright's invention was patented by J. R. & H. Elkington, who specify in their application for patent that the solution of gold in prussiate of potash shall be boiling, thus recognizing the fact that a hot solution of cyanide will act on gold more readily than a cold solution. The first record in scientific literature of experiments in which metallic gold was dissolved in a solution of cyanide of potassium, consists in Prince Pierre Bagration's paper in the Bulletin de l'Academie Imperiale des Sciences de St. Petersburg, 1843. Bagration, who alludes to Elkingtons process, preserved cyanide of potassium in a dish, gilded on the inside. After eight days had elapsed, he noticed that the whole gold surface had been attacked. He experimented then with finely divided gold under the influence of the galvanic current; the latter he soon recognized to be of no value in the dissolving process. He precipitated the gold out of solution, by means of the electric current, on a cathode of copper. Bagration also found by continued experiment, that higher temperatures aided in the dissolving process, and taught

the precipitation of the metal from the warm solution by means of silver or copper plates without the use of the electric current. He furthermore found that the warm solution has the disadvantage of attacking the silver and copper during the precipitation process. Bagnard extended his experiments to the use of ferrocyanide solution which acts like cyanide but much more feebly. He noticed the influence of air on the reaction.

In 1844, L. Elsnor published the results of his experiments on the reactions of "reguline metals" in an aqueous solution of cyanide. He found that gold and silver were dissolved in potassium cyanide without the decomposition of water. In his own words, "The dissolution of the metal is, however, the consequence of the action of the oxygen, which, absorbed from the air, decomposes part of the cyanide." His idea was represented by the following equation:



A few years after, Faraday made use of the solubility of gold in cyanide solution for reducing the thickness of films, thus providing the basis for the modern extraction of gold. It took many years, however, before the enumerated facts were many use of for the extraction of gold from ores.

#### Patents.

Probably the first attempt to interest mining men in the use of cyanide for the extraction of gold from ores was made in 1867, when a patent was issued to Julio H. Rae, for "an improved method of treating auriferous and argentiferous ores" with a current of electricity in connection with a suitable solution- such, for instance



as cyanide of potassium. Rae's process is an agitation process; he proposed to treat the ore with a cyanide solution and then precipitate the metal by means of an electric current. His was a strong solution of cyanide, and since he had probably not carried it beyond the laboratory stage, the process sank into oblivion. Patent was also issued to Thomas C. Clark, in 1880, who roasted his ore to a red heat, and placed it in that condition in a cold bath composed of a solution of sodium chloride, prussiate of potash, and caustic soda. This action was intended as preliminary to amalgamation. In 1881, John F. Sanders, was granted a patent for a composition to dissolve the impure coatings from gold, leaving the metal free and exposed, and permitting it to be amalgamated. It is evident, therefore, that these processes bear no relation to the modern cyanide process.

No attempts at using cyanide of potassium for the extraction of metal from ores had been patented since Rae until 1885, when Jerome W. Simpson obtained a patent for a process for extracting gold, silver, and copper from ores. Simpson reduced his ore to a powder and agitated it with a solution of one pound of cyanide of potassium, one ounce of carbonate of ammonia, one-half ounce of chloride of sodium, and sixteen quarts of water. This solution was intended particularly to dissolve the sulphurets of gold, silver, and copper. The metals dissolved in the salt solution were precipitated by means of zinc, suspended therein in the form of pieces or plates. Simpson was aware that cyanide, in connection with an electric current, had been used for dissolving metal, and also that zinc had been employed as a precipitant. What he claimed as new was; (1) the process of sep-

arating gold and silver from their ores, which consists in subjecting the ore to the action of the solution of cyanide of potassium and carbonate of ammonia, and subsequently precipitating the dissolved metal by means of zinc. (2)The process of separating metals from their ores, to wit, "subjecting the ore to the action of a solution of cyanide of potassium, carbonate of ammonia, and chloride of sodium, and subsequently precipitating the dissolved metals." Simpson seems to have failed to discover the most important property of dilute cyanide solutions, namely, that of dissolving, without the addition of other chemicals, the noble (in preference to the base) metals. It has since been found that the addition of sodium chloride is of no benefit for the extraction of silver. The addition of ammonium carbonate is not beneficial to the extraction of either gold or silver, except under certain conditions, when it may be substituted advantageously by an alkali or an alkaline earth; in the presence of base metals it is a disadvantage. There is no record that Simpson's process was commercially tried previous to the issue of the MacArthur and Forrest patents in 1889. These patents were applied for in 1887, but owing the ambiguous construction of the application papers it was not until two years later that a sufficiently clear statement of their demands was made to warrant the issuance of patent rights. Louis Janin, F. M. Endlich, and N. H. Muhlenberger each filed caveats in 1885 and 1886 but neither of them were pushed on to the taking out of patent papers. Endlich became doubtful of the practicability of the cyanide process. Dixon experimented with Australian ores in 1887 but secured no patents. He refers to the work of the patent-

ees mentioned above, in a paper published in the records of the Royal Society of New South Wales, and states that he found "the aurocyanides of the alkaline metals of sufficient stability to render their use possible for the extraction of gold," but feared "the high price of cyanide, its instability when exposed to air, and its extremely poisonous properties." As far as known, Dixon's experiments did not lead to the commercial application of cyanide as a gold and silver solvent.

All of the history thus far mentioned concerns the use of cyanide as a dissolving agent from the early laboratory experiments up to its metallurgical application for ore extraction; this latter, however, did not gain any practical importance until John S. MacArthur and W. Forrest, of Glasgow, Scotland, took out their patents for the use of cyanide as a gold and silver solvent from ores, and gave thereby the cyanide process a start all over the world. Their patents mark an epoch in gold metallurgy, for the millions of dollars of bullion produced by it during the comparatively few years since its institution would, in a great measure, have been irrecoverably lost. They drew out a possible list of solvents having a greater affinity for gold than for the sulphides, which included the cyanides, and which they found to solve the problem of economically treating low grade complex ores. It is doubtless true that their process was not evolved from their own efforts and observations, but it is known that they experimented along this line for several years before the English patents controlling their process was applied for. Great credit is due MacArthur & Forrest for their strong belief

in the future of the process as was evidenced by their securing patents all over the mining world. In their United States patent, granted December 1889, the invention is described "as having principally for its object the obtaining of gold from ores, but it is also applicable for obtaining silver from ores containing it whether with or without gold, and it comprises an improved process, which, while applicable to auriferous and argentiferous ores generally, is advantageously and economically effective with refractory ores, or ores from which gold and silver have not been satisfactorily or profitably obtained by the amalgamating or other processes hitherto employed; such as ores containing sulphides, arsenides, tellurides, and compounds of base metals generally, and ores from which the gold has not been easily or completely separable on account of its existing in the ores in a state of extremely fine division." The patentees describe their invention as consisting in subjecting the ores to the action of a solution containing a small quantity of cyanide, without any other chemically active agent. In dealing with ores containing twenty ounces or less of gold or silver per ton, or gold and silver, they find it most advantageous to use a quantity of cyanide, the cyanogen of which is equal in weight of from one to four parts for every thousand parts of the ore dissolved in a quantity of water of about half the weight of the ore; they generally use a solution containing two parts of cyanogen for every thousand parts of the ore. In the case of richer ores, while increasing the quantity of cyanide to suit the greater quantity of gold or silver, they also increase the quantity of water so as to keep the solution dilute; in

other words, the cyanide solution should contain from two to eight parts, by weight, of cyanogen to one thousand parts of water, and the quantity of the solution used should be terminated by the richness of the ore. The patentees state, "By treatment of the ores with the dilute and simple solution of a cyanide, the gold or silver is, or the gold and silver are, obtained in solution, while any base metals in the ores are left undissolved except to a practically inappreciable extent; whereas, when the cyanide is used in combination with an electric current, or in conjunction with other chemically active agent, such as carbonate of ammonium, or chloride of sodium, or phosphoric acid, or when the solution contains too much cyanide, not only is there a greater expenditure of chemicals in the first instance, but the base metals are dissolved to a large extent along with the gold or silver, and their subsequent separation involves extra expense, which is saved by their process." Later on MacArthur & Forrest obtained patents covering the use of zinc in a fine state of division for the purpose of precipitating gold and silver from cyanide, chloride, bromide, thiosulphide, sulphate or other similar solutions; they further protected the use of an alkali or alkaline earth for neutralizing ores preparatory to subjecting the same to the action of cyanogen or of a cyanide. The MacArthur-Forrest patent claims consisted, therefore, in three points: (1) The application of diluted solutions of cyanide ( not exceeding eight parts of cyanide to a thousand parts of water); (2) the use of zinc in a fine state of division; (3) the preparatory treatment of the ore, which has become partially oxidized by exposure to the weather, with an alkali

or alkaline earth, for the purpose of neutralizing the salts of iron or other objectionable ingredients formed by partial oxidization.

Without entering into a judicial discussion of patent-claims and patent-rights of the cyanide process, it is necessary to date the cyanide process as a commercial success from 1890 when it was introduced on the Witwatersrand gold fields, in South Africa, as "the MacArthur-Forrest process". Its success as a metallurgical experiment may be dated from the tests made on a large scale with ore from the New Zealand Crown Mine in June and July, 1888. As would be naturally supposed, the practicability of the cyanide process once established, others endeavored to introduce improvements in its application, which they protected by letters patent. Of the great number of so called improvements many were merely slight alterations in details, which proved to be of no practical importance, but some few were steps in the right direction. In 1892, B. C. Malloy, of Johannesburg, patented one of the first "improvements", which promised for a time to be of great practical importance. His process consisted in the abolition of zinc as a precipitant of gold, and in the revivication of the cyanide of potassium in the solution, by passing the gold bearing solution over a surface of mercury in which sodium was continually deposited. As the sodium deposited and accumulated in the mercury within a shallow tray like compartment, it diffused to the mercury on the outside, over which a constant stream of gold bearing solution was kept flowing. The gold-bearing molecules dissolved as they came in contact with the sodium amalgam surface, were decomposed and gold amalgam formed. The weak point of this process

lies in the difficulty of getting a sufficiently large electrode surface, for the solution to come in contact with, without unduly increasing the amount of mercury handled. It is possible to partially overcome this difficulty by rapid agitation of the solution while over the sodium-amalgam surface. The efficiency of sodium-amalgam as a precipitant is small, but when employed in the dilute state, as in the Molloy process, it compares favorably with the electrical precipitation processes of today. Sodium-amalgam is said to act on KCN in solution, forming complexes, but in dilute working solutions this action is inappreciable.

