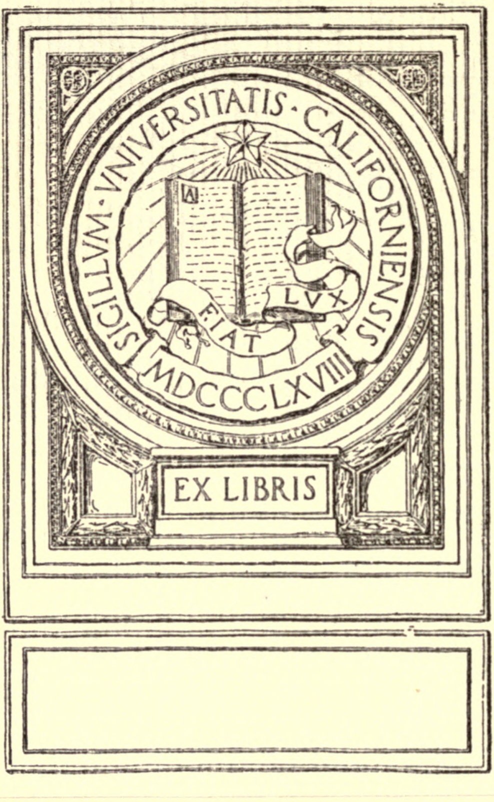

MANUAL OF CYANIDATION

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MANUAL
OF
CYANIDATION

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PREFACE

The aim of the present work has been to provide in a handy sized volume a compendium of all the ascertainable facts concerned with the cyanide process that may have a practical bearing on testing an ore, planning the flowsheet and operating the plant when erected. Purely theoretical questions have therefore been kept in the background for the most part.

The substance of the book is necessarily in large part a compilation and the author has not hesitated to quote verbatim from other writers in cases either where he has had no personal experience of the subject dealt with or where the matter treated of would seem to suffer in clarity and usefulness by being paraphrased or condensed. Credit has been given for discoveries and statements in every case where it was possible to ascertain their origin.

Explanations of procedure have often been gone into in more detail than would have been necessary for the use of those thoroughly familiar with the working of the process, but by so doing it was hoped that the book might be found useful to the student as well as to the advanced metallurgist, and that it would find a place on the laboratory bench as well as fulfilling the function of a *vade mecum* for the metallurgical engineer on his travels.

E. M. H.

San Francisco, Cal.
November, 1919

INTRODUCTION

What is commonly known as the Cyanide Process has for its object the commercially profitable recovery of gold and silver from their ores, and this is accomplished by the solvent action of an alkaline cyanide solution on the precious metals.

The first step in the process is to mill or grind the ore to a fine state of subdivision so as to expose each particle of precious metal to the action of the solvent. The degree of comminution necessary varies for each kind of ore and can only be determined by careful experiment, but it may range from a product whose largest particles will just pass through a No. 30 sieve (aperture about 0.016 in.) to one in which the whole material will pass a No. 200 sieve (aperture about 0.0025 in.).

Whatever form of crushing and grinding machinery be employed, the resulting pulp is composed of a great variety of sizes of ore particles, ranging from the maximum screen size decided upon down to grains whose angular outline can only be seen through a microscope. Mingled with these fine particles of rock there is always more or less impalpably fine and structureless matter of a clayey type, known as colloidal material. Such a variety in the constitution of the pulp produced by the mill renders it impracticable to use a single mechanical method in the application of the cyanide solution to the ore, so the pulp is classified into two products, (*a*) "sand," or that part which is sufficiently coarse to allow a solution to percolate through it by gravity, and (*b*) "slime," or that part, consisting of the finer siliceous particles and the colloidal matter, which is only percolable, even in thin cakes, by the use of some form of pressure.

"All-sliming" consists in reducing the whole of the ore to such a fine state of subdivision that it is practicable to treat it by the mechanical methods used for treating slime.

Besides containing cyanide, the solution to be used for treating the ore must contain some additional or free alkali, because nearly all ores contain acid or tend to develop it during the course of treatment, and acids react with the alkaline cyanide to form hydrocyanic acid, HCN, thus involving a loss of the active reagent KCN. The alkali most commonly used is lime, both because of its cheapness, and also because some lime is needed in any case to coagulate and settle the separated slime preparatory to the slime treatment process. Caustic soda may also be used as protective alkali, though lime is usually preferable for chemical reasons, apart from other considerations.

When the precious metals have been dissolved out of the ore it still remains to recover them from the solution, or "precipitate" them, in such a way that the solution may be available for re-use with as little loss of cyanide as possible. This is done most commonly by the use of metallic zinc, which dissolves in the solution, while the gold and silver are left behind as a fine metallic sludge. This sludge is afterward collected, filter-pressed, and melted down with suitable fluxes into bullion bars.

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CHAPTER I

NOTES ON THE CHEMISTRY OF THE PROCESS

The usual reaction given for the dissolution of gold and metallic silver in cyanide solution is known as Elsner's equation.

$$2Au + 4KCN + O + H_2O = 2KAu(CN)_2 + 2KOH$$

Silver sulphide, the form in which silver most commonly occurs in its ores, involves a different set of reactions, which are usually expressed thus:

First stage, $Ag_2S + 4KCN = 2KAg(CN)_2 + K_2S$

This, being a reversible reaction, cannot proceed far before reaching equilibrium, unless the product K_2S is removed out of the sphere of action. The potassium sulphide, however, happens to be very sensitive to oxidation so that a change rapidly takes place, probably in two directions,

(1) $K_2S + KCN + O + H_2O = KCNS + 2KOH$
(2) $2K_2S + 2O_2 + H_2O = K_2S_2O_3 + 2KOH$

This thiosulphate would tend later to oxidize to sulphate,

$$K_2S_2O_3 + 2KOH + 2O_2 = 2K_2SO_4 + H_2O$$

and perhaps also more sulphocyanate would be formed, thus, $K_2S_2O_3 + KCN = K_2SO_3 + KCNS$.

The action of cyanide on the haloid compounds of silver, *e.g.*, horn silver, differs from those already given in that oxygen does not appear as an indispensable auxiliary:

$$AgCl + 2KCN = KAg(CN)_2 + KCl$$

Reaction of Cyanide with Silver Sulphide.—In connection with the reactions usually assumed to explain the action of cyanide on silver sulphide it is interesting to note that a solution of the double cyanide of potassium and silver in presence of an excess of potassium cyanide will tolerate the presence of an appreciable amount of soluble sulphide without precipitation of the silver. The reaction expressed by the equation

$$Ag_2S + 4KCN = K_2S + 2KAg(CN)_2$$

is reversible, and its direction depends on the relative proportions of free cyanide and soluble sulphide, and the degree of their concentration. J. W. Sharwood in a letter to the Mining and Scientific Press (Sept. 20th, 1908) says

"The reaction has been more recently studied by Berthelot who found that in dilute $\left(\frac{N}{10}\right)$ solutions nearly 100 molecules of KCN were required to balance one molecule of K_2S in order to retain silver in solution, instead of four molecules as the equation seems to indicate. To be exact, the equation given by Berthelot as representing the conditions of equilibrium is

$$96KCN + Ag_2S = 2KAg(CN)_2 + 92KCN + K_2S$$

Now this proportion (96 mol. KCN to 1 mol. Ag_2S) means that 96 × 65 parts of KCN are required to dissolve and hold in solution 2 × 108 parts of silver in the form of sulphide, or 28.9 to 1, if none of the sulphide is oxidized."

The above proportions, however, would appear to be different for different concentrations of cyanide because Sharwood states in another place that

"if we dissolve a fairly large amount of Ag_2S in strong solution of KCN and dilute it shortly afterward more or less of the Ag_2S will be reprecipitated proving that some of the sulphide radicle remained in the solution;"

and showing also that the degree of concentration of the cyanide as well as its amount materially affects its capacity of holding silver in solution against the opposing force of Na_2S. Of course if the Na_2S is removed out of the sphere of action either by oxidation or precipitation as an insoluble sulphide the equilibrium is disturbed and silver sulphide continues to dissolve.

In the Mining and Scientific Press of September 26th, 1914, Harai R. Layng advances an interesting theory that silver in the form of the sulphide goes into solution as a sulphocyanate dissolved in cyanide solution, with the formula $AgCNS.KCN$. While many might not be prepared to accept without reservation his statement, rather positively made, that silver "usually enters the solutions in this form" yet the possibility is worth considering, and may throw light on some of the obscure phenomena of the process. He does not give any equation to illustrate the reaction, but the following is suggested by the present writer.

$$Ag_2S + 4KCN + O + H_2O = AgCNS.KCN + KAg(CN)_2 + 2KOH$$

The Function of Oxygen.—Oxygen appears to be an indispensable factor, either directly or indirectly, in the dissolution of gold and silver by cyanide solutions, except in the case of the haloid compounds of silver. Whether the action of oxygen in the dissolution of gold be a direct one, as illustrated by the Elsner equation, or an indirect one, in the sense of acting merely the part of a depolarizing agent, as maintained by Julian and Smart,[1] will not affect the general statement that it is a necessary adjunct for the dissolution of the precious metal.

The most generally useful agent for this purpose is atmospheric oxygen, and in many instances sufficient oxygen is absorbed by the solutions in their circulation through the plant to accomplish all that is necessary. Ores that contain reducing constituents may need more oxygen than is obtained in this way, and the additional amount is most easily supplied in the case of percolation by draining dry between washes, and by the use of air lifts or centrifugal pumps, in agitation.

The addition of chemical oxidizing agents, such as potassium ferricyanide, permanganate, sodium peroxide, and ozone, has not been attended with much success, both on account of their cost, and also because of their tendency to oxidize the cyanide to

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[1] Cyaniding Gold and Silver Ores, page 71 (Second Edition).

cyanate ($KCNO$), which is useless for the purposes of the process. They have sometimes been found useful, however, for oxidizing a highly reducing pulp before adding cyanide, as in the case of slime which has stood in dams for long periods, undergoing partial oxidation with formation of ferrous compounds.

The only reagent of the oxidizing class that has attained to any importance commercially is bromocyanogen, $BrCN$, and that only in the raw treatment of telluride and mispickel gold ores.

Julian and Smart[1] consider that the activity of this compound is not due to the liberation of cyanogen, though that probably occurs, but to a liberation of oxygen, in the sense of the equation.

$$2BrCN + KCN + 4KOH = 2KBr + 2KCN + KCNO + 2H_2O + O$$

Reducing Agents.—Oxygen being necessary in almost every instance for the dissolution of gold and silver by cyanide, it follows that any substance that has the property of denuding the solution of its dissolved oxygen will retard or completely stop the action of the cyanide. It is a common practice to test working solutions for their "reducing power" by acidifying and titrating with standard permanganate. Such a test, however, is misleading in its bearing on the dissolving efficiency of a solution, because it includes the reducing effect of such substances as ferrocyanides and sulphocyanates which have not the power of absorbing oxygen from solutions and therefore do not retard the dissolving effect of the cyanide on account of their reducing character, in fact, they are not reducers when considered from that standpoint. If, then, an estimate of the detrimental reducers present be desired the ferrocyanides and sulphocyanates should be determined separately and their equivalent in terms of standard permanganate deducted from the "total reducing power," as already found.

$$1\text{ cc }\frac{N}{10}\text{ permanganate} = 0.001619\text{ grm. }KCNS$$

$$1\text{ cc }\frac{N}{10}\text{ permanganate} = 0.036831\text{ grm. }K_4Fe\,(CN)_6$$

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[1] Cyaniding Gold and Silver Ores, page 81 (Second Edition).

Among harmful reducing agents may be mentioned organic matter, sulphuretted hydrogen, sulphurous acid, ferrous sulphide (all of which may be present in old accumulated slime dams), ferrous sulphate (which in alkaline solutions becomes ferrous hydroxide), soluble sulphides (rarely found in normal working solutions), and thiosulphates.

Effect of Temperature.—The rate of dissolution of metals in cyanide solution increases with rise of temperature. Julian and Smart[1] state that in a special series of experiments made to elucidate this point the solubility of gold increased to a maximum at 85 deg. Cent. (185 deg. F.) and then slightly decreased to the boiling point. This principle has in some instances been applied in practice, especially in the treatment of silver ores, notably in the Tonopah, Nevada, district. In most cases, however, the increased consumption of cyanide due to heating the solutions in the presence of ore more than offsets the increased extraction. From 38 deg. C. (100 deg. F.) upward the decomposition of cyanide is usually very marked, but in cold climates it is often found advantageous to maintain the temperature of the working solutions at between 15.5 and 21 deg. C. (60 to 70 deg. F.) by artificial heating.

Dissolving Effect of Zinc-potassium Cyanide.—Many writers, including Feldtmann, and Julian and Smart, maintain that gold will dissolve in pure $K_2Zn(CN)_4$ in the absence of free cyanide. If this be true of gold it would probably apply also to metallic silver, but silver compounds such as argentite do not appear to be acted on in this way. A series of experiments carried out by the writer with pure silver sulphide precipitated dried and pulverized in the laboratory showed that zinc-potassium cyanide solution equivalent to 0.3% KCN had almost no dissolving effect upon it during a 24 hour agitation in a bottle open to the air, while in solutions of the double cyanide to which varied amounts of free cyanide were added the dissolution of silver was in every case closely proportional to the quantity of free cyanide present, and corresponded with the weights of silver dissolved by solutions

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[1] Cyaniding Gold and Silver Ores, page 92 (Second Edition).

containing similar strengths of free cyanide, but with the double cyanide absent.

Cyanide Regeneration.—Arising out of the subject of the dissolving effect of the double cyanide is the question whether any part of this combined cyanide becomes available for dissolution purposes in the course of its circulation through the plant. Although the fact is doubted by some writers the indications certainly point to such a regeneration of free cyanide. This may take place from two causes, first, a precipitation of zinc by soluble sulphides formed during contact of the solution with the ore, $K_2S + K_2Zn(CN)_4 = ZnS + 4KCN$, and second, a reaction of the free alkali on the double cyanide with the formation of an alkaline zincate and free cyanide, $K_2Zn(CN)_4 + 4KOH = K_2ZnO_2 + 4KCN + 2H_2O$. This reaction is no doubt reversible so that the dissociated cyanide would tend to re-combine with zinc if any change occurred in the solution to disturb the equilibrium. Where the alkali employed is lime instead of caustic soda or potash there may be a formation of insoluble calcium zincate which renders the reaction non-reversible. In the operation of cyanide plants many metallurgists have observed that a saving in cyanide is effected by raising the lime content from a merely "protective" amount up to 0.1 % or 0.13% CaO. When the solution from the barren sump is thrown back to the mill and comes in contact with ore and fresh lime the free cyanide indicated by titration will often be found to rise in strength 40 or 50%, and its extractive efficiency will be increased in proportion to the increase of free cyanide as shown by the free cyanide determination (method No. 2, page 22). In a laboratory experiment made by the writer a solution of zinc-potassium cyanide was made up showing no free cyanide when tested by method No. 2, and having practically no dissolving effect on prepared Ag_2S. An excess of lime was added to a portion of the solution, and after a few hours' agitation it was filtered clear. A titration now indicated that about half of the combined cyanide originally present had been decomposed, and was recorded as free cyanide. This solution, when tested on prepared silver sulphide, had a dissolving efficiency equal to a freshly prepared cyanide solution of the same titration strength.