Moldenhauer, in 1893, patented a process of precipitation by means of aluminum instead of zinc. It was proved that, for good work, free acid or free alkali was necessary in the solution, and as free acid is impracticable in the cyanide process, the method was abandoned. Furthermore, it was found after the process had been at work for some time, that the aluminum deposited on the shavings, which retarded precipitation of the gold.

Johannes Pflieger patented a method in the Transvaal, in 1895, which was used on a working scale for a short time. It consisted in employing a short circuited simple cell, having zinc anodes in alkali or cyanide, and iron cathodes in the gold bearing solution, the two solutions being separated by a porous membrane. A great advantage offered by this process was that by placing electrical measuring instruments in the circuit a definite and accurate idea of the results could be obtained. As remarkably good and uniform results were obtained by this method, it is not easy to understand why the

process was finally abandoned.

C. M. Pielsticker secured New Zealand patent rights for a method in which he reverts to the use of the electric current in connection with the cyanide solution. He proposed to continuously circulate the solvent, to continuously precipitate the dissolved precious metals by means of electrolysis, and continuously regenerate thereby the reagent.

W. D. Johnson was granted a patent, in 1894, in the United States for precipitating gold and silver from their alkali solutions by means of carbon, preferably in the form of charcoal. The pulverized carbon is placed in such a manner as to form a filter for the solutions carrying the metals, thus depositing them on the carbon. By heating, the carbon is driven off and the gold and silver remains. This system did not prove to be as efficient as zinc, owing to the great number of filters required. A peculiar fact here is that, theoretically, carbon cannot precipitate gold, because it is electro-negative to the latter in a KCN solution. That it is a practical precipitant has been proven in several plants, but no satisfactory explanation of the action has ever been advanced.

The Pelatan-Clerici process was patented in 1897, and consists of a combination of the leaching and precipitation processes in the same vat. The bottom of the vat is lined with metallic copper which has a coating of mercury, thus forming the cathode of an electric couple. The electric current passes down a central shaft in the middle of the vat, and from there the current passes to the iron arms which revolve around the central shaft to which they are attached.



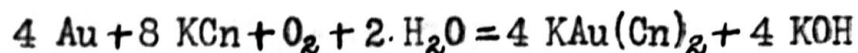
These arms form the anode while the auro-cyanide solution forms the electrolyte. A few large plants have used this process with a greater or less degree of success and it is used at the present time, but in general it may be said of all electrical precipitation methods, that they are not nearly as successful as some form of zinc precipitation.

The Siemens-Halske process for bullion precipitation by electrolysis on lead sheets was patented in 1894. The precipitation plant consists in boxes through which the gold solution passes; these contain the anodes, which are iron plates, and the cathodes, which are sheets of lead, stretched between iron wires fixed in a light wooden frame, which is suspended between the iron plates. The advantages claimed for this process are that the electrical precipitation is independent of the amount of cyanide or caustic soda contained in the solution, therefore in the treatment of the tailings very dilute solutions can be used: and, as with all electrical precipitation methods, it is free from aluminum and iron hydroxides which are a constant source of annoyance in the zinc precipitation. The Siemens-Halske process is probably the most important and most used electrical method that has been devised. There are several plants employing the process, but as previously stated, the electrical precipitation methods are, in their present state of development, not as efficient as the zinc process.

## CHEMISTRY OF THE CYANIDE PROCESS.

No other metallurgical process, having attained a prominence equal to that of the commercial use of cyanide in ore extraction, is as little understood from a chemical standpoint. In fact the fundamental principles governing the process are in dispute, owing to the complexness of the ores treated. It is doubtless true that no single equation can be formed to represent the exact reaction which occurs even when pure cyanide solution acts on pure gold. The effect of oxygen, and the instability of the cyanide itself, give rise to complications. It is these complications that have furnished a question for argument and research. The results of such investigations and observations will be discussed under the head of chemistry.

According to Elsner's equation:

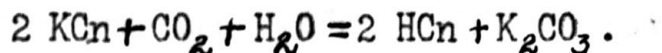


130 parts by weight of KCN in the presence of 8 parts of oxygen suffice for the solution of 196.8 parts of gold. Other equations have been advanced to explain the action and probably no one point in chemistry has been the cause of as much contention as whether or not oxygen is essential to the solution of gold by cyanide. Experiments have been sited to prove both sides of the question, but it is the consensus of opinion among chemists and metallurgists, that, in the absence of impurities, a solution of KCN in pure water will not dissolve gold. It is also agreed that sufficient oxygen for the reaction is secured from the air and, in general, no artificial means of oxidation is resorted to unless there is some reducing agent present in the ore or the water, which absorbs the oxygen. The supposition

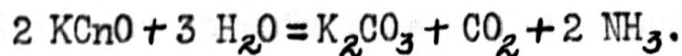
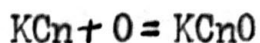
that any oxygen coming into contact with the cyanide would be taken up at once, to form cyanates, has been disproven, for experiments show that cyanide and free oxygen may exist side by side without rapid union between them taking place.

#### Decomposition of Cyanide.

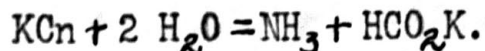
Hydrocyanic acid is one of the weakest acids, and is expelled from its salts by all mineral acids and many organic acids. Carbonic acid decomposes KCN in presence of water thus:



The smell of hydrocyanic acid, noticeable whenever KCN or its solutions are exposed to the air, is accounted for by this reaction. In the presence of air KCN takes up oxygen, and is converted first to cyanide and then to carbonate:



In dilute solutions, KCN appears to be partly changed to HCN and KOH, since the passage of a stream of neutral gas causes an evolution of hydrocyanic acid, while the solution becomes alkaline. The equation is:  $\text{KCN} + \text{H}_2\text{O} = \text{HCN} + \text{KOH}$ . If the solution is boiled with acids or alkalies, hydrolysis of the cyanide occurs rapidly, ammonia and formic acid being formed, thus:

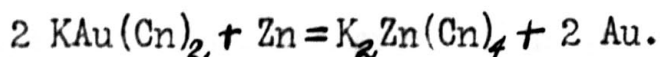


This equation does not represent the entire effect. It is obvious, from the above facts, that (1) the solution should be kept from the air as far as possible, by having closely fitting lids to the storage vats, the zinc boxes, etc., and by transferring the solution

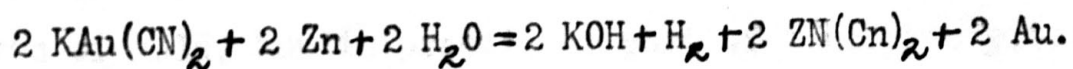
from one to the other by means other than open launders, as pipes; and (2) the solution must be kept free from acids, which can be effected by the addition of a little alkali.

#### Reaction in the Zinc Boxes.

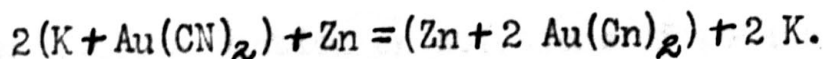
For a considerable time after the invention of the zinc precipitation method it was believed that the gold and silver was displaced according to the following simple equation:



Some considerations have been adduced which throw doubt on this view. It is well known that a considerable excess of free cyanide must be present to enable the precipitation to take place quickly and efficiently, and if the solution contains less than 0.05 % KCN the solution must be in contact with the zinc for an hour. Filiform zinc will extract gold from a solution in which pure sheet zinc fails to have any action, and this fact is probably accounted for by the presence of lead in the filiform zinc. Christy suggests the cause of this to be the setting free of hydrogen by the first action of the aurocyanide of potassium on the zinc, thus:

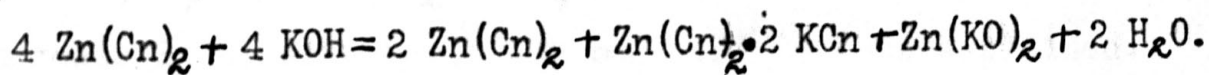


This equation practically demands the replacement of the K ions in the aurocyanide of zinc, thus:

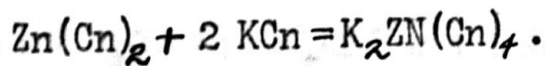


followed by a rearrangement of the molecules of zinc-gold cyanide and the replacement of the gold by another atom of zinc. This mechanism of change seems the more probable when it is remembered that the K ions in KCN itself are displaced by zinc in the presence of

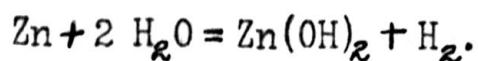
water. Assuming that hydrogen is set free as above, it would form a layer on the sheet zinc and thereby polarize and protect it from further action. On zinc shavings it is assumed that the edges serve to collect the bubbles of hydrogen and thus protect the majority of the surface from polarization. At any rate, it is true that hydrogen is given off, and other theories are given to account for its presence. It is not clear how the presence of free KCN would prevent polarization by hydrogen, and the prevention of the action may be due to the formation of a layer of insoluble  $Zn(Cn)_2$  on the surface of the zinc, which would be immediately dissolved if potassium cyanide were present. In the absence of KCN, the reaction proceeds:



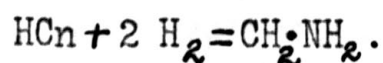
This reaction is incomplete, but unless the potassium in the solution in the solution were large, it is clear that some insoluble zinc cyanide would remain. The reaction is completed by the solution of  $Zn(Cn)_2$  in free potassium cyanide:



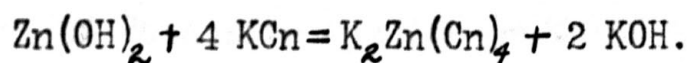
According to Christy's formula, 1 part of zinc should precipitate about 3 parts of gold, but in practice 1 part of zinc precipitates only from 0.06 to 0.2 part of gold, the zinc being wasted by direct dissolution in potassium cyanide and by the reaction described below. Pure zinc has a slower action on solutions of potassium cyanide than zinc containing lead in small quantities, and the zinc-gold couple, formed by the black deposit of gold and the unaltered zinc, is still more vigorous. This couple probably develops sufficient electromotive force to decompose water thus:



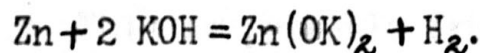
The positive element, zinc, is thus oxidized, and subsequently dissolved by the cyanide solution. As a matter of fact, bubbles of hydrogen are evolved after a time. The hydrogen probably carries off some hydrocyanic acid, forming methylamine, thus:



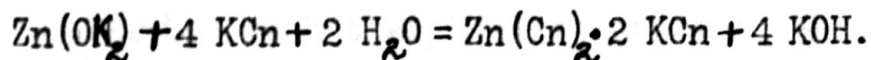
This may account for the ammonical odor occurring above the zinc boxes, but it may also be due to hydrolytic decomposition. The hydrate of zinc, formed as shown above, dissolves at once in excess of the cyanide:



and the increase in alkalinity of the solution is thus explained. Caustic potash, however, acts directly on the zinc to some extent forming potassium zincate, thus:



but the zincate is at once dissolved by KCN as follows:



In this way the solution tends to become more and more alkaline, but it is neutralized by carbonic acid absorbed from the air.

By the reactions in the zinc boxes the solution is weakened and a large amount of zinc is consumed. It has been shown that the presence of iron in the zinc boxes checks the rate of precipitation of the gold, and increases the waste of zinc and the tendency for the formation of  $\text{Zn}(\text{Cn})_2$  on the zinc. Hence iron screws etc. should not be used in the construction of the zinc boxes. The double cyanide of zinc and potassium is not available for the solution of precious

metals, and, generally speaking, no gold will be dissolved as long as there is an excess of zinc present. However, the presence of large quantities of zinc and potassium is not prejudicial to the solvent action of the simple cyanides. Hence it is, that solutions may be as good after months of use as when they were freshly prepared. The effect of temperature on precipitation is marked. As much gold will be precipitated by zinc at 80 degrees C. in two hours, as in twenty-four hours at 20 degrees. In the case of higher temperatures, however, there is an enormous waste of cyanide as is evidenced by the strong odor of urea.

#### Action of Potassium Cyanide on Metallic Salts and Minerals occurring in ores.

The ordinary gangue of most ores (viz., silica, and silicates of the alkalies and alkaline earths) exercise no direct influence on the cyanide solution. This is probably true of the carbonates of the alkaline earths also. The decomposing effect of sulphides of the heavy metals varies with the physical state of the sulphides. From experiments with various grades of pyritic ores and concentrates, it has been found that the consumption of cyanide varies from 3 to 50 pounds per ton of ore and increases with the time, the volume of the solution, and the degree of concentration, the latter being especially marked. An increase of concentration from 0.5 % to 0.9 % caused the consumption to be doubled, and a further increase to 1.5 %, doubled the consumption again. Sulphides of copper and zinc are rapidly dissolved by potassium cyanide, but it acts very little on iron. Messrs. MacArthur and Forrest found that dilute cyanide solu-

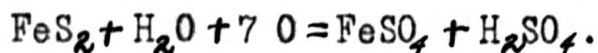
tions exercise a "selective action" in dissolving gold and silver in whatever form they may be present, in preference to sulphides or other salts of the base metals. This rule is not strictly correct for there is apparently a very great difference between the effects of chemicals on natural occurring compounds, and one that is deposited in the laboratory. For instance, cyanide of potassium solution has a strong tendency to dissolve precipitated sulphide of zinc, but its action on the natural sulphide of zinc, blende, is almost nil. The same holds for compounds of iron, and again, there is a greater consumption of cyanide in concentrates containing marcasite than when the pyrite is entirely of the ordinary yellow cubical description. The compounds of copper, physically soft, also tend to increase the consumption. Increase of consumption with increase of strength of solution has also been proven. The general rule that oxides, hydrates, carbonates, sulphates, and sulphides of those metals which are electro-positive to gold in cyanide solutions are dissolved more rapidly than the last named metal, whether it is present in the metallic form or contained in its commonly occurring salts, does not hold in all cases. MacArthur has shown that the case may be quite different when the naturally occurring minerals are concerned. For instance, not only is precipitated sulphide of copper rapidly dissolved, but also a sooty form of the same substance occasionally met with as a mineral occurring in ores. On the other hand, fused copper matte is scarcely acted on at all, and in a great many cases the same may be said of the hard dense sulphides of copper usually found in nature. Sulphides of zinc exhibit the same differences of behavior:



"black-jack" can be treated with good results, little zinc being dissolved. Again, oxide of copper, if freshly precipitated, is strongly acted upon by the cyanide, but if it is heated to dull redness in a muffle it becomes insoluble, and a large excess of this material added to a gold ore makes no difference in the percentage of extraction, while the consumption of cyanide is not increased by its presence.

#### Action of Potassium Cyanide on Oxidized Pyrite.

When the pyrites occur in tailings which have been subjected to the action of the weather for some time before treatment, compounds are formed which are more prejudicial to the solution than the sulphides. Sulphide of iron,  $FeS_2$ , is oxidized by air and water, ferrous sulphate and sulphuric acid being formed, thus:



The protosulphate suffers further oxidation, and normal ferric sulphate ( $Fe_2O_3 \cdot 3 SO_4$ ) is produced, which eventually loses acid and becomes a soluble basic sulphate,  $Fe_2O_3 \cdot 2 SO_3$ . Other basic salts of complex and unknown compositions appear to be formed also.

W. A. Caldecott has investigated the products of decomposition of iron pyrite contained in slimes accumulated in dams or pits in the Transvaal with the following results. His view is that the main stages in the oxidation of pyrite or marcasite are as follows:

- (1)  $FeS_2$ , - - - - - Iron pyrite .
- (2)  $FeS + S$ , - - - - - Ferrous sulphide and sulphur.
- (3)  $FeSO_4 + H_2SO_4$ , - - - - - Ferrous sulphate and sulphuric acid.

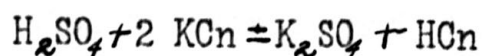
(4)  $\text{Fe}_2(\text{SO}_4)_3$ , - - - - - Ferric sulphate.

(5)  $2 \text{Fe}_2\text{O}_3 \cdot \text{SO}_3$ , - - - - - Insoluble basic ferric sulfate

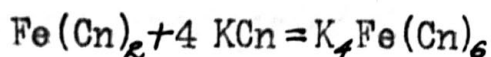
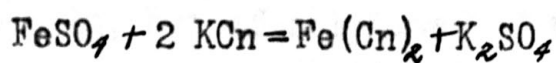
(6)  $\text{Fe}_2\text{O}_3$ , - - - - - Ferric oxide.

In the presence of much oxidized copper and pyrite, the following reactions take place:

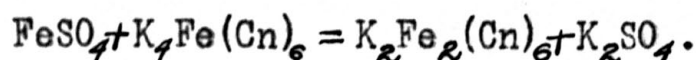
(1) The free sulphuric acid liberates hydrocyanic acid.



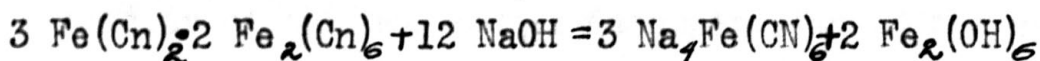
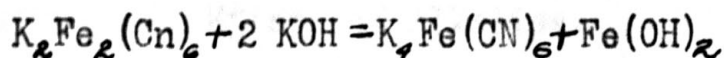
(2) Ferrous sulphate reacts on the cyanide, forming ferrous cyanide, which dissolves in the excess of potassium cyanide, so that it does not appear in the free state.



The potassic ferrocyanide, if sufficient acid be present, reacts with fresh ferrous sulphate forming a bluish-white precipitate.

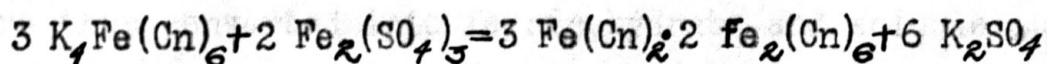


Both these precipitates are decomposed by potassium hydroxide or sodium hydroxide and therefore cannot be formed in their presence.



(3) Ferric sulphates are decomposed by potassium cyanide, hydrocyanic acid being evolved and ferric hydrate precipitated.

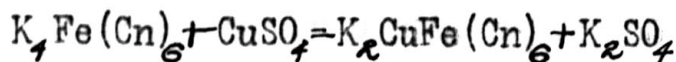
(4) A mixture of ferrous and ferric sulphates produce Prussian blue by reacting with potassium cyanide, ferrocyanide of potassium being formed at first as above: the equation being,-



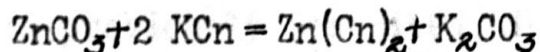
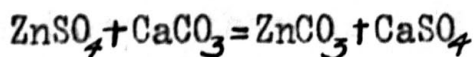
Here again the waste of cyanide is prevented by keeping the solution

alkaline.

(5) Sulphate of copper,  $\text{CuSO}_4$ , acts differently from  $\text{FeSO}_4$ , cuprous cyanide,  $\text{Cu}_2(\text{Cn})_2$ , being formed, soluble in excess of KCN to  $\text{K}_2\text{Cu}_2(\text{Cn})_4$ , a compound very prone to decomposition. Copper sulphate also gives a precipitate with potassium ferrocyanide, thus:

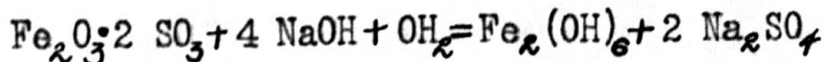


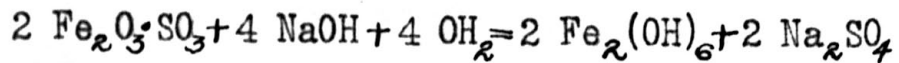
(6) Ferrous hydrate, however formed, does not act on potassium cyanide. Its only action is mechanical, as it collects in a gelatinous mass on the filters and checks the flow of the liquid. Hydrates and carbonates of copper and zinc are quickly dissolved in preference to the precious metals, but if sulphates of these metals are formed in an ore containing limestone or clay, double decomposition occurs with the production of sulphate of lime or aluminum, and oxides or carbonates of the heavy metals, which are dissolved in the cyanide, thus:



Effect of Alkalies.