Crosse's Process.[1]—Some years ago a process of cyanide regeneration was patented by Andrew F. Crosse based on the principle of precipitating the zinc as sulphide by addition of sodium sulphide to the solution, $K_2Zn(CN)_4 + Na_2S = ZnS + 2KCN + 2NaCN$. For this purpose the solution was to be heated to about 65 degrees C. (149 deg. F.), and the reagent added until a faint permanent trace of soluble sulphide was indicated in the solution on addition of a few drops of lead acetate. This, however, did not complete the process, because a difficulty arose due to the almost invariable presence in the stock solution of sodium zincate, Na_2ZnO_2: this zincate was acted upon by the sodium sulphide to form caustic soda; $Na_2ZnO_2 + Na_2S + 2H_2O = ZnS + 4NaOH$. The caustic soda, gradually building up in the stock solutions, attacked the zinc in the precipitation boxes, involving an additional zinc consumption and also causing such an evolution of hydrogen as to carry precipitate out of the boxes and send up the assay of the tails to a prohibitive extent. Crosse proposed to meet this condition as follows: when the caustic soda had accumulated to a dangerous extent in the stock solutions the use of Na_2S was to be temporarily suspended and the reaction was to be effected by charging the tankful of solution with hydrogen sulphide generated from the zinc sulphide already filtered out and accumulated during the first stage of the process, causing the reaction, $H_2S + 2NaOH = Na_2S + 2H_2O$. This part of the procedure, however, has so far proved impracticable on a working scale, and the process as a whole has not found any useful application.

It seems likely that the first stage of Crosse's process might attain practical importance when aluminium is used as the precipitant, in the case of certain ores in the cyaniding of which zinc is detrimental to extraction of the silver, and yet which contain some zinc soluble in the cyanide solutions.

Several other methods of regenerating cyanide solutions have been proposed, chiefly based on the principle of precipitating

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[1] *Journal* Chem. Metall. and Min. Soc. of S. A., Mar., 1903.

the base metal contents with acids, but none of them has attained to any commercial importance.

Fouling of Solutions.—This expression is indefinite, and is usually used to convey the idea of gradual deterioration in the dissolving power of a mill solution during repeated contact with the ore, due to a progressive accumulation in the solution of deleterious substances contained in the pulp. One class of such substances, already alluded to, consists of reducing agents, organic or otherwise, that have the property of denuding the solutions of their dissolved oxygen, and the remedy is obviously oxidation in some form. In regard to the accumulation of base metals, there is a tendency for the activity of the solution to be retarded by the presence of any unnecessary substance, due to an increase in viscosity and the consequent resistance to the migration of the ions. It seems doubtful, however, whether there is any ulterior and more serious harm done by the presence of these metals. Copper is often said to diminish the dissolving activity of a cyanide solution to a very serious extent, but the writer has not seen conclusive evidence of this in solutions assaying up to 2 or 3 lb. of copper per ton. Julian and Smart[1] state that "the solubility of silver is usually small when copper is present, whereas the solubility of gold is not generally affected to any thing like the same extent." The zinc derived from precipitation seems not to affect extraction to any appreciable extent in the majority of instances, but there are cases especially when treating ores containing antimony and arsenic, where the presence of zinc lowers the extraction of the silver as much as 10% and in a less degree that of the gold. When zinc is found to have this effect, lead in solution will often act similarly, and the addition of lead salts may have to be avoided. The writer has, however, recently come across several instances where the presence of zinc from zinc precipitation was distinctly detrimental to silver extraction though the addition of litharge was beneficial.

Strength of the Solution in Cyanide.—When the cyanide process was first introduced a great point was made of the selective action

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[1] Cyaniding Gold and Silver Ores, page 113 (Second Edition).

of weak solutions, it being asserted that a weak solution had a tendency to select the gold and silver for attack in preference to the base metals. Julian and Smart's experiments,[1] on the contrary, went to show that the ratio of gold to pyrite dissolved in a weak solution did not differ from the ratio of that in a strong solution. However that may be, it is a fact well known in practice that the use of a solution stronger than is necessary involves an increased chemical loss of cyanide without any corresponding gain in extraction of the precious metals. The mechanical loss in residues will obviously be greater with strong solutions than with weak, so that from every point of view it is important not to use a stronger solution than is necessary to obtain the maximum extraction that will be commercially profitable. The most suitable strength for any given ore can only be found by experiment, but, in general, silver ores require much stronger solutions than gold ores. In the treatment of slime the most usual cyanide strength for all-gold ores ranges from 0.005% to 0.05%, while for silver and silver-gold ores it may be 0.1% to 0.15 %, in terms of KCN, and in Pachuca strengths up to 0.4% KCN are frequently used. In the leaching of sand it is usual to employ a much stronger solution than in the agitation of slime, the idea being to obtain dissolution with a small volume of strong liquor in a short period of time so as to allow as long a time as possible for the subsequent washing out of the dissolved metals. For sand leaching, whether of gold or silver ores, the strength of cyanide commonly ranges from 0.2% to 0.3% KCN.

Sources of Cyanide Loss. (1) *The Precious Metals.*—The amount of cyanide that combines with the gold in an average ore is negligible, but with silver it is different owing to the much larger quantity of metal that has to be dealt with to produce commercial results. If the final result be summarized by the equation, $Ag_2S + 5KCN + O + H_2O = 2KAg(CN)_2 + KCNS + 2KOH$ it appears that 4 molecules of cyanide are appropriated by 2 atoms of silver, while a fifth molecule goes to form sulphocyanate, a substance which will be alluded to later. Leaving

[1] Julian & Smart, p. 86 (2d ed.).

the latter out of account for the present, the quantity of cyanide tied up in combination with silver amounts to about 0.08 lb. KCN or 0.06 lb. NaCN per troy oz., and 1.2 grm. KCN or 0.9 grm. NaCN per gram of silver, so that the apparent loss of cyanide from this cause may be considerable.

2. *Loss in Zinc Precipitation.*—In precipitating the silver from the silver-potassium-cyanide combination by means of zinc, not only is the combined cyanide not liberated, but there is a further consumption involved. The reaction in the zinc boxes is rather obscure, but tests for free cyanide made on entering and leaving almost invariably show a marked loss of strength, amounting sometimes to as much as 0.03% or 0.6 lb. per ton of solution. A part of this combined cyanide is rendered available later on when the solution comes in contact with fresh ore and lime in the mill, but part of it, varying in amount according to circumstances, may be classed as a permanent loss.

For reactions in zinc precipitation see Chapter X.

3. *Mechanical Losses.*—Omitting losses of solution by leakage and runovers, there is always a loss of cyanide in the residue, both in the percolation and agitation processes, because it is not possible to give sufficient water wash at the finish to displace all the solution. The amount of such mechanical loss varies within wide limits and will depend upon the strength of the treatment solution, the degree of moisture discharged in the residues, and the method used for the final de-watering of the slime. For instance, in the case of slime a filter provides a nearer approach to displacement of the solution by water than dilution of the pulp followed by settlement and decantation, and therefore employs the water wash to better advantage.

The loss of cyanide in sand residue is usually inconsiderable, but in the treatment of slime it may often be from ¼ to ½ lb. per ton of ore, and has even been known to run as high as 1½ lb.

4. *Decomposition Loss.*—In the absence of protective alkali, acids present in the ore decompose the cyanide with formation of hydrocyanic acid. Heating of solutions tends to the formation of formates, HCO_2R and possibly acetates, CH_3CO_2R. Oxida-

tion by atmospheric air forms cyanate, KCNO, but Clennell states[1] that this reaction is slow in the case of solutions and that air or oxygen may be injected into a cyanide solution for a long time without any appreciable oxidation of cyanide taking place. On the other hand the writer in making some comparative tests between mechanical and Pachuca agitation found that the air agitation increased the consumption of cyanide by ½ to 1 lb. per ton of ore when working with solutions of 0.2% KCN. This, however, may have been due rather to the oxidizing action of the air on the finely divided particles of an ore rather sensitive to oxidation. When insufficient free alkali is present carbonic acid gas (CO_2) reacts with the cyanide thus:

$$KCN + CO_2 + H_2O = HCN + KHCO_3$$

The bicarbonates, $KHCO_3$ are no protection against CO_2 and it is for this reason that in the titration for protective alkali phenolphthalein is preferred to methyl orange since the bicarbonates are alkaline to the latter but acid to phenolphthalein.

There is also, especially in weak solution, a tendency to a decomposition known as hydrolysis:

$$KCN + H_2O = KOH + HCN$$

This is, however, a reversible reaction and in the presence of a sufficient excess of caustic alkali the tendency is minimized to a large extent.

Recently some interesting experiments have been made on the Rand by H. M. Leslie,[2] and the loss of cyanide in the form of HCN demonstrated by covering the tops of the tanks, drawing the gas off by suction, and passing it into a solution of caustic soda, thus recovering a large proportion of the cyanide lost in treatment. It is to be noted, however, that in these instances the protective alkalinity was low, that is, in the neighborhood of 0.01% CaO whereas when tried at a mill where the alkalinity was higher, that is, 0.025% CaO Mr. Leslie was able to detect only

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[1] The Cyanide Handbook, page 109 (Second Edition).

[2] *Journal Chem. Met. and Min. Soc. of S. A.*, Vol. 16, page 36.

traces of HCN gas being given off. H. A. White in criticizing the above mentioned article is inclined to attribute the loss of HCN rather to the action of the carbonic acid in the atmosphere and sees further losses by hydrolytic decompositions resulting in the formation of ammonium compounds such as ammonium formate and carbonate. Mr. White in an estimate of the sources of cyanide loss on the Rand, where the total consumption is about 0.4 lb KCN per ton of ore gives the following tabulation

Loss by dilution (*i.e.*, mechanical loss in residue)....	30 per cent.
Loss by cyanicides..................................	25 per cent.
Loss as HCN gas...................................	25 per cent.
Loss as NH_3 gas and N compounds................	20 per cent.

5. *Losses Due to Base Metals.*—All base metals in the ore that are soluble in cyanide solution of course involve consumption of cyanide, but the most common instance is probably copper. For the dissolution of metallic copper Julian and Smart[1] give the following equation: $2Cu + 4KCN + 2H_2O = K_2Cu_2(CN)_4 + 2KOH + H_2$, showing two parts by weight of cyanide consumed by one part of copper.

Clennell, however, gives an equation in which oxygen has a part and free H is not evolved,[2]
$Cu_2 + 8KCN + H_2O + O = 6KCN . Cu_2(CN)_2 + 2KOH$, with a consumption of four parts of cyanide to one of copper. For copper carbonate the same writer gives, $2CuCO_3 + 7KCN + 2KOH = Cu_2(CN)_2 . 4KCN + KCNO + 2K_2CO_3 + H_2O$, showing a consumption of three and a half parts of cyanide to one of copper.[3] The sulphides of copper are less readily acted on but these also will gradually go into solution, with the formation, probably, of cupric thiocyanate.

6. *Loss in the Form of Ferrocyanide.*—Finely divided metallic iron, such as may result from crushing and grinding operations, is slowly acted on by cyanide, as expressed in the equation

$$Fe + 6KCN + 2H_2O = K_4Fe(CN)_6 + 2KOH + H_2.$$

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[1] Cyaniding Gold and Silver Ores, page 112 (Second Edition).

[2] The Cyanide Handbook, page 80 (Second Edition).

[3] *Ibid.*, page 112.

Pyrite and marcasite also dissolve to some extent, with formation of ferrocyanide. Ferrous hydroxide is acted on rapidly,

$$Fe(OH)_2 + 2KCN = Fe(CN)_2 + 2KOH, \text{ and } Fe(CN)_2 + 4KCN = K_4Fe(CN)_6.$$

Insoluble basic ferric sulphate, $2Fe_2(SO_4)_3.Fe_2O_3$, commonly present in partly oxidized pyrite, is another source of formation of ferrocyanide. When ferrocyanides are found in any considerable amount in the solution after treatment of an ore it is often possible to effect a large saving in cyanide by attacking the trouble at its source. In ordinary milling practice metallic iron is not an important consumer of cyanide, as witnessed by the fact that in many mills only traces of ferrocyanide may be found in the cyanide solution, but should any considerable loss of cyanide be suspected from this cause, the remedy would lie in reducing as much as possible the iron surfaces used for grinding, more particularly by the use of flint pebbles in place of iron balls for regrinding and silex or El Oro liners instead of iron or steel. The effect of iron may often be seen to a very marked degree in experimental work. When grinding samples to a slime or to pass a fine mesh sieve on a small scale, the ore is often put into a small tube mill and ground until the whole of it will pass the desired mesh screen with the result that the sample is seriously overground, not only resulting in a product much finer, on the whole, than that aimed at, but containing an amount of infinitesimally fine iron far in excess of that which would be obtained in practice. If such a sample is weighed up wet and at once transferred to cyanide treatment, the cyanide consumption due to formation of ferrocyanide may run into several pounds per ton of ore. Precautions to minimize the formation of ferrocyanide from other sources are dealt with under the heading of "Cyaniding of Concentrate." (Chap. IX.)

7. *Loss in the Form of Sulphocyanate.*—In the treatment of silver ores this compound is almost invariably present, as seen from the equations,

$$Ag_2S + 4KCN = 2KAg(CN)_2 + K_2S$$
$$K_2S + KCN + O + H_2O = KCNS + 2KOH$$

where, although five molecules of KCN are present, only four are used to combine with the silver, the fifth being consumed by combination with sulphur. It might be thought that this fifth molecule of cyanide could be kept out of combination by the addition of a lead salt, and consequent removal of the sulphur as lead sulphide, but Caldecott[1] has found that PbS is acted on by cyanide with formation of KCNS, which probably accounts for the fact that the amount of KCNS usually present in a working solution does not seem to vary greatly whether lead be in use or not. Thus, when a lead compound is not used, one-fifth of the cyanide consumed usefully in dissolving silver sulphide is lost in this form. When lead is added during treatment there is a tendency for this loss to be less than one-fifth. Sulphocyanate is also formed by decomposition products of the base sulphides in the ore.