Since acidity of the ore causes decomposition of the cyanide, and obvious method of reducing the loss is to add alkali in some form. Before doing this the free sulphuric acid and soluble salts may be removed by leaching with water, and then a solution of caustic soda or lime is run in on the ore, and after standing some time is drained off and followed by the cyanide solution. The insoluble basic salts are thus converted into ferric hydrate and soluble sulphates-

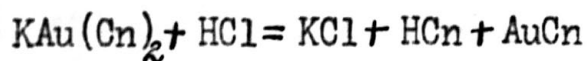




Although the action of the iron salts is slower, an excess of lime is less detrimental than sodium hydroxide to the cyanide solution, and does not attack the zinc. It is found that, even after treatment of the oxidized pyrite by alkalis, the loss of cyanide is much greater than in the case of free milling ores. It is now usually regarded as more advantageous to add lime as a dry powder to the ore before it is charged into the vats, instead of alkaline solution. The necessary amount is added to each truck load of tailings, if treated in that manner, or to the vat directly. The lime is also, in some cases, added to the ore as it is treated to the stamp mill, thereby intimately mixing the two before treatment with cyanide is reached. The exact amount of lime necessary for the ore is determined experimentally.

#### Reprecipitation of Gold and Silver in the Leaching Vats.

If the solution is acid there is said to be some danger of a precipitation of gold previously described, insoluble aurous cyanide being thrown down, according to the equation:



This, however, need not be feared as long as there is an excess of potassium cyanide.

## LOSS OF CYANIDE.

From the chemistry given in the preceeding pages, it will be seen that the secondary reactions occurring between cyanogen and certain soluble salts (such, for instance, as the soluble salts of iron resulting from pyritic decomposition) are accompanied by a considerable loss of cyanide. This is proven by the enormous consumption of the chemical in comparison with what is required for the actual dissolution of gold and silver. Other sources of cyanide consumption are:-

(1) Practical experience has shown that the presence of certain soluble salts of copper, bismuth, antimony, arsenic, manganese, sulphur, and tellurium have a deliterous action on cyanide solutions, in many cases rendering the process wholly inapplicable.

(2) A very common source of cyanide consumption is sulphuric acid and certain soluble and insoluble iron salts found in oxidized tailings, as the products of the decomposition of pyrite.

(3) A distinct loss is sometimes traceable to the presence, in old tailings, of organic matter, such as grass-roots, fragments of decayed brushwood, etc.

(4) Some writers have laid considerable stress on the losses occurring from the mere deliquescence of potassium cyanide, and the consequent volatilization of hydrocyanic acid gas. There appears to be a loss due to the absorption of carbonic acid gas from the air (where the surface of the solution is in contact with the air) displacing cyanogen and liberating hydrocyanic acid gas. A further loss has been traced to the oxidation of cyanide of potassium in the pre-

sence of air, and its partial conversion into a cyanate, and then into a carbonate. These changes may be all grouped under the one head of losses due to contact with the atmosphere. Such losses unquestionably do occur, as shown by laboratory experiments; but, as is sometimes the case with small tests, these results are not borne out in practice. It is the consensus of opinion among cyanide men that the losses of cyanide caused by air and carbonic acid are, in reality, very insignificant. From the chemical equations, one would be led to suppose that dilute solutions of potassium cyanide are easily decomposed and oxidized into the cyanate, but, on the contrary, agitation of the solution by compressed air for hours at a time has no appreciable effect.

(5) Losses of cyanide by absorption of wooden vats may be appreciable at first, but after they have been in use for some time they probably become insignificant. This loss, of course, is avoided in iron tanks, but the iron must be coated with a cyanide proof paint to prevent serious action on the iron.

(6) Theoretically there is a weakening of the solution of cyanide in the zinc precipitation boxes by decomposition resulting from the action of the zinc-gold couple; that is, by the electrolytic interaction of the metallic deposit on the zinc, and the unaltered zinc. Sufficient electromotive force is thus established to decompose water, with the liberation of hydrogen in the form of bubbles. Hydrocyanic acid gas is supposed to be carried off, mechanically, by this hydrogen; accordingly, one might expect a more vigorous electrolytic action from the solutions rich in gold, with a correspond-

ingly increased evolution of hydrogen, but in reality no more cyanide will be consumed in a solution rich in gold, than in a solution poor in gold.

(7) Some loss is due to discharging the leached residues saturated with a weak cyanide solution, but filter presses are overcoming the loss.

(8) A considerable loss is incurred by running to waste of superfluous weak solutions.

(9) A variable loss occurs during the first stage of the process, traceable to the action of "cyanicides" or cyanide consumers in the ore, and to the presence of organic matter; also to the dissolution of the gold and silver, as accounted for by Elsner's reaction.

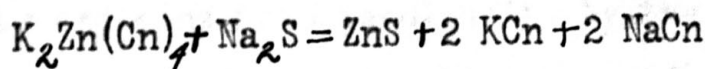
## PRECIPITATION DIFFICULTIES.

In the practical application of the cyanide process there are many difficulties to contend with, one of the greatest of these being the refusal of the cyanide solution to give up the gold it carries in solution. Occasionally the extraction runs as low as 50 % of the total assay value of the solution, and the perplexity of the situation is serious. Very often the trouble lies in the foulness of the solution, and means for correcting it must be devised. If there is a slimes plant in connection, a quarter of a ton of fresh water is introduced every day for every ton of ore crushed. This regenerates the solution very satisfactorily. Some solutions that can dissolve the gold from the ore in a cyanide vat very perfectly, but cannot get rid of it in the precipitation boxes, need a corrective rather than a regeneration. Probably the greatest troubles in the solutions are due to three principal causes;-(1) To the presence of organic matter. (2) To the presence of the double salts of zinc and cyanogen. These salts are formed by the solutions in passing through the zinc boxes and can be most troublesome. (3) To the presence in the solutions of the products of the decomposition of pyrite, i.e., ferrous salts and sulphates.

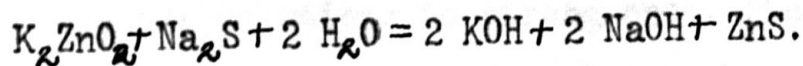
The correction of this latter trouble is the more difficult problem and as yet has not been satisfactorily worked out. The usual courses followed for counteracting the three evils are:- The addition of lead acetate to the solutions, which is rather efficient though quite costly. The addition of sulphuric acid to the sumps is practiced but it is a very drastic measure. Even though the cost may



be nominal as far as the acid is concerned, still the consumption of KCN (130 parts of KCN combine with 98 parts of  $H_2SO_4$ ) in this operation is very considerable, besides introducing a most dangerous element to life. Carbonate of soda is added to the solution in some plants on the Rand and very favorable results have been obtained. The most modern method for regenerating the solutions is one advanced by A. F. Crosse, of South Africa, and as yet results of a practical test have not been obtained. He proposes to use sodium sulfide for the precipitation of the zinc and the recovery of the cyanogen from the double cyanide of zinc and potassium,  $K_2Zn(Cn)_4$ . The cyanogen would be combined with an alkali according to the following equation:



The continuous use of an alkaline sulphide would be open to some objection, owing to the slight but gradual increase of caustic alkali, formed by the decomposition of potassium zincate, as:



In order to prevent this, the precipitated sulphide is collected in a filter press and used for the generation of hydrogen sulphide. Any silver sulphide precipitated with the zinc is also recovered in the process of regenerating the sulphide.

## RATE OF SOLUTION OF GOLD IN CYANIDE.

J. S. Maclaurin, of Auckland University, New Zealand, demonstrated that the rate of dissolution of pure gold in potassium cyanide solutions passes through a maximum when proceeding from dilute to concentrated solutions. The maximum is reached when the solution contains 0.25 % of KCN. The solubility of gold is very slight in solutions containing less than 0.005 %, but increases rapidly as the strength rises to 0.01 %, when the rate of dissolution is ten times as great as in the 0.005 % solution, and about half as great as that in the 0.25 %. The rate increases slowly as the strength rises to 0.25 %, and thereafter decreases much more slowly, until in a 15 % solution the rate of dissolution is about equal to that in 0.01 % solutions. Christy explains this peculiarity by the existence of two factors, the electromotive force of gold in cyanide solutions, which may be assumed to be proportional to the solubility of oxygen, and consequently decreases as the transition is made from pure water through dilute cyanide solutions to strong ones. The electromotive force of gold continually rises as the solution becomes stronger, but is never large enough to enable gold to dissolve in the absence of oxygen. It is true that in many mill operations, the free oxygen may be sometimes removed and aeration is necessary. In long continued treatment of concentrates it may be necessary to draw off the solution, partially dry the ore, and turn it over to expose it to the air before a fair percentage of the gold can be dissolved.

Investigations on the effect of heating cyanide solutions have given the important facts that the rate of solution of the gold is a

function of time and temperature, but the degree of concentration of the solution is less important. It was found that the quantity of gold dissolved in a certain solution at 60 degrees F. would be increased thirty-one times at a temperature of 160 degrees F., but at 100 degrees the dissolved gold would be reprecipitated. A very appreciable difference is noticed in large cyanide mills, between the extraction during the hot and cold seasons, and in some places steam is passed into the solutions to keep them at a constant temperature, about 80 degrees F., the entire year. Practice has proved special heating plants to be more expensive than is warranted by the increased extraction; but whenever possible, waste steam, as from an exhaust, is passed into the solutions to furnish the heat.

## THE CYANIDE PROCESS.

The fundamental principle upon which the cyanide process is based is the dissolving power of cyanogen when brought into contact with gold or silver, and the subsequent precipitation of these metals from such a solution by means of metallic zinc, carbon, or some application of the electric current. The practical adaptation of this principle has given rise to a multitude of variations, for the modern cyanide process is not a set formula having the same application to all cases and all conditions. In general, it is true that no one method is applicable to a number of kinds of ores, though it is true that several modifications of the one method may be applied with somewhat equal results on a single ore. Just as there is a wide variation in the physical and chemical state of ores, so is there a wide variation in the mode of treating them. Generally speaking, the cyanide process may be economically applied to the extraction of gold or silver from any ore, which indicates the possibilities of the process. Improved methods of milling have converted an immense number of auriferous and argentiferous deposits into economically workable properties, although they were too low grade to be handled prior to the application of the better methods of treatment. Of these milling advancements, cyanide treatment is probably the most radical and of the greatest importance. The great diversity of details in applying the action of cyanogen, indicate the prominence which it has attained and the immense amount of investigation that it has excited. Very satisfactory results are being obtained from the cyanide method in several large plants, but at no place is the process con-

sidered to be perfect.

There are several points to be considered preliminary to the cyanide treatment, and these points may directly affect the method of procedure in the process, and the construction of the plant. Among these factors to be considered are the physical and chemical states in which the gold and silver exists; the method of crushing, whether wet or dry, and the most suitable machines to be selected; whether advantageous to concentrate or amalgamate previous to cyanide treatment; and, whether roasting of the raw ore would be economical, in preference to fine grinding. Consequently, such considerations would be very incomplete without an estimate for each method investigated, as to cost of treatment on a working scale and probable profit. This question of profits is the most essential consideration, for the decision between two methods may depend on a few cents difference in the saving, and in treating gold and silver ores there is very often but a narrow margin between profit, in one case, and loss in the other. If such investigations are systematically made and tabulated, it is probable that the success of the selected process will be assured. This is laboratory work and, at the same time, an exhaustive set of tests should be made to determine the proper strength of solution to be used for that special ore.

The cyanide process comprises four distinct operations: (1) Preparation of the ore for treatment. (2) Solution of the gold in  $KCN$  (3) Precipitation of the dissolved gold by means of zinc; and (4) Conversion of the precipitated gold into bullion by fusion. In tracing out these operations, the different modes of practice will be given.

## CYANIDE PLANT.

The proper site for a cyanide plant will in each case depend upon a variety of local conditions, such as the source of water supply, the accessibility of material to be treated, the proximity of ground for discharging upon, and the means adopted for transporting the ore. In most instances an advantage will be obtained, obviously, by constructing the plant on sloping ground. If a flat site is chosen, the leaching tanks must be raised on piers of masonry, in order to get the necessary gradient for the flow of the solutions from one tier of vats to the next below. Usually where flat sites are necessary, the storage tanks are dispensed with altogether, and the solutions pumped directly to the leaching tanks. One instance in which level ground would be an advantage, is where there would be difficulty in getting the ore to the leaching tanks. The weight of the tanks, and ore and solution which they contain, is so great as to require solid ground for the foundation to rest on. In the greater number of cyanide plants in the United States, hillside locations have been obtained, but in South Africa the country is so level as to necessitate the erection of piers for the storage tanks and slime plant. Very often no buildings are used to shelter the tanks, only the precipitation boxes and clean-up apparatus requiring shelter. It is obvious, however, that any process employing much machinery would have to be protected from the elements. The construction of the several parts of cyanide plants will be touched upon as the process is described, where any unusual method of construction is encountered.

## CYANIDE PRACTICE.

In order to give the best practice in cyaniding, two cases will be taken, one to show the method as it is employed in South Africa and Australia, and another to show American tendencies in the use of cyanide for ore extraction. The Transvaal is generally conceded to be the home of cyaniding, and to the cyanide men of that district is due much of the credit for the advancement and improvement of the process. South African ores are well adapted to treatment by cyanide as are many American ores, but in Australia, many complex problems are met with and solved. The cases to be taken for illustration are partially hypothetical, but consist, in the main, of the most modern applications.

It so happens that at the present time a very radical change is anticipated in the cyanide treatment of ores on the Witwatersrand.

The cyanide experts are divided into two schools on the subject but as the results obtained seem to be in favor of the proposed changes, they will be given. An outline of the present practice is as follows:

- (1) Single breaking of the ore after passing over sorting tables, down to 2-inch to 3-inch cubes.
- (2) Milling with from 24 to 40 mesh screens.
- (3) Outside plate amalgamation.
- (4) Spitzlутten separation of from 5 % to 15 % of the heavier particles of the pulp, with subsequent cyanide treatment of the separated product, occupying from fourteen to forty days.
- (5) Subsequent separation of sands representing from 55 % to 70 % of the pulp/ with a treatment of from five to ten days; simul-

taneous separation of from 22 % to 35 % of the slimes, and treatment by decantation, the time of treatment varying according to the number of washes given.

The above schedule is sometimes varied by the use of vanners for concentrators, followed by chlorination of the concentrates, while in other cases concentration in any form is quite neglected. The use of vanners, however, has almost gone out, and therefore that aspect of practice may be neglected.

The alternative method proposed at present is,-

(1) Stage breaking down to 3/4 inch to 1 1/2 inch cubes.

(2) Milling with a coarse mesh of 15, followed by ordinary plate amalgamation.

(3) Classification of the coarse product in the mill pulp by spitzluten.

Regrinding of the coarse products (coarse products meaning everything in the pulp that would remain on a 60 mesh screen) in tube mills, or other suitable grinding machines. In the event of grinding pans being used, amalgamation in the pans would be resorted to.

(5) Passing the reground products over a further set of amalgamated plates.

(6) Automatic agitation of the whole pulp in cyanide solution, in a specially designed plant requiring no inter-pumping, making thereby practically an all sliming plant.

(7) Filter pressing of the pulp after agitation.

Before entering into the proposed changes, it may not be out of



place to state that, by the decantation method, profits have been made with an ore that assays only \$ 1.50 per ton. It is admitted that the percentage extraction by the decantation process is not as high as desirable, but since profits can be returned by this method, it is contended by those who are inclined to reject the proposed method, that a profit on a 60 or 80 % extraction is far better than a loss on a 99 % extraction. From several published annual returns of leading mines on the Witwatersrand, the following results of the old method of treatment are obtained:- In one mine a total of 151,674 tons of concentrates were treated, having an average value of \$ 12.25 and extraction averaging 79.83% was obtained. Again, from 1,031,095 tons treated, of an average of \$ 4.20, the average extraction was 75.21 %. In another case, from 494, 119 tons treated, of an average value of \$ 2.15, an average extraction of 83.79 % was obtained. From another statement, the average extraction of a number of mines was 89.34 %. In the mills, an average of about 60 % of the total gold values is obtained, but on the other three products, concentrates, sands and slimes, there are possibilities of improvement in extraction. Another contention of the new school is that, since two separate plants with two separate sets of men are required to treat the products of the old method, the cost is naturally higher than it would be if the product, after leaving the plates, could be treated in one plant. The lengthy time required for percolation of concentrates and sands requires enormous capacity of plant. The decantation treatment of slimes also calls for big tank capacity to allow for time required for settlement, intercirculation, agitation, wash-

ing, etc.

Three main advantages are claimed for the new process: (1) Lower cost of plant, meaning less initial outlay; (2) higher extraction; and (3) lower working costs. The proposed scheme of operations will be dealt with in the order given in the outline.

#### Stage Breaking.

It is agreed that the process of reducing 3-inch cubes to 1 1/2 inch cubes can be more effectively and more economically carried out in breakers than in a battery. The immediate effect of fine breaking is to increase mill duty, and the increase thus gained is obtained at a lower working cost, both in capital outlay and cost of operating, than could be gained by increasing the number of stamps.

#### Milling with a Coarse Mesh.

It was not many years ago when the object of every mill man was to carry out his milling in such a way as to produce the minimum amount of slimes. The finer the mesh used the greater the percent of slimes, and if a coarse mesh was used the extraction by amalgamation and cyaniding was appreciably reduced. A mesh of about 25 represented the compromise between these two features. Then followed a time when a much coarser mesh was advised in order to increase the stamp duty, on the argument that the subsequent reduction of the coarser particles could be more economically and effectively carried out on the principle of attrition. Plate amalgamation and cyanide practice need not be prejudicially effected, as the reground product could be again passed over amalgamated plates, and in the cyanide treatment the finely ground sands would give a better extraction

in a shorter treatment. This idea was made thoroughly practicable by the introduction of tube mills and grinding pans, and is now accepted as the best method.

#### Classification of Coarse Products.

An exhaustive series of analyses on South African ores, showed clearly that the causes of the low extraction in the treatment of the sands and slimes by the methods already described, lay entirely in the fact that the gold was inaccessible in the coarser products. The highest values in the residues in the treatment of concentrates, sands and slimes, were invariably found in the coarser products. Re-grinding these coarse products and agitating them in a cyanide solution enabled a high extraction to be obtained in a short treatment. Proper use of the spitzluten to separate out these heavier particles has done away with Frue vanners, for the spitzluten does the work equally as well as a vanner and at practically no cost.

There seems to be some growing doubt on the Witwatersrand as to whether the tube mill is the best form of machine to use in regrinding the coarse products. Considerable favor is being shown the grinding pan of the Wheeler type, which has been used in West Australia for some time with great satisfaction to the cyanide men. There seems to be a disposition to amalgamate the reground products, for a considerable extraction can be obtained in this way even from comparatively clean concentrates. Obviously the tube mill is not adapted to this work, for the operator cannot see on the inside of the mill and the flint has a tendency to flour the mercury. The grinding pan, on the contrary, is well known as an amalgamator. It is accessible

and easy to control. West Australian practice shows that one 16 foot tube mill has the same capacity as two and a half 5 foot grinding pans, or five pans will give the same output as two mills. The horse power required for a grinding pan is six, while for a 16 foot tube mill cannot be taken as less than thirty-five; moreover, the starting load of the inert mass in the mill is three times the running load. The cost of a pan erected is about \$ 800.00, equivalent to \$ 2,000.00 for 2 1/2 pans - while a 16 foot tube mill costs about \$ 3,500.00. To reline the tube mill requires seven days, while new shoes and dies may be fitted in a pan in a few hours. It is known that about 8 % of the time the tube mill is idle owing to relining. Thus it is seen that in favor of the pans are: (1) Lower initial outlay; (2) accessibility; (3) possibility of internal amalgamation; (4) an easily handled unit both in power and capacity; (5) saving in time of renewal of wearing parts.