The Use of Lead Salts.—In the treatment of silver ores the addition of a lead compound is often a material aid to extraction. This is commonly explained by the reaction $K_2S + PbO + H_2O = PbS + 2KOH$ but since K_2S is rarely or never found in working solutions Clennell[2] suggests that the lead is rather to be considered as an aid in the attack on the silver, $Ag_2S + PbO + 4KCN = 2KAg(CN)_2 + PbS + K_2O$. Results obtained by various metallurgists in actual practice go to show that the efficacy of any specific lead compound is usually proportional to the lead content, and that therefore litharge is more efficient than lead acetate, weight for weight.

In regard to the amount of the lead compound used a critical point is usually found where the benefit is at a maximum and beyond which it gradually declines to zero and may even become a minus quantity.

Premature Precipitation.—It sometimes happens that gold and silver which have passed into solution are precipitated while still in contact with the ore and pass out of the plant with the residue. This may result in a serious lowering of recovery and

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[1] Mining and Scientific Press, May 2, 1908.

[2] The Cyanide Handbook, page 118 (Second Edition).

consequent loss of precious metal. The most common instance of this is probably the case of graphitic ores. The power of charcoal to precipitate gold out of cyanide solution has long been known, and it appears that graphite, and perhaps certain other organic substances, possess a similar property. W. R. Feldtmann,[1] after an extensive investigation, concludes that gold is precipitated on charcoal and graphite not as metal but as a compound, possibly a carbonyl aurocyanide, $AuCN.CO(CN)_2$, and that this compound is soluble to the extent of about 75% in solutions of alkaline sulphides. It has been found that the graphite in such ores is very amenable to the flotation process, so that a concentrate can be made consisting of the sulphide and nearly all the graphite. This concentrate, if gold is the only valuable constituent, is readily treated by cyanide after a dead roast. The flotation tailing, if sufficiently valuable, can then be cyanided with normal results. In cases where it is not advisable or convenient to roast a graphitic concentrate F. A. Beauchamp has found that the adverse action of the graphite may in some instances be to a large extent neutralized by giving the material a series of very short treatments, say for an hour or two hours, in cyanide solution, filtering and washing between each one.

This experience would tend to show that the precipitating action of the graphite is somewhat slow.

Relative Dissolving Efficiency of Sodium and Potassium Cyanide.—There is probably no potassium cyanide now made for commercial purposes. With the development of the cyanide process the custom grew up, for economic reasons, of manufacturing what was called a "double salt" composed of potassium and sodium cyanides in varying proportions. This material was compounded so as to have a cyanogen content of 98% to 99% in terms of KCN. On account of the fact, however, that sodium cyanide contains so much more cyanogen per unit of weight than the potassium salt, the mixture could contain a considerable percentage of impurities and still show the standard strength when titrated in terms of KCN, in fact it is probable that the

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[1] Proceedings I. M. M. (London), April, 1915.

strength was adjusted to the desired point by "diluting" the product with some presumably inert substance such as sodium carbonate or sodium chloride. It is this product that is here referred to as commercial potassium cyanide. Sodium cyanide on the other hand does not lend itself to adulteration in this way and a material that gives a high titration in cyanogen content may be taken at its face value as regards composition. The sodium cyanide now usually put on the market will test up to about 97% to 98% in NaCN (51.4% to 52% in cyanogen content out of a possible maximum of 53%) and is therefore a very high grade product.

The question then arises, is the relative efficiency of the two kinds of cyanide strictly proportional to their cyanogen content per unit of weight? This is usually assumed to be the case in practice and no doubt it is so in many instances. Clennell[1] states that he has always found that "solutions containing equal amounts of cyanogen give identical results, whatever the nature of the metal with which it is combined." This, however, is not the point. The real question is, do the two kinds of dry cyanide salts when added to a mixture of water and ground ore undergoing extraction treatment produce a solution with a cyanogen content proportional to that shown by titration to be contained in the dry salts respectively? And it would seem that in some cases at least this question must be answered in the negative.

At the mill of the Creston Colorada Company in 1907 the writer made a comparison between the use of the double salt (98 to 99% in terms of KCN) and sodium cyanide (130% in terms of KCN). Each kind of cyanide was used exclusively every alternate month, the test extending over eight or nine months. The result showed that the weight of cyanide charged out of the store per ton of ore was the same, pound for pound, regardless of which kind was used, the cyanide strength of the stock solutions being maintained uniformly at 0.1% free cyanide in terms of KCN during the whole test. This meant that a pound of sodium

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[1] Mining and Scientific Press, March 28, 1914.

cyanide, paid for at the rate of 130% KCN, did no more work than a pound of the double salt paid for at the 99% rate.

Again, at the Butters Divisadero Company's mill in 1914 the writer made a careful survey of the figures for the previous seven years. During all this time the treatment conditions had been uniform and also the character and grade of ore, so far as could be observed. Sodium cyanide (130% KCN) was in use from May, 1906 to March, 1908, and again from Jan., 1912 to December, 1913. "Cyanide" (99% KCN) was in use from April, 1908 to December, 1911. The solution strength maintained throughout was 0.1% free cyanide in terms of KCN by the usual titration methods. The number of pounds of each kind of cyanide charged out of the store was then worked out per ton of ore over the several periods and also the combined extraction in gold and silver. The resulting figures were,

Sodium cyanide (130% KCN)		"Cyanide" (99% KCN)	
Pounds per ton of ore	Extraction, per cent.	Pounds per ton of ore	Extraction, per cent.
2.64	88.6	2.17	92.0

The result of this comparison is even more remarkable than that at the Creston Colorada since not only was the number of pounds of 99% "cyanide" consumed per ton less than that of sodium cyanide (130%) but the extraction obtained during the use of the 99% cyanide was over 3% higher.

The writer has no explanation of this phenomenon to offer, and it seems all the more difficult to account for since there is little doubt that the "Cyanide" (99%) in question was the double salt and contained a considerable proportion of sodium cyanide as well as some presumably inert substance such as sodium chloride (though as a matter of fact no analyses are available). Whatever be the reason the conclusion would appear to be that at least in the case of some ores the "Cyanide" (double salt, 99% KCN) is more economical in use than sodium cyanide (130% KCN) and that it would be worth while investigating the question in each individual case.

In connection with the apparent increase in extraction at Divisadero due to the use of the double salt the question suggests itself whether the presence of a certain proportion of the potassium base may have the effect of making the solution more active for dissolving the precious metals. R. C. Kline[1] states that he has noticed a tendency to an increased extraction when potassium salts are present in the solution.

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[1] Mining and Scientific Press, March 18, 1911.

CHAPTER II

TESTING AND ANALYSIS OF SOLUTIONS

It is obviously of the highest importance in controlling the action of a solution to find out what strength of cyanide and alkali it contains at any given time. Under this head are the tests for free cyanide, "total" cyanide, hydrocyanic acid, and free or "protective" alkali.

Besides these constituents, which have to be determined frequently each day, it is useful at times to ascertain the presence, (and if present the amount), of the following substances, zinc, copper, ferrocyanide, and sulphocyanate.

It is customary to express cyanide strength in terms of potassium cyanide (KCN) of 100% strength, even though sodium cyanide (NaCN), carrying a higher percentage of cyanogen (CN) per unit of weight, may be actually in use.

This seems somewhat illogical and the only thing to be said in its defence is that the theory and practice of cyanidation having been born and having grown up under the formula KCN a change of notation would mean readjustment of all the old standards and, for a time at least, a mental translation of the new notation to the old before it would convey any meaning. In view of the varying cyanogen content per unit of weight in the commercial cyanide now put on the market the obvious standard to use would be the percentage of CN but if in the future the pure sodium cyanide (98 to 99% NaCN) is to be universally supplied it would probably be more convenient to think and speak in terms of that salt and the titration of the solution could be reported in pounds NaCN per ton of solution. If a change is to be made, however, it would seem that it would only come about by concerted action between the various technical societies throughout the world.

The tests for free cyanide, total cyanide and hydrocyanic acid are made volumetrically, by slowly running silver nitrate solution from a burette into a known quantity of the cyanide solution. For this purpose a standard solution of silver nitrate is usually made up, of such a strength that the result may be read off without calculation.

Such a standard solution of silver nitrate may be made by dissolving 3.262 grams of silver nitrate ($AgNO_3$) in distilled water and making up to 1 litre and this is the standard adopted and referred to in the following pages. (Should the least milkiness develop while dissolving it is an indication that the water is not sufficiently pure). The reaction taking place between the silver nitrate and potassium cyanide may be represented by the following equation:

$$AgNO_3 + 2KCN = AgK(CN)_2 + KNO_3$$

Thus 169.9 gm. $AgNO_3$ saturates 130.2 gm. KCN
or 3.262 gm. $AgNO_3$ saturates 2.5 gm. KCN

Therefore 1 cc of above solution (=0.003262 gm. $AgNO_3$) saturates 0.0025 gm. KCN. If 25 cc of mill solution is taken for titration, then 1 cc of nitrate solution = 0.0025 gm. KCN = 0.01% or 0.1 kilo KCN per metric ton or 0.2 lb. KCN per 2000 lb. ton of solution. Similarly, a solution containing 6.525 grams of silver nitrate per litre may be made up, 1 cc of which contains 0.006525 grams of $AgNO_3$ equal to 0.005 gm. of KCN. Then if 10 cc of mill solution be taken for titration, 1 cc of nitrate solution = 0.005 gm. of KCN = 0.05% = 0.5 kilo per metric ton or 1 pound per 2000 lb. ton of solution; while if 50 cc of mill solution be taken, 1 cc of nitrate solution = 0.01% = 0.1 kilo per metric ton or 0.2 lb. per 2000 lb. ton of solution. Probably the most common strength for the standard silver nitrate solution is 13.05 grams per litre, and is such that if 10 cc of mill solution be taken for titration 1 cc of nitrate solution equals 0.1% KCN or 2 lb. per ton. This strength, however, is not recommended for reasons which will be stated later.

Free Cyanide Determination.—Two methods are in use for this purpose, and there is a third that may be used on occasion, but it should be understood that there is no direct method yet devised which will give better than comparatively correct results.

If the determination, however, can be relied on to be relatively uniform and consistent, that is all that is required for practical working.

Method No. 1.—Take 25 cc of the solution to be tested and place in an Erlenmeyer flask, add a few drops of 5% solution of potassium iodide free from alkali, and slowly run in the silver nitrate from the burette, shaking the flask meanwhile, until a distinct yellow opalescence appears. Then read off the number of cc of silver nitrate used and multiply by 0.01, and the result will be the amount of free cyanide in the solution in terms of percentage. To get the cyanide in kilos per metric ton of solution multiply the number of cc of silver nitrate by 0.1 or if the cyanide in terms of lb. per 2000-lb. ton of solution is desired multiply the number of cc of silver nitrate by 0.2. If the cyanide strength is over 0.2% KCN it is better to take only 10 cc of the cyanide solution for the test. In this case, when using as the standard solution 3.262 grams $AgNO_3$ per litre, after the burette reading has been noted move the decimal point one place to the left and divide by 4. For instance if B = burette reading in cc then $\frac{B\times 0.1}{4} = \%$ KCN or $\frac{B}{4}$ = kilos per metric ton or $\frac{B\times 2}{4} = \frac{B}{2}$ = lb. KCN per 2000-lb. ton of solution.

Method No. 1 is not to be recommended when the solution contains zinc (as it usually does), because the results are not uniform. When there is no free alkali in the cyanide solution the value obtained for free cyanide may approximate the correct one, but if any free alkali is present (and it almost always is, in a working solution), the cyanide reading will be in excess of the correct figure, and this excess error will be greater in proportion as the amount of free alkali is greater, until, if the latter happens to be about 0.08% (in terms of NaOH) or over, the reading supposed to represent free cyanide will coincide exactly with the amount of "total" cyanide, as shown in the method to be described later.

✓ *Method No.* 2.—In place of the foregoing the following is recommended, as giving the most correct and consistent reading for free cyanide under ordinary conditions. It needs a certain amount of practice to stop at the exact end-point of the reaction, but precision is easily and rapidly acquired.

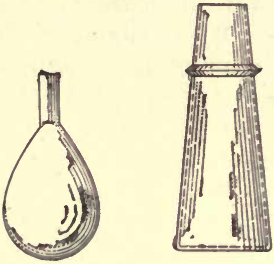
Fig. 1. Fig. 2.
Parting Flasks.

Take 25 cc of the solution to be tested. (If not absolutely clear and colorless to transmitted light, filter and re-filter until it is.) Pour into a bulb parting-flask with the neck cut short, Fig. 1, or into a small conical parting-flask, Fig. 2, first having seen that the flask is as clean and transparent as the solution, for which purpose wash with some weak hydrochloric acid solution and rinse well. Do not add water to the solution to be tested, because it will tend to dissociate some of the zinc-potassium cyanide and make the free cyanide reading too high. Place the burette opposite a window with a good light (but not in the direct rays of the sun), and fix a black card vertically about 4 inches behind the spot where the flask is to be held, so that the light will strike the surface of the solution over the top of the card leaving the solution itself in shadow. Then, keeping the eye on a level with the flask, watch the behavior of every addition of silver nitrate as it touches the solution, not keeping it in continuous agitation but allowing the bluish-white cloud formed by each addition of silver to hang for a second or two before shaking and dissolving. The finish is indicated by the first appearance of a bluish haze dulling the original brilliancy of the solution. With a very little practice the operator will find he can check himself and others within a tenth of a cc on the burette. To ensure the success and delicacy of this reaction it is important that the silver nitrate solution should be dilute, and for this reason the standard already recommended (3.262 grm. $AgNO_3$ per litre) is to be preferred to the usual standard of 13.05 grm. per litre. With a strong silver nitrate solution the tendency is to precipitate flocks instead of the dispersed opalescence necessary for a clearly defined end point.

Number of cc of standard $AgNO_3 \times 0.01$ = % free KCN or $\times$ 0.1 = kilos KCN per metric ton of solution or $\times$ 0.2 = pounds KCN per ton (2000 lb.) of solution.