#### Automatic Agitation.

The design of the ordinary decantation plant for the treatment of slimes may be described as a row of tanks just above the ground level in which the various operations of decantation and washing are carried out. In the ordinary decantation method there is considerable interhandling necessitated by the arrangement of the tanks, but this has been avoided by arranging conical tanks so as to form a separator. The clear liquid flows off the top while the thick pulp flows from the bottom of one vat into the top of the next vat below. An extraction of as high as 93 % has been obtained by this method without any interpumping or handling. There is a stirring gear in the

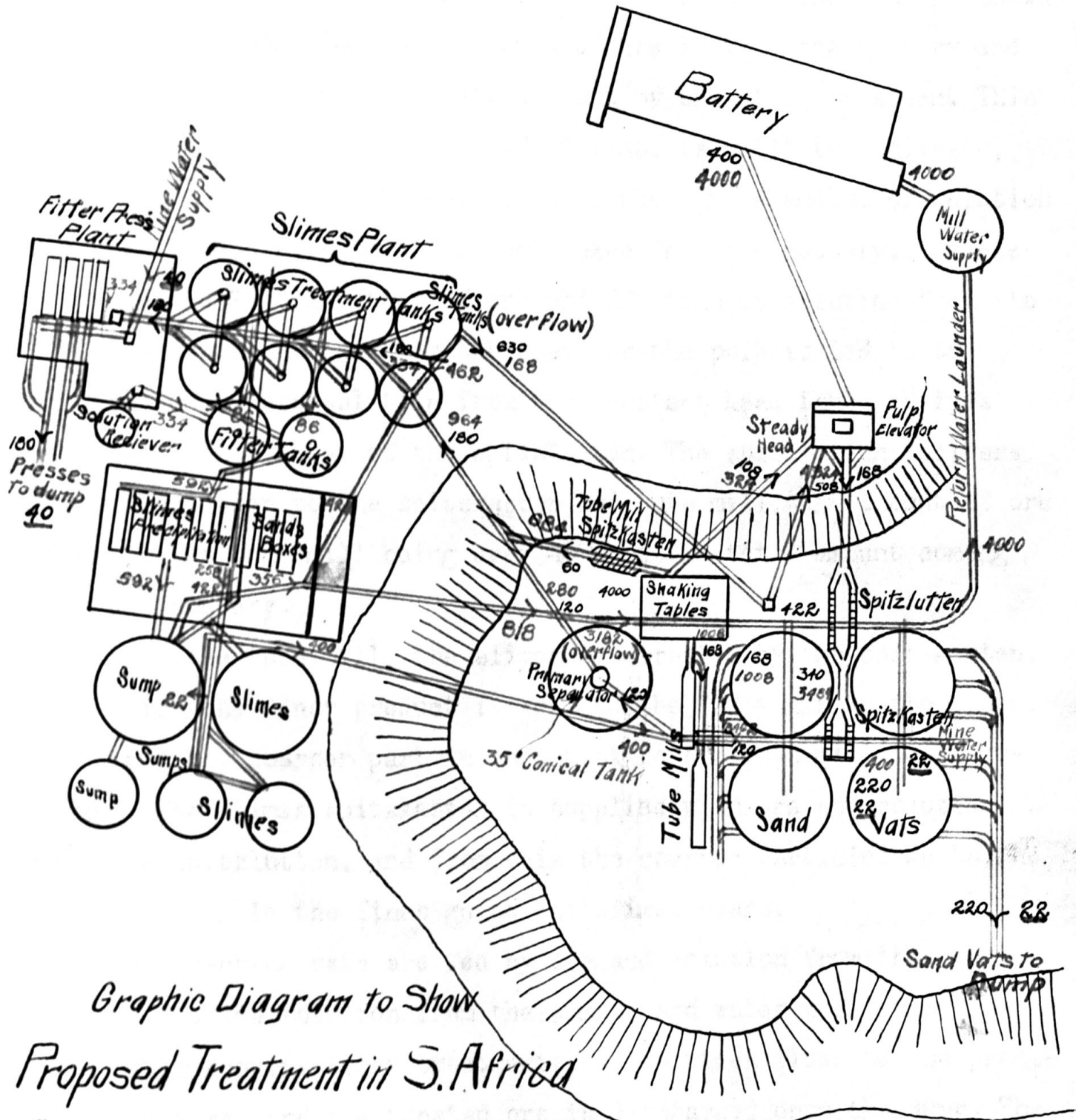
center of the tank. The serious difficulty with this method was that the final product at the discharge was not heavy enough, for too great an amount of solution was lost. This was overcome by lifting the final product to a slimes dam, the solution flowing into a central standpipe. The cost of such a plant is about one-third of that of a decantation plant, and the operating expenses are lower, for gravity does the work of the pumps in the decantation process.

#### Filter Pressing.

The return of the solution from the dam was not as satisfactory as desired, and filter presses have lately been installed to take the place of the dams. The densest product that can be discharged from a decantation plant is one containing about 50 % by weight of dry slimes, and it follows that either extensive washings must be resorted to, which means expense, or a low extraction must be accepted as being satisfactory. One objection that has been raised to filter presses is the cost of the operation; but when it is considered that there is no treatment in the press and that it is to be used merely as a machine for separating gold solution from slimes - the gold having been already dissolved - it must be apparent that the capacity of a filter press will be much larger than where treatment in the press has to be resorted to.

#### Modern Plant.

The accompanying graphic diagram illustrates the distribution of pulp and solution throughout a plant arranged for the new treatment proposed. The basis for the distribution is on the assumption that 400 tons of ore will be crushed every twenty-four hours, using



Graphic Diagram to Show  
Proposed Treatment in S. Africa

Figures relating to ore shown thus 120

" " " Solution " " 280

" " " Water wash " " 22

All Figures indicate Tons per 24 hours

ten tons of solution for every ton of ore crushed. The diagram shows 4,000 tons of solution and 400 tons of ore leaving the battery and this is delivered to the treatment tank by the elevator shown. This elevator has an effective lift of 70 feet, is built in duplicate, either side being able to take care of the entire amount of solution and ore. In addition to the amount taken from the battery, the elevator has to lift 108 tons of ore and 324 tons of solution from the tube mill spitzkasten. From the elevator the pulp is led to the spitzluten, where solution from the constant head tank provides current at the bottom of the spitzluten. The spitzluten delivers ore and solution to the spitzkasten and tube mill, the amount of ore going to the tube mill being over 40 % of the total amount coming from the battery.

From the tube mill, the slimes are treated by the spitzkasten, from which the finer product is sent to the "A" tanks in the slimes plant and the coarser part is sent back to the tube mill to be re-ground. The slimes spitzkasten is supplied with ore and solution from the spitzluten, and from this the coarser particles go to the sands vats while the fines go to the slimes plant.

The leaching vats are fed by ore and solution from the spitzkasten, cyanide solution from the sumps, and water wash from the mine water supply. After treatment, the solution flows to the precipitation boxes and the treated ore is discharged onto the dump. The primary separator is supplied by the spitzkasten and discharges the slime and part of the solution into the "A" tanks, while the greater part of the solution flows into the return water launder.

The separation done by the "A" tank is very good and is meeting with considerable favor in South Africa, where water is rather scarce and loss by evaporation is excessive. When the usual method of economizing mill water; - running into dams - is resorted to on the Witwatersrand, the loss is so great as to cause mills to shut down at times. Here the evaporation amounts to about 500 gallons for every ton of ore treated, or is 60 inches on each surface exposed. With the use of the conical settler the loss has been cut down to less than 200 gallons for every ton of ore treated, owing to the small surface of the tank that is exposed.

In the six slimes treatment tanks, over 93 % of the gold is got into solution, the time of contact in traveling through amounting to about six hours. The slimes treatment tanks get ore and solution from the "A" tanks and solution from the sumps, and deliver into the filter presses and precipitation boxes direct.

The filter presses receive wash water from the mine water supply in addition to the ore and slimes from the slimes treatment tank. They discharge the solution into the precipitation boxes, while the treated ore and wash water goes to the dump. With the dump and precipitation boxes, the cycle is completed and it will be noticed from the figures on the diagram, that no allowance is made for leakage or evaporation of the solution. Such losses and leakages are made up from the mine water supply. The solutions are brought up to their necessary strengths in the extractor house, or, if necessary, in the leaching tanks. The tanks are provided with scraping gear, which consists of a central shaft, to which are two arms connecting on the



inside periphery of the tank with a light angle-iron arm. It should make about one-half revolution per minute to prevent the slimes from settling on the sides of the tank.

There is a decided growing tendency on the Rand to discard milling in favor of breaking and grinding. The efficiency of the stamp mill is lamentably low, while tube mills, edge runners, and grinding pans are fairly efficient. The cost of such a plant as has been described would be about \$ 350,000.00 inclusive of precipitating house, smelting furnace, etc., while with the proposed replacement of stamps by grinding machines, a similar plant to give higher extraction would cost about 35 % less, or probably \$ 225,000.00

Estimated Cost of Treating Leaching Sands.

(Per Ton Treated)

Salaries - - - - -	\$ 0.032
Wages - - - - -	0.068
Contractors - - - - -	0.150
Cyanide - - - - -	0.080
Chemicals, etc., - - - - -	0.042
Zinc - - - - -	0.038
Sundry stores - - - - -	0.016
Clean-up - - - - -	0.028
Colored labor - - - - -	0.036
Power - - - - -	0.068
Repairs - - - - -	0.032
Light - - - - -	<u>0.032</u>
Cost per ton treated - - - - -	\$ 0.622
Cost per ton milled - - - - -	\$ 0.373

Estimated Cost of Treating Current Slimes  
 Recieved From Primary Settler.

Solution of gold:

Wages - - - - -	-\$ 0.020	
Sundry stores - - - - -	0.002	
Chemicals - - - - -	0.120	
Power - - - - -	<u>0.020</u>	\$ 0.162

Filter pressing and residues:

Wages - - - - -	0.180	
Sundry stores - - - - -	0.020	
Cloths - - - - -	0.020	
Air - - - - -	0.010	
Machinery repairs - - - - -	0.050	
Power - - - - -	<u>0.040</u>	0.320

Precipitation and clean up:

Wages - - - - -	0.020	
Zinc - - - - -	0.030	
Acid - - - - -	0.030	
Sundry stores - - - - -	<u>0.010</u>	0.090

Supervision:

Salaries - - - - -	<u>0.040</u>	<u>0.040</u>
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Cost per ton treated - - - - -		\$ 0.612
Cost per ton milled - - - - -		\$ 0.289

These figures may be compared with the following actual costs of filter pressing 18,000 tons per month at the Horse Shoe Gold Mine, West Australia:

Solution of gold:

Attendance - - - - - \$ 0.0192

By agitation (16 hours).

Power - - - - - 0.0740

Light - - - - - 0.0008

Repairs - - - - - 0.0256

\$ 0.1196

Pressing:

Attendance - - - - - 0.0106

Power - - - - - 0.0542

Repairs - - - - - 0.0326

Washing - - - - - 0.0106

Maintenance - - - - - 0.0024

Emptying (labor) - - - - - 0.1540

Light - - - - - 0.0020

Air - - - - - 0.0800

Cloths - - - - - 0.0260

0.3724

Cost per ton treated - - - - - - \$ 0.4920

## HOMESTAKE.

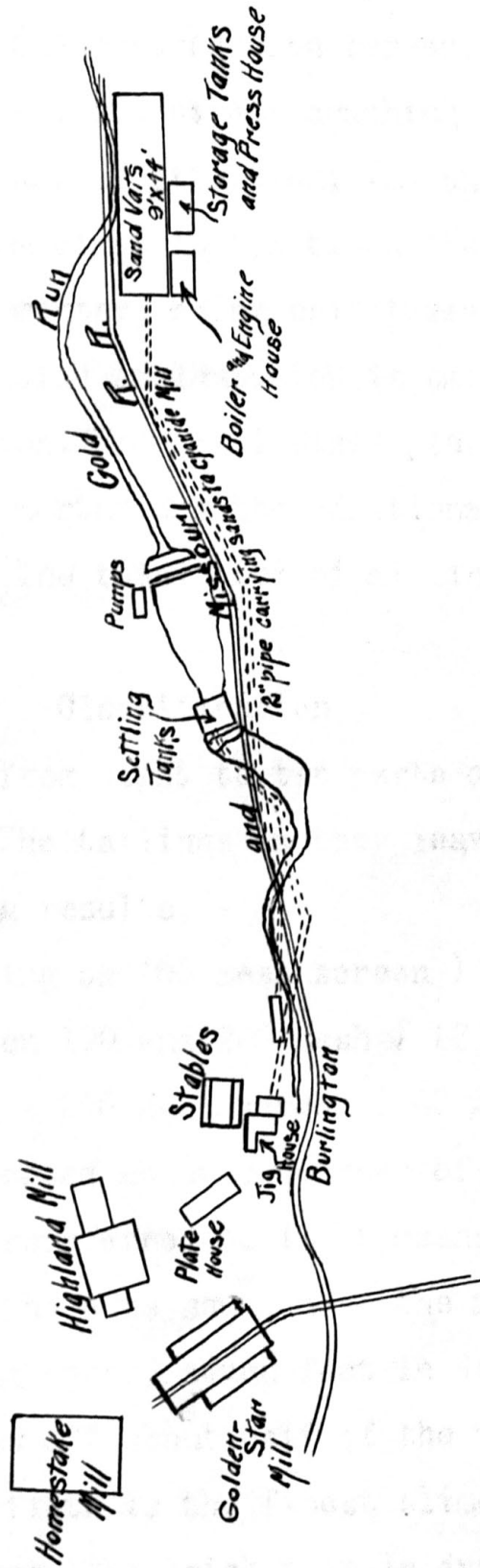
Undoubtedly one of the best cyanide plants in the United States is that of the Homestake Mining Company, near Lead, South Dakota. This company now owns or controls over 250 claims, covering about 8,000 feet along the strike of the lode. There is sufficient low grade ore blocked out on this property to keep the mills in operation for over twenty years if no further development work is done. The present capacity of the plant is 4,000 tons per day, being the largest sand treatment plant in the world.

The ore is obtained from open cuts, but principally from underground workings, the lowest level worked being 1,100 feet below the surface. The vein is from 300 to 500 feet wide, the rock of both walls being carbonaceous slate penetrated by a system of porphyry dikes. The Homestake lower level ore, comprising the greater part of that mined at Lead, is a hornblende, garnetiferous shist or slate, which has been crushed and infiltrated with free silica and pyrite, the latter being about seven or eight percent of the ore, and comprising pyrite, pyrrhotite and traces only of arsenopyrite and chalcopyrite.

### Milling

The ore is first broken in rotary breakers at the hoist to a maximum size of four inches. From the hoist bins at Lead, the broken ore is trammed to the several stamp mills which have a combined capacity of 1,000 stamps. The ore then passes from the mill bins to the stamps which are of the Homestake pattern, weighing about 900 pounds each and having a drop of ten and one-half inches, with a

# Plate II



General Plan Showing Location of Homestake Cyanide Plant

Scale: 1" = 400'

fall of eighty-one times per minute. These stamps maintain a capacity of fully four tons per day, yielding a product of which about eighty percent passes a one-hundred mesh screen. This duty is made possible by: (1) the slates and pyrite crushing readily and the quartz forming a good means of attrition; (2) the large proportion of water used, being from eight to ten times the weight of the ore crushed; (3) the narrow mortar, being only twelve inches wide at the lip. Both outside and inside amalgamation is employed as the pulp is excellent for amalgamation. Four full sized plates are arranged in series in front of each mortar and the additional plates are credited with a great saving. The total cost of milling at Lead is about forty cents per ton.

#### Classification

The pulp contains from eight to ten parts of water for every part of ore by weight. The tailings as they leave the mill are classified with the following results, -

COARSE ( remaining on 100 mesh screen ) 22 %

MIDDLES ( between 100 and 200 mesh ) 18 %

FINES ( passing a 200 mesh screen ) 60 %

The cyanide plant is located about a quarter of a mile from the Lead mills, the pulp being transferred to it by means of a 12 inch pipe line, which leads from the cone house near the mills. At the cone house are twelve settling cones, seven feet in diameter and with 50 degree sides which throw off about half of the water and a fifth of the solid matter. This fifth is the finest slime, about 1.76 % being middles and 98.24 % fines. The thick pulp is drawn from the bottom

of the cones and contains all of the leachable materials and some of the slimes. Six more gravity-settling cones in the plant proper continue the classification, the sizing being about the same as that done in the cone house. The overflow is in composition, similar to that of the first set of cones, and is conducted to a settling tank, whence it is drawn for the purpose of sluicing out the leachable material after its treatment has been completed. This practice will soon be changed, as is latter explained, for it is this overflow that contains the slimes. The underflow from the settling cones is quite thick and passes to 36 hydraulic classifying cones, arranged to discharge the sand and introduce water. This completes the classification, which is rather lengthy but does away with the costly operation of double treatment. At the Homestake no dependence is placed in the German spitzluten and spitzkasten, the stand being that they are very imperfect machines when compared with modern cone-classifiers.

#### Cyanide Treatment

The pulp has been separated into non-leachable slimes, about 30 % of the ore crushed, a direct leaching product amounting to, 70 % of the tailings, which, although clean and free from mud, is still of a very fine texture. Following is an average test of the sizing:

COARSE (Remaining on 100 mesh) 40.5 %  
MIDDLES ( 100 to 200 mesh ) 30.8 %  
FINES ( Passing 200 mesh ) 28.7 %

An even finer product than this may be treated from ores obtained

from the lower levels. The leaching rate maintained is from three to four inches per hour. The leachable pulp containing about 10 % of pyrite is ready for the vats, where from 3 to 5 pounds of lime per ton is added. If the lime is added in the stamps the amalgamation is seriously interfered with. It has been found here that the average size of the particles of lime that go into the vats in an unslacked condition, has an important bearing on the subsequent cyanide decomposition and gold precipitation. This is due to the amount of easily decomposed sulphides in the Homestake ore that require a low and somewhat constant alkalinity for successful treatment. The lime should be pure, for the magnesia of impure lime is very detrimental to the cyanide operation.

The classified pulp having been made alkaline, it is passed to the distributors which travel from one vat to another. There are fourteen vats, each 44 feet in diameter, 9 feet deep and holding 610 tons of sand. It takes about 11 hours to fill one vat, permitting about 5 days contact with the solution. After filling, the drain-valve is opened, the top leveled, and the stronger stock solution, 0.14 % KCN, is run on. Contact with this solution is about three days, including drainings for the purpose of drawing in air. This solution having a normal strength of about 0.14 % KCN, is run to the weak precipitation tanks, which are twenty-six feet in diameter by nineteen feet deep, and hold 300 tons of solution. Following the three days treatment with the strong solution, is a two days treatment with a solution having a normal strength of 0.10 % KCN. Then this solution is run to the strong precipitation tanks which are similar



to the weak precipitation tanks. After the weak solution has been drawn off, wash water is brought into the charge, and the washing continued until the solution is down to 0.03 or 0.02 % KCN and from 5 to 7 cents in value per ton.

The charge is then sluiced, which is done by two men in about two hours. Formerly a three inch hose with low pressure was used at the first of the sluicing, to employ the overflow from the settling cones, but the installation of the slimes plant will do away with this and clear water under high pressure will be used for the entire sluicing operation. The canvas filter which covers a cocoa matting filter on the bottom of the tank is washed clean. The tank is then filled with clear water and is ready for the next charging.