(NOTE).—It sometimes happens (that is, in the case of certain solutions) that the addition of a few drops of potassium iodide solution before titration will render this first haziness more easily distinguished and may give a slightly lower reading than without ~~The reason of this phenomenon is not clear and the writer is not prepared to offer any explanation of it~~. In such cases, however, there is no harm in making use of it so long as it is clearly understood that it is the first faint blue-white opalescence that is to be looked for and not the yellow tint.

Walter Virgoe[1] has pointed out that cyanide solutions containing copper, when titrated with silver nitrate without potassium iodide indicator, yield a result higher than the true value for free cyanide, owing to part of the copper-potassium cyanide being recorded as free cyanide. Thus, while in presence of zinc and free alkali the addition of neutral potassium iodide as indicator gives a higher and less correct reading than is recorded without the indicator, when copper is present without zinc the opposite is the case and the addition of neutral iodide indicator gives a lower reading than without, this lower reading being the correct one. So, when zinc is absent and copper present, method No. 1 should be used.

Method No. 3.—When copper and zinc are present together both tendencies are displayed in proportion to the relative amounts of each metal, and the result is thereby complicated. In this case, it is advisable to make titrations both with and without KI indicator and to base solution control on the lower of the two readings; or a method devised by Clennell[2] for determining free cyanide may be tried, though the writer has not always found it to give concordant results.

First determine protective alkali, by Green's method, carefully running in the acid drop by drop and recording the number

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[1] Proceedings I. M. M. (London), Vol. X, page 102.

[2] Chemistry of Cyanide Solutions, page 25 (Second Edition).

of drops needed. (For this purpose and for the subsequent neutralization a mineral acid should be used, say decinormal nitric acid, because oxalic would precipitate the lime in the cyanide solution as oxalate, which would mask the end-point in the test for cyanide.) Then measure out another 25 cc of solution to be tested and run into it the same number of drops of decinormal acid as was previously recorded. The whole of the protective alkali in the cyanide solution is thus exactly neutralized so that it does not militate against the use of the iodide indicator. To the test solution so neutralized add at least 2 cc of neutral 5% solution of potassium iodide, and titrate with standard silver nitrate to first white flocculent precipitate. Calculate free cyanide as already explained.

The writer has found this white end point rather indeterminate, but the yellow end point is well marked and the readings uniform though the latter gives a higher and probably less correct result than the white end point.

In the use of this method the quantity of KI added has an important bearing on the result of the titration. The writer has found that smaller quantities than that stated give proportionally higher cyanide readings, though the amount of iodide may be materially increased without much apparent effect on the result.

Total Cyanide Determination.—The term "total" cyanide is generally limited to mean zinc-potassium cyanide + free potassium cyanide + hydrocyanic acid. To determine this, make up an indicator consisting of 4 grams caustic soda and 1 gram potassium iodide in 100 cc of distilled water. Take 25 cc of the solution to be tested, add 3 to 5 cc of the indicator, and titrate with standard silver nitrate until a distinct lemon yellow opalescence appears.

Number of cc $AgNO_3$ used $\times 0.01 = \%$ "total" cyanide. Keep stock bottle of indicator well stoppered to avoid absorption of carbonic acid, which will tend to make the test cloudy with carbonate of lime. Should a whitish precipitate of zinc dense enough to mask the color come down before the yellow tint appears Clennell recommends the addition of a little ammonia before beginning the titration, but this is usually unnecessary.

Hydrocyanic Acid Determination.—This of course need only be looked for in solutions containing no protective alkalinity.

First determine free cyanide. Then take another portion, add a little solution of potassium or sodium bicarbonate free from monocarbonate, and titrate again for free cyanide. (Free carbonic acid must be absent.) The difference in the reading between the two titrations gives the equivalent in KCN of the hydrocyanic acid present (Bettel).[1]

Method for Detecting Traces of Cyanide in Water.[2]—Evaporate 500 cc of the solution with 3 or 4 drops of ammonium sulphide. Bring to dryness on the water bath and take up with a small quantity of water or water and alcohol. Filter, and add a drop of ferric chloride solution. A red coloration indicates the presence of cyanide. By careful evaporation 1 part in 100,000 can be detected. The author states that by evaporating solutions containing known quantities of cyanide in this manner he has found it possible to obtain a rough colorimetric estimation of the amount of cyanide present.

TABLE 1.—SHOWS VARIOUS STANDARD STRENGTHS OF SILVER NITRATE AND VARYING FACTORS

Number of cc. of cyanide solution taken	Standard nitrate solutions, grams $AgNO_3$ per litre	Equivalent of 1 cc. of silver nitrate solution		
		KCN, per cent.	Kilos KCN per metric ton	Lb. KCN per 2000 lb. ton
50	3.262	0.005	0.05	0.1
25	3.262	0.010	0.10	0.2
10	3.262	0.025	0.25	0.5
50	6.525	0.010	0.10	0.2
25	6.525	0.020	0.20	0.4
10	6.525	0.050	0.50	1.0
50	13.050	0.020	0.20	0.4
25	13.050	0.040	0.40	0.8
10	13.050	0.100	1.00	2.0

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[1] *Proceedings* Chem., Met. and Min. Soc. of South America, Vol. I, page 165.

[2] Gerard W. Williams, *Journal Chem., Metall. and Min. Soc. of S. A.*, May, 1904.

Protective Alkali Determination.—This term is usually defined as "the alkaline hydrates and half the monocarbonates," whose action is to protect the cyanide from decomposition, by acids developed in the ore and by atmospheric carbon dioxide.

Either oxalic acid or a mineral acid may be used as a standard solution. Objections have been raised to the use of oxalic on the ground that it is unstable in weak solutions. This is no doubt true where standard solutions are to be kept in stock for an indefinite time, and where extreme accuracy is required, but it has been proved by the writer that decinormal oxalic acid may be kept in a bottle ¾ full of air in a tropical climate for at least 4 weeks without showing any appreciable deterioration when checked against carefully standardized decinormal alkali. The advantage of oxalic is that if a good brand be used and the bottle of crystals be kept well stoppered, it may be weighed up, dissolved, and used without standardization. Of course, theoretical accuracy is not attained without standardization on account of uncertain hydration of the crystals, but with the precautions mentioned such a solution is perfectly adequate for all purposes of alkali control in plant solutions.

The decinormal solution is made by dissolving 6.3 grams of the oxalic acid crystals in distilled water and making up to a litre. Decinormal solution of nitric or some other mineral acid may be used if preferred, but it is necessary that this should be standardized against a standard alkali. For this purpose Sutton recommends sodium carbonate as being the most useful. To make the decinormal solution take 10 or 12 grams of pure sodium carbonate, ignite gently, and cool under the desiccator. Then quickly weigh up 5.3 grams of this and dissolve in hot distilled water. When cool put into a litre flask and fill up to the mark with distilled water.

A decinormal nitric acid solution contains 6.3 grams of HNO_3 per litre. If the acid is about 1.4 sp. gr. put 6.5 cc of colorless acid into a litre flask and fill up to the mark with distilled water. A given volume of the decinormal soda solution is taken (say 50 cc), and a few drops of methyl orange (not phenolphthalein) indicator added. The nitric acid solution is then run in from a burette until the color of the indicator shows that the solution is neutral. If the nitric acid solution is decinormal it should take exactly 50 cc of it to neutralize the soda. If it needs more or less than 50 cc it is incorrect, and the strength may be rectified by adding water or acid, as the case may be, after calculating the amount necessary. It should then be verified by another titration.

As an indicator for determination of protective alkali phenolphthalein is generally used, as it gives a value for monocarbonates corresponding with the definition of protective alkali already mentioned. Moreover, methyl orange cannot be used as indicator with oxalic acid. Dissolve about 0.5 gram of phenolphthalein in a little alcohol and add water up to 100 cc. Should a milkiness form on adding water, more alcohol is needed.

Two methods are in general use for determining protective alkali, (1) Clennell's, and (2) Green's.

Clennell's Method for Protective Alkali.[1]—This is carried out on the same portion of solution that was used for the free cyanide test, method No. 2. After the necessary amount of silver nitrate has been run in to determine free cyanide[2] a few drops of the phenolphthalein indicator are added, giving a rose pink color to the solution if free alkali is present; it is then titrated with the decinormal acid until the pink color is dispelled. Taking 25 cc of original cyanide solution, the number of cc of $\frac{N}{10}$ acid used $\times$ $0.016 = \%$ free alkali in terms of NaOH, or $\times\ 0.0112 = \%$ in terms of CaO. If 10 cc of original cyanide solution was taken, multiply the number of cc of $\frac{N}{10}$ acid used by 0.04 to get percentage of alkalinity in terms of NaOH or by 0.028 to get percentage of alkalinity in terms of CaO.

$$H_2C_2O_4 . 2H_2O + 2NaOH = Na_2C_2O_4 + 4H_2O$$

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[1] Chemistry of Cyanide Solutions, page 63 (Second Edition).

[2] Clennell's statement that "addition of even a considerable quantity of silver in excess of the necessary amount does not materially affect the result" is not true of solutions containing zinc.

Atomic weights: 126 oxalic = 80 sodium hydroxide.

Decinormal oxalic acid = 6.3 grams per litre, equivalent to 4.0 grams of NaOH. 1 cc $\frac{N}{10}$ acid = 0.0063 gm. = 0.004 gm. NaOH.

If 25 cc of mill solution be taken for the test then 1 cc $\frac{N}{10}$ acid = 0.004 gm. NaOH = 0.016%, therefore No. of cc $\frac{N}{10}$ acid × 0.016 = % NaOH.

Special standard acid is frequently used in preference to $\frac{N}{10}$.

(*a*) *To Read in Terms of NaOH.*—If 3.937 grams of oxalic acid crystals be dissolved and made up to 1 litre with water, then, taking 25 cc of original cyanide solution the number of cc of standard acid used × 0.01 = % free alkali in terms of NaOH, or × 0.1 its equivalent in kilos of NaOH per metric ton or × 0.2 = its equivalent in lb. of NaOH per ton of 2000 lb.

If 10 cc of original cyanide solution be taken, the special solution is made by dissolving 1.575 gm. of oxalic acid in water and making up to a litre In this case 1 cc of acid = 0.01% NaOH.

(*b*) *To Read in Terms of CaO.*—If 5.625 grams of oxalic acid per litre be used, then the number of cc × 0.01 = % of free alkali in terms of CaO, or × 0.1 = its equivalent in kilos per metric ton or × 0.2 = its equivalent in lb. of CaO per ton of 2000 lb.

If 10 cc of original cyanide solution be taken the special solution is made by dissolving 2.25 gm. oxalic acid per litre, 1 cc of which = 0.01% CaO.

This method is perfectly satisfactory in solutions containing zinc, provided that the preliminary titration for free cyanide given under method No. 2 has been carefully performed and addition of silver has been stopped before more than the merest trace of zinc has been precipitated. Green has shown that as soon as sufficient silver has been added to throw down a precipitate of zinc there is a formation of acid zinc nitrate which destroys the reliability of the test for alkali.

Green's Method for Protective Alkali.[1]—In order to avoid the risk just mentioned Green devised the following method.

First determine total cyanide, as described, recording the number of cc of silver nitrate used. Take a new 25 cc of the solution to be tested and add an excess of potassium ferrocyanide (usually 5 to 10 drops of 10% solution will suffice), then run in from the burette the number of cc of silver nitrate recorded in the preliminary test, and a few drops over. Then add phenolphthalein indicator, and titrate with standard or decinormal acid. Calculate protective alkali as in Clennell's method.

TABLE 2.—SHOWS VARIOUS STANDARD STRENGTHS OF OXALIC ACID SOLUTION, AND VARYING FACTORS

Number of cc of cyanide solution taken	Standard oxalic acid solution, grams of $H_2C_2O_4.2H_2O$ per litre	Equivalent of 1 cc oxalic acid solution					
		Per cent.		Kilos per metric ton		Pounds per 2000 lb. ton	
		NaOH	CaO	NaOH	CaO	NaOH	CaO
50	6.3 gr. = $\frac{N}{10}$	0.008	0.0056	0.08	0.056	0.16	0.112
25	6.3 gr. = $\frac{N}{10}$	0.016	0.0112	0.16	0.112	0.32	0.224
10	6.3 gr. = $\frac{N}{10}$	0.040	0.0280	0.40	0.280	0.80	0.560
50	7.875 gr.	0.010	......	0.10	.....	0.20	
25	3.937 gr.	0.010	......	0.10	.....	0.20	
10	1.575 gr.	0.010	......	0.10	.....	0.20	
50	11.250 gr.	.....	0.01	....	0.1	....	0.2
25	5.625 gr.	.....	0.01..	....	0.1	....	0.2
10	2.250 gr.	.....	0.01..	.....	0.1	....	0.2

Available Alkalinity of Lime.[2]—The sample as delivered for analysis is contained in an air-tight vessel, having been passed

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[1] *Proceedings* I. M. M. (London), Oct., 1901.

[2] Abstract from report of South African Engineering Standards' Committee. *Mining and Scientific Press*, May 3, 1913.

through a 30-mesh sieve. It is crushed with a Wedgewood mortar and pestle, and the whole passed through a 60-mesh sieve, this operation being performed as quickly as possible. It is then placed in a clean, dry, wide-mouthed bottle fitted with a tight-fitting dry glass-stopper so as to prevent any access of air. Two grams of the sample is carefully weighed out and agitated with 1 litre of a 2 per cent. cane-sugar solution (or 1 gm. with ½ litre of a 2 per cent. sugar solution). If a shaking machine be available, 2 hours continuous agitation should be given, if not, 6 hours intermittent agitation, every care being taken to prevent coagulation of the lime, in order that the lime and solution may be brought into the most intimate contact during this period. When the agitation is finished the solution is filtered as quickly as possible, and aliquot portions titrated with N/10 or N/5 acid, using rosolic acid as indicator, avoiding delay so as to obviate undue exposure to the atmosphere. The distilled water used in the above determination must be made neutral to rosolic acid to counteract the presence of dissolved CO_2.

Zinc Determination.—To ascertain the presence of zinc add a few drops of sodium sulphide solution to 50 cc of the solution to be tested (which should be strongly alkaline), and heat to boiling point. If zinc is present it will come down as a flocculent white precipitate. It sometimes happens that a cyanide solution will contain an appreciable quantity of alumina, in which case a precipitate of aluminium hydroxide may be formed on adding sodium sulphide. This precipitate, however, is gelatinous and can be easily distinguished from the flocculent precipitate of zinc sulphide.