#### Precipitation.

The strong solution which was run to the weak precipitation tanks has a value of about \$ 2.00 per ton and a strength of 0.10 % KCN. When one tank is full, or containing about 300 tons of solution it is agitated with compressed air, about 60 pounds of zinc powder being sprayed in as an emulsion at the same time. The mixture is then pumped through two large filter presses, 36 inches square, containing twenty-four frames each four inches in depth. The cloth and frame catches the gold, silver and excess of zinc, and the barren solution passes to the weak solution storage tank, whence it passes to the sand vats as weak solution. Its value has been reduced by this operation from \$ 2.00 to 5 or 10 cents per ton, being a precipitation of from 95 to 97.5 %. This efficiency is attributed to the fact that the cloths are coated with about an eighth of an inch of zinc and pre-

precipitate, compelling all of the solution to get into intimate contact with the fine zinc, which is true of no other precipitation process. These presses are not opened for a month and then two men clean them out in about six hours, obtaining about \$ 50,000.00 worth of precipitate. This time includes the replacing of worn cloths and is much cheaper than any other form of clean up,- as from zinc boxes or electrolytic precipitation.

The weak solution used to leach the second time was run to the strong precipitation tanks, or rather collecting tanks, for they are made up to 0.14 % KCN and pumped directly to the strong solution storage vats, without having been precipitated. It is then sent to the early treatment of the charge, although it contains values of from 30 to 50 cents per ton at the outset. In this way the weak solution of the one treatment becomes the strong solution of the next treatment, and the values are all collected in the weak precipitation tanks.

#### Refining Precipitates.

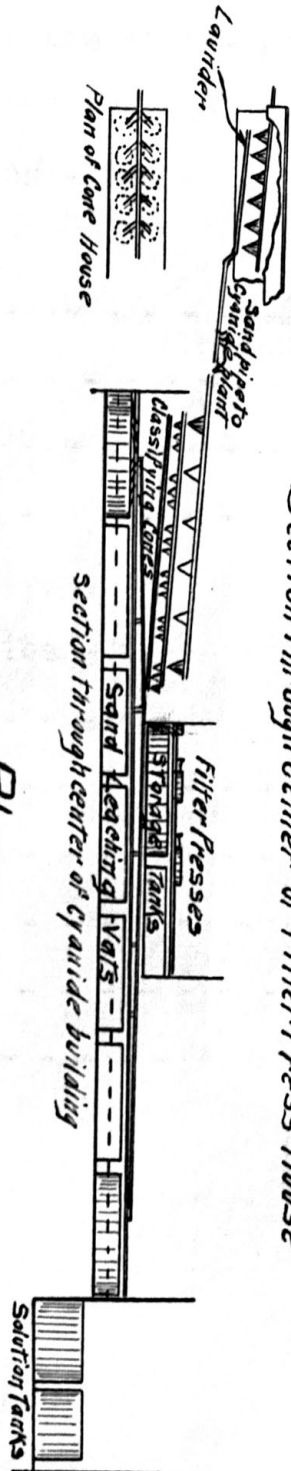
Ordinarily the losses from refining zinc box precipitates are from 2 to 5 % of the total values precipitated, but by the Homestake method, less than 1 % is lost. After the precipitate is removed from the presses it is placed in lead lined vats, equipped with an exhaust fan and a hood. Here the precipitate is treated with dilute hydrochloric acid, and mechanically agitated. After settling, the supernatant liquid is forced through a filter press with compressed air. Sulphuric acid is next added, and the mixture is agitated and heat applied. It is again settled and run through a press. Wash water is

then added and the whole mixture put through the press, where it is further washed. The total values of the wash water and acid liquors flowing from the press from a \$ 50,000.00 treatment is less than \$ 20.00, and part of this is caught in a large settling tank. The precipitates after treating with acid is then dried by steam. Not all of the moisture is removed, but it is then thoroughly mixed with coke, litharge, borax, and silica, and sprinkled with a solution of lead acetate. This mass is then briquetted under a pressure of 4,000 to 6,000 pounds per square inch. The zinc having been removed by the acid treatment, and the briquettes having been dried, a borax slag develops on the outside when charged to the cupel, and they fuse quietly, quickly, and at a low heat, without dust or volatilization losses. The lead button carries the values, and this button is cupelled. The resultant metal is then 975 to 985 fine, and is cast into bars. The cupel slag and cupel bottom is treated in a blast furnace to recover the lead, the blast furnace slag being worth less than \$ 5.00 per ton. The total cost of this refining amounts to less than three-fourths of one percent.

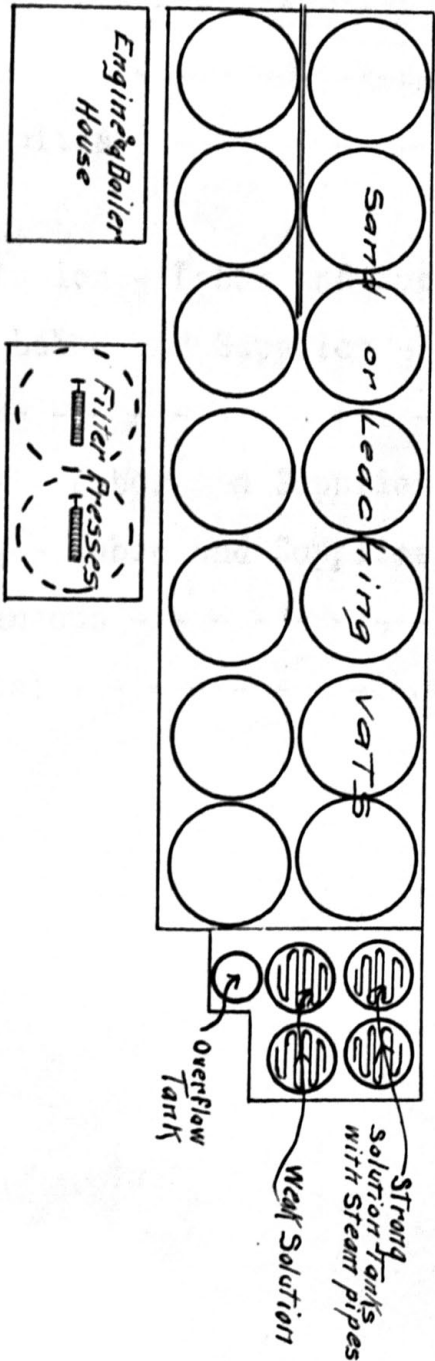
The average extraction from the Homestake plant is 74.7 %, which would be low for an oxidized or porous ore; but, since such a high extraction is obtained from the amalgamated plates, since the tailings are very low grade, and since the values are largely disseminated throughout the ore, it is the economic percentage, yielding the maximum profits.

# Plate III

Elevation of Cone House      Section Through Center of Filter Press House



Plan



Plans and Sections of Homestake Cyanide Works.

Scale: 1" = 7'

### HOMESTAKE COSTS.

The following are average costs per ton at the Lead cyanide plant, where the average values of the ore treated is \$ 1.65 per ton:

Classification-Labor and supplies - - - - - \$ 0.017

**Treatment:**

Cyanide - - - - - -\$ 0.152

Labor - - - - - 0.030

Lime - - - - - 0.022

Supplies - - - - - 0.005

0.209 0.209

Precipitation - Labor and Supplies - - - - - 0.026

Power - Labor and Supplies - - - - - -0.051

Water - - - - - 0.026

Assaying - Labor and Supplies - - - - - 0.013

Refining - Labor and Supplies - - - - - -0.006

Miscellaneous - - - - - 0.005

Total - - - - - -\$ 0.353

## HOMESTAKE SLIMES PLANT.

The success of the Homestake cyanide plant is almost wholly due to C. W. Merrill, who worked out the method of treatment employed in the sands plant. He holds patents on these processes, even to the refining of the gold, and the dividends being paid by the Homestake company is sufficient evidence of the efficiency of the method. In December, 1904, patents were taken out by Merrill to control the treatment which he has devised for the slimes plant, which is in course of construction below the Homestake cyanide plant. The slimes which the overflow from the gravity-settling cones in the cyanide plant, have been used to wash out the treated material from the sand tanks heretofore, thus being wasted. These slimes carry values of about \$ 1.10 per ton, and from an exhaustive series of tests and experiments conducted previous to applying for patents, the process will permit of a considerable saving even on slimes so low grade as those at the Homestake. As the plant is not in operation, it is impossible at this time to give an accurate description of the process or to give details of the success of the plan.

Two patents were obtained by Merrill to control this method of treating slimes - an invention of a process of treating precious-metal-bearing materials, and, a process of removing solid or semi-solid material from containers or pressure filters. The latter is that of an improvement for a filter press.

It is a well known fact that often cyanide solutions have little or no action on ores or tailings containing reducing salts, such as iron. The oxygen is absorbed from the solution and the dis-

solving action is thus prevented. There is also a great consumption of cyanide in such cases. Merrill proposes to remove any liquid from the interstitial spaces of the slimes, and then submit the slimes to the action of compressed air. When this higher oxidation is complete the metallurgical treatment is continued. It is essential that the container be such as will permit the oxidizing agent to be applied under pressure after the removal of the liquid. After all reducing agents have been oxidized the material is subjected to cyanide treatment as applied at the Homestake.

Merrill's improved filter-press method depends upon the fact that in ordinary filter presses, the units of which the press is composed, have to be separated by hand and the material which was retained on the cloth would have to be separated from the distance frame or container; but he proposes to get this material out of the units without handling or moving the units. To accomplish this, an inlet is provided for each distance frame or container, through which a liquid or gas is introduced under pressure and the solid or semi-solid or unfilterable contents sluiced or forced out through an exit from the frame. This exit may be the opening through which the material to be filtered is originally introduced or it may be an independent outlet.

This arrangement permits of a much greater capacity than in the old presses, for the time required for handling the units is greatly reduced. The cost of treatment is likewise reduced, the capacity being greater, the wear on cloth being less, and the length of press being reduced.

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