If silver is present also it should first be removed by shaking up some of the solution with a little aluminium dust and filtering, as the black precipitate of silver will mask the white sulphide of zinc.

To determine the amount of zinc, the method recommended by Gerard Williams[1] is probably the most generally useful. Silver

[1] *Journal Chem., Metal. and Min. Soc. of S. A.*, May, 1904.

and lead, if present, interfere and should be removed by precipitation with aluminium before proceeding with the test. Lime, also, has in some cases appeared to impair the accuracy of results, and is better removed.

The reagents needed are aluminium dust, commercially known as aluminium bronze powder, crystals of neutral potassium or sodium oxalate, sodium sulphide solution, a decinormal solution of iodine and a decinormal solution of sodium thiosulphate ($Na_2S_2O_3.5H_2O$).

The thiosulphate solution is made by dissolving 24.827 grams of the salt in distilled water and making up to 1 litre. If the crystals are chemically pure and have been kept well stoppered so that they are neither damp nor coated with white powder (indicating dehydration), this solution may be taken as standard for the purposes of this test, though for accurate chemical research work it would be necessary to standardize it against a strictly decinormal iodine solution, or by other methods given in works on volumetric analysis.

A decinormal solution of iodine contains 12.7 grams of the element per litre. To expedite the dissolution of the iodine Clennell[1] recommends the following procedure: weigh out 18 or 20 grams of potassium iodide and dissolve about 15 grams in 20 or 25 cc of distilled water, reserving the remainder of the iodide crystals until the finish. Then weigh up 12.7 grams of c.p. iodine and transfer quickly to the beaker containing the iodide solution. Stir with a glass rod until no more will dissolve, dilute with distilled water, and decant carefully into a litre flask. Add the remainder of the iodide to the beaker with a little water, and the rest of the iodine should soon dissolve; if not, a gram or two more of the iodide may be added. When all the iodine solution has been transferred to the litre flask, fill up to the mark with distilled water, and transfer the whole to a stoppered bottle, which should be kept in a dark cupboard.

To check this solution against the decinormal thiosulphate take 25 cc in a small beaker and run in the thiosulphate solution

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[1] Private communication.

from a burette until the brown color is almost dispelled. Then add a few drops of a solution of boiled starch as indicator, and continue the titration until the blue color disappears. Should it need more or less than 25 cc of thiosulphate to produce this result, the difference should be noted on the label of the iodine bottle and the proper correction made for it when using the iodine solution for subsequent titrations If it appears from the look of the thiosulphate crystals that they are of doubtful hydration, it will be better to take the iodine solution as the standard and make the necessary correction on the thiosulphate solution. For instance, if the 25 cc of thiosulphate solution was neutralized by 24 cc of iodine solution, 1 cc thiosulphate = 0.96 cc iodine. Then if the thiosulphate be considered standard and if in an analytical titration 13 cc of iodine is needed, the correction for the iodine would be 0.96 : 1 :: 13 : 13.54, indicating that 13.54 cc of strictly $\frac{N}{10}$ iodine had been used. If on the other hand the iodine be considered standard, then

24 cc of iodine = 25 cc thiosulphate
and 1 cc of iodine = 1.04 cc thiosulphate

so if in an analytical titration 13 cc of thiosulphate is needed, the correction for the thiosulphate would be 1.04 : 1 :: 13 : 12.5, indicating that 12.5 cc of strictly $\frac{N}{10}$ thiosulphate had been used.

To determine zinc take about 150 cc of the cyanide solution to be tested and dissolve in it a few crystals of potassium oxalate. This will precipitate the lime as an oxalate and substitute caustic potash in the solution.

Should silver and lead be absent the zinc determination may be proceeded with after filtering out the lime, but if either of them is present, it should be first removed. To effect this, take the filtrate after lime precipitation and place in a small stoppered bottle, add 40 or 50 mgm. of aluminium powder and shake briskly for 4 or 5 minutes. A drop of the liquor may then be withdrawn on a glass rod and brought in contact with a piece of filter paper moistened with sodium sulphide solution. Should any stain result the precipitation is not complete and agitation must be continued, with addition of more aluminium powder if necessary.

Since caustic soda is needed in order to precipitate metals in a cyanide solution by means of aluminium the solution must contain from 0.05% to 0.1% of free alkali; if it carries much less than this, a tiny splinter of caustic soda may be added.

When precipitation of silver and lead is complete the solution is filtered, and the filtrate is then ready for the zinc determination.

Measure out 100 cc of this filtrate, make strongly alkaline with caustic soda, add excess of sodium sulphide solution, and heat almost to boiling point. By this time all the zinc should have come down as zinc sulphide, which is filtered out, and well washed with hot water, the washing of precipitate and filter paper being continued until the water coming away shows no stain when tested with lead acetate paper or solution, indicating that all soluble sulphide has been eliminated. The filter paper with precipitate is then transferred to a small stoppered flask and agitated violently with a little distilled water to disintegrate the paper. An amount of decinormal iodine solution is then run in from a burette, agitating the flask meanwhile, until an excess is present, as indicated by a permanent dark brown color. This excess is then titrated back with decinormal thiosulphate.

The original number of cc of iodine solution run in less the excess indicated by thiosulphate represents the quantity used up in decomposing the zinc sulphide.

$$1 \text{ cc of } \frac{N}{10} \text{ iodine} = 0.00327 \text{ gram of zinc.}$$

The filter paper pulp usually acts as an indicator in this case, rendering the use of starch unnecessary, but the latter indicator may be used in the ordinary way if desired.

Copper Determination.—The first essential is completely to decompose all the cyanogen compounds. For this purpose Clennell[1] recommends evaporating 100 cc of the solution to be tested with 5 cc of nitric and 10 cc of sulphuric acid until dense white

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[1] Chemistry of Cyanide Solutions, page 134 (Second Edition).

fumes are freely given off. Sharwood[1] prefers to evaporate 100 cc of the solution to dryness twice with nitric acid and then dissolve the residue in dilute sulphuric. If in either case a yellowish flocculent precipitate remains undissolved an incomplete decomposition of cyanogen compounds should be suspected. In some instances the writer has found it necessary, after evaporating to dryness twice with nitric acid, to add strong sulphuric and evaporate to white fumes in order to complete the decomposition of the cyanides.

When a large quantity of solution is taken for evaporation the lime separates out as calcium sulphate at the finish so it is better in such cases before starting the analysis to precipitate the lime with sodium carbonate or potassium oxalate and filter it out. When decomposition is complete, whatever method be used, the assay is diluted with water and boiled until all anhydrous salts are dissolved. Sometimes a slight insoluble residue is left which may be filtered out, washing the filter paper with hot water and adding the washings to the filtrate. Next place in the solution a cylinder of thin sheet aluminium and boil until all the copper is precipitated. If hydrogen is not freely given off add more sulphuric acid. When the action is sufficiently brisk the copper deposited on the aluminium will be detached and form spongy masses, leaving the aluminium surface clean. When the copper is precipitated remove the aluminium cylinder and wash it with distilled water. Should any copper adhere to the surface it may be detached with a rubber-tipped glass rod and washed into the beaker. Decant the solution carefully and wash the spongy copper several times by decantation with hot water, avoiding exposure to the air so as to minimize the risk of oxidation.

The copper is now dissolved in 5 to 10 cc of nitric acid, and boiled until all brown fumes are expelled, and the determination made by any of the standard methods, such as electrolysis, titration with cyanide, or the potassium iodide method.

The Cyanide Method for Copper.—Unless a high degree of accuracy is required the cyanide method may be made to give quite satisfactory results.

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[1] Engineering and Mining Journal, 1898, page 216.

For this test a standard copper solution is prepared by dissolving 10 grams of pure copper foil in 100 cc of nitric acid, boiling to expel brown fumes, and diluting to 1 litre, 1 cc of this solution will contain 0.01 gram of copper and 0.1 cc of nitric acid.

If silver was present in the original cyanide solution some or all of it will have passed into the copper nitrate solution, and should be removed by adding a drop or two of hydrochloric acid, filtering, and washing the precipitate. The solution is then neutralized with sodium carbonate in slight excess, and about 1 cc of ammonia added, or sufficient to give a clear blue solution, which is put in a beaker of suitable size. Next, put in a similar beaker a measured amount of the standard copper solution, approximating in copper content the quantity of copper contained in the solution to be tested, and accompanied by about the same amount of nitric acid. Add a drop or two of hydrochloric acid (if that was added for precipitating silver, as described), and neutralize with sodium carbonate. Then add 1 cc of ammonia, or the same quantity as was added to the other beaker, and distilled water to make an equal volume in the two beakers.

The standard and the assay (which should both be at the same temperature), are then titrated with a solution of potassium or sodium cyanide of 0.5 to 2% strength according to circumstances, until only a faint violet tint remains.

The burette reading obtained in testing the standard copper indicates the copper value of each cc of cyanide solution under the conditions of the test, and from this the copper content of the assay is calculated.

Colorimetric Estimation of Copper.—Where copper is present in very small amounts it is usually best estimated colorimetrically. For this purpose the standard copper solution already described may be diluted to $\frac{1}{10}$ its strength so that 1 cc will contain 0.001 gram of copper instead of 0.01 gram.

The cyanide solution is evaporated with acids as for the regular copper determination and after diluting, boiling, and cooling, a few cc of ammonia is added or sufficient to neutralize the acid and produce the characteristic blue color. When small quantities of iron are present it may be necessary to heat the resulting solution to assist the precipitation of the ferric hydroxide which should be then filtered out, but it is better to avoid heating if possible after addition of the ammonia as this sometimes tends to cause reduction of intensity in the blue color especially where only traces of copper are present. Even in the cold the blue color tends to fade on standing so no time should be lost in proceeding with the test.

Method of Procedure.—Two colorless glass cylinders marked at 100 cc are taken. Into one is poured the blue ammonia-copper solution to be tested, and distilled water added up to the mark. Into the other cylinder is poured a few cc of ammonia and distilled water up to within a short distance of the mark, space being left for the addition of the standard copper solution. The latter is then run into the cylinder drop by drop until the color matches that of the solution in the first cylinder. The number of cc of standard copper solution run in is then read off the burette and the copper content of the test calculated therefrom. In matching the color it is of great assistance to have the cylinders enclosed in a box open at the bottom, in such a way that the light may be reflected upward from a sheet of white paper and the color judged and compared by looking down vertically through the column of liquid.

Ferrocyanide Determination.—The most reliable method is probably that of determining the iron and calculating to ferrocyanide.

It must of course be assumed that all the iron found occurs in that form, but as ferricyanides are most unlikely to be present in ordinary circumstances the assumption is a fairly safe one.

As in the case of the copper determination (see page 33) especial care must be taken to decompose the cyanogen compounds, either by evaporating with nitric and sulphuric acids till white fumes are given off or by evaporating twice with nitric acid to dryness and then with strong sulphuric to white fumes; and as before, if dealing with a large volume of original solution it is better to begin by removing the lime with sodium carbonate or potassium oxalate. The solution resulting from the evaporation is cooled, diluted and boiled to dissolve anhydrous salts. It is then cooled again, neutralized with ammonia in slight excess, and heated to boiling. The ferric hydroxide formed is filtered out and washed on the filter.

Potassium Sulphocyanate Method for Estimating the Iron.—If the quantity of iron is very small it is best estimated colorimetrically by at once dissolving the precipitate in hydrochloric acid and carefully washing the filter paper. A standard solution of ferric iron is prepared by dissolving 0.1 gram of C.P. iron wire in nitrohydrochloric acid. Dilute the solution thus obtained and precipitate the iron with ammonia. Filter out the ferric hydroxide and wash thoroughly and then re-dissolve in dilute hydrochloric acid and add distilled water to make up to 1 litre. The test is made similarly to that described for the colorimetric estimation of copper. Two colorless glass cylinders marked at 100 cc are taken. In one of them is placed 1 or 2 cc of hydrochloric acid and 10 or 15 cc of a solution of potassium or sodium sulphocyanate of such a strength as to have an excess of the latter after running in the standard iron. Fill up nearly to the mark with distilled water leaving space for addition of the standard iron. Into the other cylinder pour a similar quantity of hydrochloric acid and sulphocyanate solution and a measured amount of the solution to be tested filling up to the mark with distilled water. The formation of a blood-red color will indicate the presence of sulphocyanate. Standard iron solution is then added from a burette to the first cylinder until the color corresponds with that of the solution being tested. Should the first cylinder lack much of being filled to the mark by this time a little more water may be added and if necessary a few more drops of the iron solution. Here as in the copper estimation it is well to enclose the cylinders in a box open at the bottom in such a way that the light may be reflected upward from a sheet of white paper and the color compared by looking down vertically through the column of liquid. Care should be taken not to have so much iron present in the test as to make the color too deep for comparison. When adding standard solution from the burette it is important that the contents of the cylinder be well mixed after each addition, and for this purpose it is better to turn the liquid out into a beaker each time.

Reduction Method for Estimating the Iron.—Should the amount of iron obtained from the cyanide solutions be large enough to be determined by reduction and direct titration, the procedure may be as follows: onto the filter paper containing the washed precipitate of ferric hydroxide pour some dilute sulphuric acid to dissolve it, taking care to wash the paper free from iron. Clennell[1] recommends boiling this solution with clean aluminium turnings until a drop of the liquid no longer imparts a red tint to a drop of potassium sulphocyanate solution. The flask is then cooled rapidly without removing the aluminium and titrated with standard potassium permanganate. Evolution of hydrogen must have entirely ceased before titration.

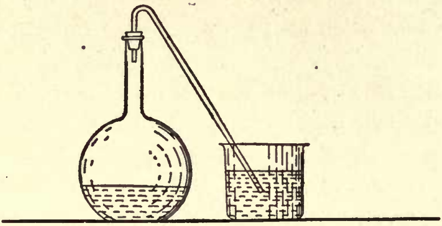

Fig. 3.—Apparatus for Making Ferrous Iron Solution.

The decinormal permanganate solution is made by dissolving 3.156 grams of the pure dry salt in distilled water and making up to 1 litre.

1 cc of this solution = 0.0056 gram Fe or 0.0369 gram $K_4Fe(CN)_6$ (potassium ferrocyanide).

If the permanganate is pure and dry the solution made up as above will be almost exactly decinormal, but it may be standardized against an iron solution, if desired.

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[1] Personal communication to the writer.

Fit a boiling flask with a rubber stopper through which passes a short length of glass tube to act as a steam vent, or better still, a bent tube long enough for its free end to pass below the surface of some water in a beaker. Remove the stopper and pour in about 100 cc of dilute sulphuric acid and a few grams of sodium carbonate to expel the air by formation of carbonic acid. Then place in the flask 0.1 gram of pure soft iron wire previously cleaned with scouring paper, and allow to dissolve. 0.1 gram of such iron wire may be considered to contain 0.0996 gram of pure iron. Cool the solution rapidly and titrate with the permanganate solution until a faint rose color remains permanent. With a strictly decinormal solution of permanganate 17.85 cc should be required to oxidize 0.1 gram of iron, or 17.78 cc for the 0.1 gram of wire taken.

Sulphocyanate Determination.—The method usually given for detection of sulphocyanate is to acidify and add a ferric salt, a blood-red color indicating the presence of KCNS. Silver and copper, however, interfere, and if present in sufficient amount may entirely prevent the formation of the characteristic color, so they must be removed before any determination can be made. The silver may be conveniently precipitated by shaking up a portion of the solution in a bottle with a little aluminium dust. There is usually sufficient caustic soda in working solutions to effect this reaction, but if not a tiny splinter of caustic soda may be added. In order to find out when precipitation is complete a little of the solution may be filtered and added to a flask containing dilute sodium sulphide; if any discoloration ensues agitation should be continued, or if necessary more aluminium dust added. If not, the solution may be filtered, and the test proceeded with.

If, after removing the silver, copper is still present, the writer has found that it may be conveniently eliminated by acidifying in presence of an excess of potassium ferrocyanide, the copper being precipitated as insoluble ferrocyanide and the excess of potassium ferrocyanide being removed as prussian blue on the addition of an excess of a ferric salt.

The procedure is as follows: If neither silver nor copper exist in the solution, or if silver alone was present and has been thrown out as described, take 100 cc of solution, acidify with a measured quantity of sulphuric acid, say 10 cc, add 10 cc of solution of a ferric salt (preferably ferric sulphate), and titrate with decinormal permanganate until the red color of the ferric sulphocyanate is dispelled.

If ferrocyanide is present it will come down as prussian blue, and must be filtered out, and it is safer to filter always, since small amounts of prussian blue may not be detected from the appearance of the liquor, and yet may be present in sufficient amount to impair the accuracy of the subsequent titration. When much prussian blue is produced, to avoid washing the precipitate, take an aliquot portion of the filtrate for titration, say 60 cc which in this case represents 50 cc of the original solution to be tested.

If the amount of ferrocyanide is large it may be necessary to add more ferric salt, because the red ferric sulphocyanate will not be formed until the whole of the ferrocyanide has been changed into prussian blue.

When Copper is present in the solution to be tested (silver being absent, or having been precipitated and filtered out as described), take 100 cc, add, for example, 5 cc of 1% solution of ferrocyanide, then 10 cc of sulphuric acid, and then 5 cc of ferric sulphate, making 120 cc in all. Filter, and measure out for titration 60 cc of the filtrate, representing 50 cc of the original solution, and titrate with $\frac{N}{10}$ permanganate.

The preparation of the decinormal permanganate has been already described. (Ferrocyanide determination.)

$$1 \text{ cc } \frac{N}{10} \text{ permanganate} = 0.001619 \text{ gm. KCNS.}$$

Sharwood[1] considers the determination by permanganate unreliable and prefers to estimate sulphocyanate colorimetrically, using the reverse process to that described for small amounts of

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[1] Engineering and Mining Journal, 1898, page 216.

iron. The writer, however, has found the permanganate method to give closely concordant results when tested on made-up solutions, provided that silver and copper are first eliminated.

Estimation of Other Compounds.—In trying to account for all the sources of cyanide loss during treatment of an ore it is sometimes useful to test the solution for cyanates, ammonia and ammonia compounds, formates and acetates.

Cyanate Determination.—Clennell[1] gives among other methods the following by O. Herting based on the fact that when KCNO is treated with an acid it does not decompose into HCN but reacts in the sense of the equation,

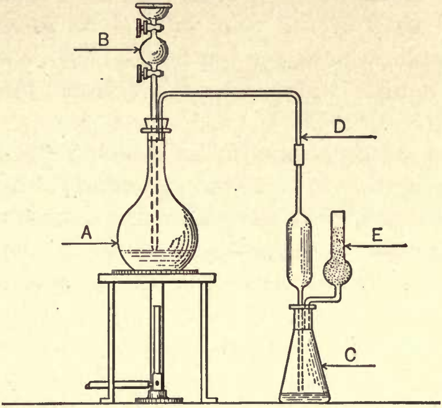

Fig. 4.—Ammonia Distillation Apparatus.

$$KCNO + 2HCl + H_2O = KCl + NH_4Cl + CO_2$$

By determining therefore the amount of ammonium compound formed a measure of the cyanate originally present is obtained.

The accompanying diagram, Fig. 4, shows the general arrangement of apparatus recommended by Sutton for the distillation of ammonia.

A is a distilling flask holding 200 or 300 cc fitted with a rubber stopper having two holes for glass tubes to pass through it.

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[1] Chemistry of Cyanide Solutions, page 108 (Second Edition).

B is a bulb tube with glass stopper above and stopcock below the bulb.

C is the receiver with rubber stopper through which the distillate tube *D* enters, and another tube *E* for escape of superfluous gases. The tube *E* is partly filled with glass beads or glass wool. In order to prevent the possibility of spray entering the distillate tube from the boiling liquid it is well to place a plug of glass wool in the neck of flask *A* just below the end of the distillate tube.

The distilling tube *D* should extend down almost to but not into the acid. Sutton recommends that both ends of the distillate tube and that of the vent tube *E* be ground obliquely. Should the distillate flask get too hot it may be stood in a dish of water, and a little water played over it at intervals with a wash bottle.

The method of procedure is as follows: Take a measured amount of the solution to be tested and add HCl in slight excess: evaporate to dryness over the water bath: dissolve the residue in water and transfer to the boiling flask *A*. Place 20 to 30 cc of strong NaOH solution in the bulb *B* and stopper the flask carefully. Loosen the stopper of flask *C* and pour in through the glass wool of tube *E* a measured quantity of decinormal acid. (The quantity of acid should of course be more than sufficient to combine with the whole amount of ammonia given off.) A few drops of methyl orange may be added if desired.

All being ready the stopcock in tube *B* is opened and the caustic solution allowed to run into flask *A*, which is then brought to a gentle boil. Boiling should be continued till at least two-thirds of the liquid in the flask has distilled over. When the operation is finished disconnect flask *C* and wash down the acid solution contained in the glass wool of tube *E* into the flask, great care being taken that every trace of acid is displaced. The contents of flask *C* are then titrated with decinormal alkali to ascertain the quantity of acid neutralized by the ammonia.

$$1 \text{ cc } \frac{N}{10} \text{ acid consumed} = 0.0042 \text{ gm. CNO}$$

Should ammonium compounds be present in the solution to be tested they must of course be estimated separately and their equivalent deducted from the value found before calculating the result to terms of CNO.

Determination of Ammonia and Ammonium Salts.—The following method is given by Clennell.[1] "Add excess of $AgNO_3$, *i.e.*, sufficient to precipitate all cyanogen compounds, then a little NaCl to remove excess of $AgNO_3$, filter, wash, evaporate filtrate on a water bath to a moderate bulk, distil with caustic soda or sodium carbonate and collect distillate in $\frac{N}{10}$ acid. Titrate residual acid with $\frac{N}{10}$ alkali and methyl orange. If free ammonia only is to be estimated use water instead of NaOH or Na_2CO_3 for distilling. In this case also the preliminary evaporation on water bath must of course be omitted."

Estimation of Formates.—The presence of formates in a cyanide solution is usually taken as a measure of the amount of ammonia and ammonium compounds formed by decomposition of the cyanide, according to the equation,

$$KCN + 2H_2O = HCO_2K + NH_3$$

The following method of determination is a modification by the writer of a method suggested by Clennell, and depends on the reduction of silver from the nitrate by a formate, in the sense of the equation,

$$2NaCO_2H + 2AgNO_3 = 2Ag + CO_2 + HCO_2H + 2NaNO_3$$

It has to be assumed that formates are the only substances present capable of reducing the nitrate of silver to the metallic state.

Procedure.—Take 100 cc of the solution to be tested. Add decinormal H_2SO_4 to neutralize the alkali exactly, using the amount required in the protective alkali test. Add $AgNO_3$ in slight excess and bring to boil. A blackening of the white precipitate first thrown down may be taken to indicate the presence

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[1] The Cyanide Handbook, page 525 (Second Edition).

of formates. Care should be taken to see that the filtrate contains free silver in solution. The precipitate is then washed to free it from the excess of silver and treated with ammonia or sodium thiosulphate to remove the silver cyanide which would otherwise dissolve to a sufficient extent in the subsequent nitric acid treatment to vitiate the results. The remaining blue-black precipitate is well washed to remove every trace of soluble silver and dissolved in 10% boiling HNO_3 solution. This solution is then titrated for its silver content with decinormal potassium sulphocyanate (Volhard's method).

$$1 \text{ cc } \frac{N}{10} \text{ KCNS} = 0.010788 \text{ gm. Ag} = 0.00450 \text{ gm. } CO_2H$$

This method may tend to give a result somewhat too low since precipitated metallic silver is slightly soluble in both ammonia (Prescott & Johnson, page 46, sixth edition) and hypo (H. F. Collins. The Metallurgy of Silver, page 187). The present writer has not determined the possible limit of error.

Volhard's Silver Determination.—This requires (1) a decinormal ammonium or potassium sulphocyanate solution, (2) a decinormal silver nitrate solution, (3) ferric indicator. A decinormal silver nitrate solution contains 16.988 grams of $AgNO_3$ per litre, or 10.788 grams of Ag. It may be made up by direct weighing of the pure salt or if preferred by dissolving 10.788 grams of pure silver in nitric acid.

Ferric Indicator.—This may be a saturated solution of ferric alum or a 10% solution of pure ferric sulphate. 5 cc of either of these is used for a titration.

Decinormal Ammonium or Potassium Sulphocyanate.—This salt being deliquescent cannot be weighed up correctly so an approximate solution is prepared by dissolving about 8 grams of the ammonium salt or 10 grams of the potassium salt in a litre of water. (Should the salt be unusually moist it will be better to increase the amounts.) The strength of this solution is then determined by titration with the decinormal silver nitrate, and a strictly decinormal solution of sulphocyanate prepared therefrom by adding the right amount of water. It should contain of the potassium salt 9.7 grams per litre or of the ammonium salt 7.6 grams per litre.

Method of Procedure.—Take 50 cc of the $\frac{N}{10}$ $AgNO_3$ solution and place in a flask. Add 5 cc of ferric indicator and about 10 cc of pure nitric acid. The acid used for this purpose should be diluted with about one-fourth its volume of distilled water, and boiled until colorless to expel the lower oxides of nitrogen. The sulphocyanate is then added from a burette. At first a white precipitate will be thrown down, and a momentary red coloration will be produced which will instantly disappear on stirring or shaking. As soon as all the silver has been precipitated one extra drop of the sulphocyanate solution will produce a permanent faint reddish coloration which is to be taken as the endpoint. If the sulphocyanate solution is strictly decinormal it will need exactly 50 cc of it to neutralize the silver nitrate taken, or

$$1 \text{ cc } \frac{N}{10} \text{ KCNS} = 0.010788 \text{ gm. Ag}$$

Estimation of Acetates.—Julian and Smart[1] state that cyanide solutions undergoing decomposition form not only formates but acetates. Clennell in a private communication to the writer states that he has never found acetates present in a working cyanide solution unless lead acetate was being used, but on the assumption that acetates may perhaps be formed in certain circumstances he suggests the following equation.

$$4KCN + 5H_2O = CH_3CO_2K + 2KCNO + KOH + 2NH_3$$

His method of determining acetates is as follows:

To 50 cc of the solution to be tested add neutral solution of silver sulphate in slight excess. Agitate, settle, and filter, wash with cold water. The acetates will be in the filtrate. Evaporate the filtrate to a small bulk, transfer to a boiling flask, and distill in the presence of sulphuric acid, collecting the distillate in 50 cc of decinormal NaOH. Continue distillation until about

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[1] Cyaniding Gold and Silver Ores, page 109 (Second Edition).

10 cc remains in the flask, say for one hour. Titrate the residual NaOH in distillate with $\frac{N}{10}$ acid.

$$1 \text{ cc } \frac{N}{10} \text{ alkali consumed} = 0.0059 \text{ gm. } CH_3CO_2$$

This method assumes that the filtrate from precipitation with Ag_2SO_4 contains no volatile acid except acetic. The presence of acetic acid in the distillate may be confirmed by making slightly acid with HCl, adding ferric chloride and boiling. A deep red color should be produced on addition of $FeCl_3$ giving a light red flocculent precipitate on boiling.

Estimation of Soluble Sulphides.—This is seldom called for, but it happens occasionally that an appreciable amount of sulphide is developed in the cyanidation of concentrate, and some sulphide is sometimes apparent in the press effluent from aluminium precipitation which it may be desirable to estimate. For such small amounts Clennell[1] recommends the colorimetric method.

For this purpose a weak standard solution of sodium sulphide is prepared, as described below. Two glass cylinders marked at 100 cc are taken, one of which is filled to the mark with the solution to be tested and a few drops of concentrated solution of sodium plumbite or alkaline lead tartrate added. If sulphides are present a brown coloration will become apparent. To the other cylinder which is filled nearly to the mark with water a similar amount of lead solution is added and the standard Na_2S solution run in from a burette until the brown coloration produced matches that of the test. The amount of sodium sulphide in the test cylinder is then calculated from the quantity of Na_2S used to match it.

It sometimes happens that the tone of color produced in the cyanide solution does not compare well with that in the water cylinder, and if the solution to be tested contains only traces of silver as would be the case with barren precipitate press effluent some of this solution may be shaken up with a little lead carbon-

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[1] The Cyanide Handbook, page 523 (Second Edition).

ate or litharge and filtered and the filtrate used instead of water for making up the second cylinder.

Standardizing the Sodium Sulphide.—This is best done with standard zinc solution as described by Sutton (Vol. Anal.). The sulphide solution may be made by taking a measured amount of solution of caustic soda and saturating half of it with H_2S gas and then adding the remainder of the caustic solution, or the fresh c.p. crystals may be used. It is best to make a fairly strong solution for the standardization and afterward dilute down to a suitable strength by adding the correct amount of water, since the method of standardizing is not satisfactory with very dilute solution.

The standard zinc solution is made by dissolving 44 grams of pure zinc sulphate in distilled water and making up to 1 litre. Or 10 grams of pure metallic zinc may be dissolved in hydrochloric acid and made up to 1 litre with distilled water. 1 cc of this solution will then equal 0.01 gram of metallic zinc. The indicator may be made by dissolving a little lead tartrate in a solution of caustic soda.

50 cc of the standard zinc solution is now taken and treated with ammonia to which a little ammonium carbonate solution has been added. A precipitate of zinc hydroxide will first be formed and addition of the ammonia must be continued until this re-dissolves to a clear solution. The sodium sulphide solution to be standardized is then run in from a burette and at intervals a drop of the liquid is withdrawn on a glass rod and placed on a piece of white filter paper: as soon as ever the filter paper has absorbed it a drop of the lead indicator is placed on the filter paper sufficiently near that the edges of the two circles of moisture will overlap one another. If no coloration appears at the point of contact the titration is not complete and addition of the sulphide is continued, a drop being withdrawn at inte vals and tested in contact with the lead indicator on the filter paper, until a black or brown line appears at the point of contact of the two circles of moisture, indicating the end of the titration.

1 cc standard zinc solution =

0.01194 gm. Na_2S
0.01687 gm. K_2S

When placing the drop of liquor from the test on the filter paper the precipitate of zinc sulphide will be seen to remain in the center portion of the circle while the clear liquor will expand beyond it: if the lead indicator spreads across this clear liquor and touches the inner circle of zinc sulphide it will react to give a black coloration regardless of whether the titration is finished or not so care should be taken to place the indicator sufficiently far away that it may only come in contact with the clear liquor without touching the precipitate.

The first trial will probably be overdone but a careful repetition will usually enable the exact end point to be reached. When the strength of the sodium sulphide solution has been determined a given amount may be measured out and diluted to a suitable strength for the purpose desired. This very weak sol tion deteriorates rapidly and a fresh batch should be diluted every few days. Even the strong stock solution decomposes steadily and should be frequently re-standardized.

Estimation of Oxygen in Working Cyanide Solutions.—Until recently no simple and reliable method has been known for the estimation of free oxygen in working cyanide solutions so that the one here described supplies something that has long been needed. The matter takes on an added importance in the light of the results being obtained by the Crowe Vacuum Process for precipitation.

This method was devised by H. A. White and is described in the Journal of the Chem., Metall. and Min. Soc. of S. A. for June, 1918. It is based on the degree of coloration imparted to a solution of pyrogallic acid or some similar substance in the presence of caustic soda by the oxygen contained in the solution.

The description is given chiefly in the author's own words.

"*Apparatus Required.*—One dozen (nominal) 250 cc stoppered bottles, fairly regular in diameter and capacity: one 50 cc burette.

"*Chemicals Required.*—Sodium hydrate 2N solution (80 gm. per litre), pyrogallic acid or other suitable developer, brown dye." (H. A. White.)

Preparation of standards for comparison: A suitable volume of distilled water must first be saturated with air and for this purpose the simplest method is probably that recommended by Sutton.[1] Two large acid bottles are taken and half filled with distilled water; they are then well shaken for a few minutes or to save trouble may be put on the agitator wheel for an hour or so. The air in the two vacant spaces is then replaced by pouring the contents of one bottle into the other and vice versa after which the water is evenly divided between the two bottles and again subjected to agitation; this is repeated several times after which it may be considered to be saturated and the bottles are allowed to stand at rest for half an hour before being used. This water is then used as a standard by which to make up the standard colors for the comparisons. The point of saturation of water with atmospheric oxygen will of course vary with the temperature and pressure so in order to find the amount of oxygen present at saturation point under various conditions the following table by Roscoe and Lunt[2] is referred to, after taking a reading of the temperature of the water and the barometric pressure.

The table is calculated on the basis of a barometric pressure of 760 millimeters but if the observed pressure is below this $^{1}/_{76}$ of the value must be subtracted for each 10 millimeters difference while if the observed pressure is above 760 millimeters $^{1}/_{76}$ of the value given must be added for every 10 millimeters difference. For instance, suppose that the temperature of the water after aeration is 21 deg. C. and the barometric pressure 620 millimeters, then from the table the number of cc of oxygen representing the saturation point at 21 deg. C. (for a barometric pressure of 760 millimeters) is 6.16. The difference in the barometric pressures is $760 - 620 = 140$ and the correction is therefore $6.16 - (^{14}/_{76} \times 6.16) = 5.026$, showing that under the conditions assumed the

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[1] Volumetric Analysis, page 279 (Ninth Edition).

[2] Quoted by Sutton, *ibid.*, p. 280, from the Journal of The Chemical Society (London), 1889, p. 532.

TABLE 3.—OXYGEN DISSOLVED BY DISTILLED WATER. 5–30°C.

Temp., deg. C.	Cc oxygen, N.T.P. per litre Aq.	Difference for 0.5°C.	Temp., deg. C.	Cc oxygen, N.T.P. per litre Aq.	Difference for 0.5°C.
5.0	8.68	....	18.0	6.54	0.07
5.5	8.58	0.10	18.5	6.47	0.07
6.0	8.49	0.09	19.0	6.40	0.06
6.5	8.40	0.09	19.5	6.34	0.06
7.0	8.31	0.09	20.0	6.28	0.06
7.5	8.22	0.09	20.5	6.22	0.06
8.0	8.13	0.09	21.0	6.16	0.06
8.5	8.04	0.09	21.5	6.10	0.06
9.0	7.95	0.09	22.0	6.04	0.05
9.5	7.86	0.09	22.5	5.99	0.05
10.0	7.77	0.09	23.0	5.94	0.05
10.5	7.68	0.08	23.5	5.89	0.05
11.0	7.60	0.08	24.0	5.84	0.04
11.5	7.52	0.08	24.5	5.80	0.04
12.0	7.44	0.08	25.0	5.76	0.04
12.5	7.36	0.08	25.5	5.72	0.04
13.0	7.28	0.08	26.0	5.68	0.04
13.5	7.20	0.08	26.5	5.64	0.04
14.0	7.12	0.08	27.0	5.60	0.03
14.5	7.04	0.08	27.5	5.57	0.03
15.0	6.96	0.08	28.0	5.54	0.03
15.5	6.89	0.07	28.5	5.51	0.03
16.0	6.82	0.07	29.0	5.48	0.03
16.5	6.75	0.07	29.5	5.45	0.02
17.0	6.68	0.07	30.0	5.43	
17.5	6.61	0.07			

water would contain 5.026 cc of oxygen at saturation point. To obtain the result in milligrams of oxygen multiply the above figure by 1.429 which is the weight in milligrams of 1 cc of oxygen at normal temperature and pressure. The figure thus obtained in this case is 7.18 milligrams.

H. A. White continues,

> "Fill (with a siphon), completely, one of the 250 cc (nominal) bottles with the water and add 0.100 gm. of solid "pyro"—the densely crystalline form sold as "Pyraxe" is preferred—and shake all the crystals

below the water surface. Then add from a burette, dipping below solution level, 1 cc of the 2N NaHO and immediately close the bottle with the stopper in such a manner as to exclude any air bubble at all. Shake up till all the crystals are dissolved, and observe the colour.

"This colour is then matched with the solution of a brown dye. I find that 'Diamond' brown requires a little acid, methyl orange and a small amount of chromate of potash to match the shade exactly. Make up about three litres of the dye solution, adjusted till another 250 cc bottle-full of it will exactly match the water tested when both are held together to the light.

"A somewhat better match, preserving the tint well on dilution, may be made by using the brown colour obtained by shaking up a solution of pyrogallol or eikonogen with excess of air till further darkening has ceased. In this case some slight change of tint takes place in a day or two.

"Take eight of these nominal 250 cc bottles and determine the exact capacity, which will be very close to 300 cc, and number 1 to 8. Into No. 1 pour the matched dye solution and insert the stopper without leaving any air bubble. This will represent the amount of oxygen found in the water sample, and the other bottles will have poured into them a proportionate quantity of the dye solution till the series represents 7, 6, 5, 4, 3, 2, 1 and ½ mgm. of oxygen per litre respectively.

"If the water sample does not come out sufficiently near 7.0 mgm. per litre, the following figures will have to be varied to correspond; and the capacity of the bottles, if more than 2 cc or 3 cc away from 300 cc, will also naturally affect the calculation.

"Assuming 300 cc bottles and 7.0 mgm. oxygen per litre in the sample water—

Into No. 1 bottle pour 300 cc matched dye solution and fill with water.
Into No. 2 bottle pour 257 cc matched dye solution and fill with water.
Into No. 3 bottle pour 214 cc matched dye solution and fill with water.
Into No. 4 bottle pour 172 cc matched dye solution and fill with water.
Into No. 5 bottle pour 129 cc matched dye solution and fill with water.
Into No. 6 bottle pour 86 cc matched dye solution and fill with water.
Into No. 7 bottle pour 43 cc matched dye solution and fill with water.
Into No. 8 bottle pour 21.5 cc matched dye solution and fill with water.

In any case the standards are adjusted to represent from 7 down to ½ mgm. per litre of oxygen, and are so marked, preferably, with a diamond on top of the stopper. The four remaining bottles are marked *A*, *B*, *C* and *D* in the same way."

"Any solution may now be tested by plunging a bottle, held vertically, well below solution level, jarring it and moving to allow all air bubbles to escape, then dropping in the stopper while still under solution. The stopper may be given a downward twist when removed from the solution and the bottle will then be sufficiently air-tight for the few moments which elapse before testing. Using a siphon to fill the bottle is obviously a more accurate method, but that given is sufficient and more easily applied. Solutions in contact with pulp must be settled in a two-litre bottle filled in the above manner, and the sample bottle filled therefrom by a siphon.

"In testing the sample of solution, the stopper is gently removed with an upward twist so as not to lose any liquid. The 0.100 gm. pyrogallol (of which a sufficient number of lots is ready weighed out) is then dropped into the bottle and shaken down below solution level; 1 cc of 2N NaHO is then run in from the burette with point below solution level, and the stopper instantly replaced in such a way that no air bubble is left. After shaking till all crystals are dissolved, the sample bottle is placed between the two standards it more nearly resembles, and thus the oxygen may be determined to 0.5 mgm. per litre. If the colour is not exactly the right tint it will become so within 15 minutes—some solutions retain a sort of purplish tint with pyrogallol for a short time."

Assay of Cyanide Solutions for Gold and Silver. *Method No.* 1.—Evaporate a known quantity in a square boat made of lead foil, scorify, and cupel. This is not reliable for low grade solutions because the amount that can be taken for assay is too small.

Method No. 2.—Evaporate a known quantity in a large porcelain dish whose bottom is covered with a thin layer of litharge. When dry remove the litharge, carefully clean the dish with small pieces of filter paper dipped in dilute hydrochloric acid, the paper being dried, burnt, and added to the litharge. The latter is then fluxed and fused down in a crucible. The method may be used for amounts of solution up to 20 assay tons and is reliable though tedious.

Method No. 3 (Virgoe).—To a known amount of solution add weak copper sulphate solution in slight excess, that is, until the clear filtrate shows a faint blue tint, then acidify with hydrochloric or sulphuric acid, and filter. It is also advisable to add a little sodium sulphite solution to ensure that the last traces of gold are thrown down. Should the solution be very weak in cyanide it is advisable to strengthen it to about 0.05% KCN before precipitating. For 10 assay tons of solution Clennell[1] recommends

Copper sulphate (10% $CuSO_4.5H_2O$)	20 cc
Sodium sulphite (15% Na_2SO_3)	20 cc
Sulphuric acid (10% H_2SO_4)	10 cc

These are added in the order given stirring after each addition. The precipitate is then filtered, but washing is unnecessary.

When sufficiently drained 60 grams of the following flux are placed in the same filter paper:

Borax	30 parts
Litharge	30 parts
Charcoal	1 part

The paper is then folded up and fused down in a small crucible, the resulting button of lead being cupelled.

Method No. 4 (Chiddey-Magenau).—To 10 A.T. of solution (which if very weak in cyanide should be strengthened up to about 0.5% by addition of the proper quantity of concentrated solution) add 10 or 12 cc of a 10 per cent. solution of lead acetate and about 0.3 to 0.4 gram of zinc dust: stir and bring to a boil. Then add 15 cc of strong HCl and leave on the hot plate until the excess zinc has all dissolved, when the beaker should be removed and allowed to cool. Decant the solution and wash the spongy lead several times in large volumes of water. The lead may then be collected and pressed into a ball between the fingers. This ball is then folded up in lead foil in which a vent has been left for escape of steam, and cupelled.

Colorimetric Estimation of Gold in Cyanide Solution.—The following method was devised by C. W. Dowsett, Mill Superintendent of the Dome Mines in Ontario.

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[1] The Cyanide Handbook, page 517 (Second Edition).

Take approximately 1000 cc of solution in a tight stoppered jar. A Mason fruit jar or pickle bottle is convenient.

Add 2 to 3 drops of lead acetate. (Clear saturated solution.)

Add a pinch of zinc dust. (Approx. 2 grams.)

Add 10 to 55 cc saturated NaCN solution to bring up to about 0.1% KCN.

Shake well for about 20 to 30 seconds.

Pour out into a large evaporating dish.

Settle and decant clear solution.

Add 10 cc aqua regia and evaporate to 3 cc.

Pour into small test tube and cool thoroughly. (Cooling is very important.)

Add a few drops of fresh, saturated, stannous chloride solution.

The presence of gold will be indicated by a purplish ring or, if the tube is shaken, by a purplish tinge throughout.

The whole operation may be performed over a spirit lamp or similar flame and does not take over 5 minutes. The presence of as little as 0.02 dwt. gold per ton of solution is very plainly shown.

Colorimetric Estimation of Silver in Cyanide Solutions.—A few drops of a 10% solution of sodium sulphide added to 25 to 50 cc of the solution to be tested, gives a pure white precipitate of zinc sulphide in the absence of silver. The precipitate becomes brownish in the presence of silver, and the depth of color is a very close indication of the amount present.

CHAPTER III

SAND LEACHING

Sand Leaching.—When the cyanide process was first introduced its application was confined to leaching, that is, treatment by percolation. By this method only such material could be worked as was sufficiently granular in texture to allow the necessary amount of solution to percolate through it in a reasonable time. By crushing the ore dry, it is sometimes possible to leach it direct without removing the slime, but in the majority of cases this is not feasible, and it is necessary to separate the sand from the slime, leaching only the former and treating the latter by agitation.

Separation of Sand and Slime.—In cyaniding the current tailing produced by amalgamating or concentrating mills, a rough separation of the percolable material was originally made by running the pulp into pits where the sandy part was deposited, allowing the slime to overflow and either letting it go to waste or settling it in large dams from which the clear water could be pumped back to the mill.

Later, a system was developed whereby the pulp from the mill was led to collecting tanks, in which the sand was retained and the slime flowed away, not yet to any method of treatment but to storage dams.

There were two methods in use at this time in South Africa for effecting the separation of the sand, one was by the Butters and Mein distributor, and the other by the "nigger with a hose."

The Butters and Mein distributor consists of a circular hopper from which radiate pipes of various lengths and diameters, whose extremities are bent horizontally at a right angle, and sometimes flattened to produce a fan-shaped discharge. The length and diameter of each pipe is so calculated as to deliver an equal tonnage of pulp to a proportional ring-area of the tank. The hopper is supported or suspended level with the top of a circular tank, on a bearing, so that when the stream is led into it the discharge from the pipes causes the whole to revolve slowly, with the object of making a uniform deposit of sand over the surface of the tank bottom.

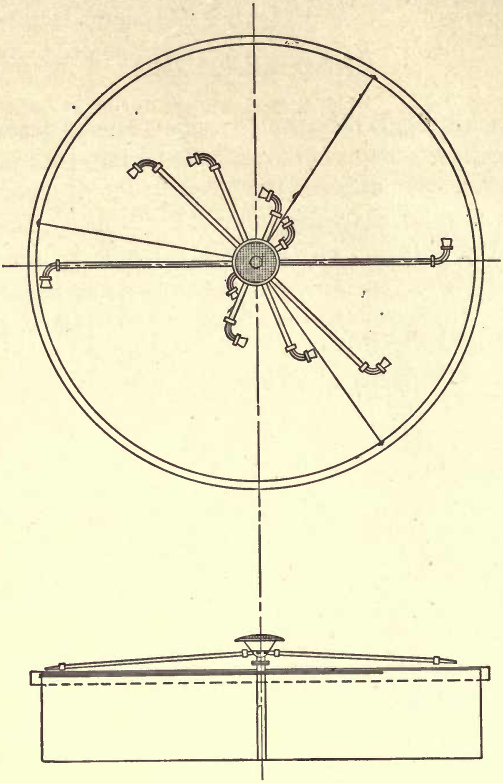

Fig. 5.—Butters and Mein Distributor.

In the original form of this machine the tank was fitted with an annular overflow launder, and was filled to the brim with water before the pulp was turned into it. When in action, the sand sank down to the bottom, and the slime and surplus water overflowed into the ring-launder until the tank was filled with sand. The drawback to this system as a separator of sand and slime was that in the event of a shut-down of the mill all the slime in suspension above the sand settled down on it forming an impervious layer. The machine itself, however, has come back into favor as a method of filling a tank with sand from which the slime has been previously separated, a function which it performs very satisfactorily.

An improved form of this machine, when used as a separator, consists in discarding the top overflow and making two or more openings, capable of being closed by water-tight doors, in the side of the tank near the bottom. On the inner side of the tank at the point where the doors are placed an adjustable weir is attached, whose height of discharge can be raised as the level of the sand rises, either by slipping narrow wooden slats into a pair of grooves, or by rolling up a canvas curtain over a wooden grating. This system works fairly well, but has a tendency to leave groove-shaped depressions containing solid deposits of slime.

With the "Nigger and Hose" system the pulp is led to a large spitzkasten or spitzlutte where part of the slime and water overflows; the remainder or underflow discharges into a large rubber hose leading to the sand tank to be filled. The tank, in this system as in the one already described, is fitted with low exit doors and adjustable weirs for the removal of the slime and water. A Kaffir is stationed in the tank to keep the discharge end of the hose moving from place to place so as to build up a deposit of sand as evenly and as free from slime as possible.

The quality of the charges produced by both these methods is fairly satisfactory, but varies within wide limits according to the diligence and efficiency of the shiftman. It is found difficult, however, to get a satisfactory direct treatment of such charges, owing to the density of packing and unhomogeneous nature of the deposit, and they are almost invariably transferred to another tank for the leaching treatment.

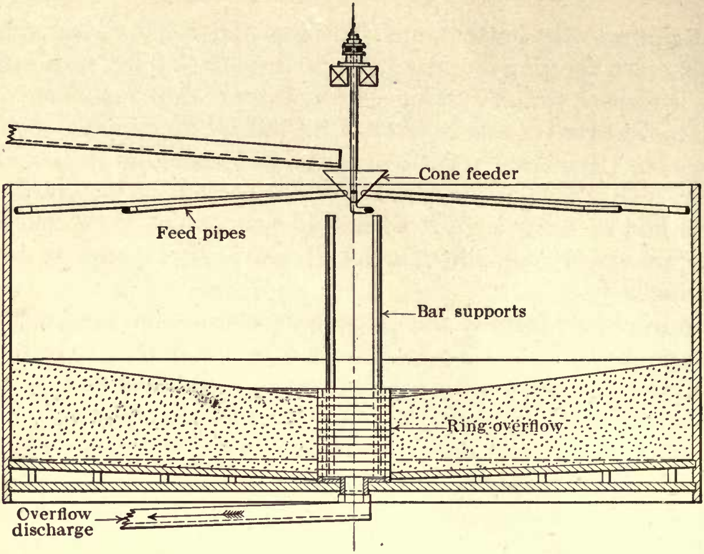

FIG. 6.—Hamilton Collecting Tank.

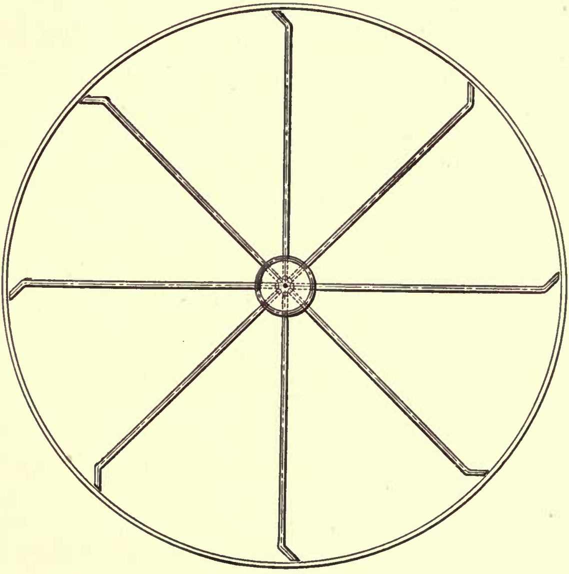

FIG. 7.—Distributing Pipes, Hamilton Collecting Tank.

Hamilton Collecting Tank.—To obviate the erratic deposition resulting from the Butters and Mein distributor, and to combine the advantage of the hose distribution with that of automatic and mechanical efficiency, the writer designed a collecting tank in which the pulp entered at the periphery and the slime flowed off at the centre. The intake of pulp was effected by a mechanism similar to the Butters and Mein distributor except that all the pipes were of the same length and extended almost to the side of the tank, being bent at their extremities not at a right angle but at 45 deg. so that the pulp would discharge against the side of the tank without making any splash and yet exert sufficient tangential force to cause the hopper to revolve on its ball bearing.

The central overflow weir was circular and its height was adjusted to the rising level of the sand deposit by adding, periodically, tongued and grooved rings 1½ to 2 inches in height and thus gradually building up the discharge level until the tank was full of sand.

During filling the pulp was thrown uniformly against the side of the tank whence it flowed down vertically to the bottom, and then toward the centre discharge, the sandy portion building up in the form of an inverted cone, sloping toward the centre, and the slime and water flowing over the centre weir. In working, the sand surface was never allowed to become submerged except for a few inches around the central overflow, so that the sand particles could settle down one on another leaving the slime to flow on over the surface till it reached the small tranquil pool just around the overflow which served to trap and hold the finest of the sand. Very little slime was deposited in this pool because as the film of water and slime flowing over the inclined surface converged toward the centre the velocity increased sufficiently to hold the slime in suspension until it had passed over the weir. The tank was fitted with a filter bottom of the usual construction but coned slightly toward the centre, for draining the charge dry when collected.

This system proved a great improvement over previous methods especially in handling the fine material resulting from tube mill regrinding, being capable of separating sand, a large proportion of which was finer than 150 mesh and yielding a leaching product containing only from 4% to 5% of slime uniformly distributed through it. The following table gives a screen analysis of sand charges collected by this system at a mill where really fine regrinding was practised. This material when transferred to the leaching tanks percolated at the rate of 1½ inches an hour through charges 3 ft. in depth.

+60	0.0%
− 60+ 80	2.9%
− 80+100	11.6%
−100+200	36.8%
−200 sand	43.9%
Colloidal material	4.8%

Double Treatment.—In the course of experiments in direct treatment it was found that when a charge was moistened with cyanide solution before being transferred to the treatment tank the extraction was invariably better than when the first contact with cyanide took place after transferring, and this was attributed to the increased activity of the solution caused by the exposure of the cyanide-coated particles of ore to the action of the air in transferring. This discovery led to the universal adoption on the Rand of the double treatment system of leaching, which was the standard practice for many years and is largely made use of up to the present time (1919).

At the Homestake mine where the assay value of the original sand was so low as to make economy in handling absolutely necessary to a commercially profitable treatment C. W. Merrill devised a separating system whereby the double treatment was rendered unnecessary. The principle involved was a complete separation of the slime before collecting the sand leaching charge, and this was effected by means of a series of Merrill Cones. This series is composed essentially of two sets of gravity cones followed by one set of smaller hydraulic cones, the underflow of each set forming the feed of the next set in the series.

The underflow or finished product from the hydraulic cones passes to a Butters and Mein distributor tank of the original type with peripheral overflow and the charge after collection is so homogeneous and porous that leaching can be completed without any transfer. The aeration, which proved to be an indispensable adjunct to the successful treatment of this ore, is effected by draining the charge dry and introducing compressed air under the filter bottom for a set period of time before applying the first cyanide wash. The aeration is repeated at intervals during treatment, between solution washes.

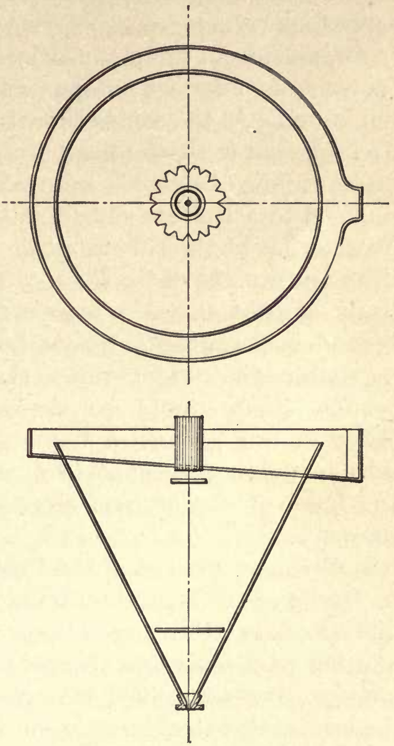

FIG. 8.—The Caldecott Cone.

Caldecott's Choke-discharge Cone.—For making a clean separation of the sand Caldecott uses a cone with a metal disc or diaphragm fixed on the inside near the apex. This disc may have a plain or serrated edge and is so placed as to leave an annular space between its edge and the side of the cone. At the apex is a plug cock or other device for regulating the rate of discharge. When working, the latter is regulated so as to maintain a solid bed of settled sand extending up to within 10 in. of the overflow. Under these conditions a thick sluggish stream of almost clean sand issues from the bottom while the water and suspended slime overflows around the top. The purpose of the diaphragm is to minimize the chances of a funnel forming in the bed of sand, through which the slimy water would break and issue from the spigot.

As a certain amount of slime is entangled with the sand produced in this way, Caldecott recommends passing the product to a second diaphragm cone with clean water to complete the separation.

The sand from this second cone is run onto a revolving horizontal vacuum filter where the residual moisture is reduced to a minimum. At the completion of each revolution of the table the drained sand is automatically scraped off into a hopper where it meets a flow of cyanide solution, and the pulp thus formed is pumped to a Butters and Mein Collecting tank with peripheral overflow, as at the Homestake.

The advantage of the Caldecott cone over the Merrill seems to be in the possibility of a more complete elimination of the slime, because in the latter the spigot discharge is free and it would seem inevitable that more slime should escape with the finished product than would be the case with a choke discharge. When milling is done in water the Caldecott system also permits of pulping the sand in cyanide solution before collecting the leaching charge, and thus lengthening the time available for contact.

Both these systems of sand collection utilize the earlier form of Butters and Mein distributor tank, but with the important difference that this apparatus is used exclusively as a collector and not as a separator of sand from slime. The tank with its annular overflow is filled with water or cyanide solution and the clean separated sand is fed to the distributing hopper in a medium of water or solution, the solids gradually replacing the liquid until the tank is filled with sand.

The two main results accomplished by this change are first, the impossibility of collecting a slimy charge provided that the previous separation has been properly conducted, and second, the formation of a sand charge of such uniform texture and percolating properties that the whole leaching treatment can be carried out in the same tank, yielding the maximum extraction obtainable and obviating the cost of a transfer to another tank during the treatment.

A charge collected in this way possesses superior uniformity of structure and leaching and aerating qualities to one collected by any of the methods using an adjustable overflow weir which keeps the sand surface only partly submerged, on account of the different positions assumed by the individual particles relatively to one another in the two cases. When angular particles in a water medium are rolled over one another or down an incline composed of similar particles there is a strong tendency for them to come to rest and find lodgment with their flat surfaces in contact with one another, forming an interlocking and compact mass with a minimum of interstitial spaces, whereas when they sink slowly through a tranquil body of water they are likely to remain in the relative positions in which they first come to rest and that will be, as often as not, with angles in contact with plane surfaces, giving more interstitial capacity and therefore making the whole charge more permeable to solution and to air.

By this method a uniform mixture of coarse and fine particles is also attained, and the deposition of patches, bands, and strata of alternating coarse and fine material is avoided, thus allowing of a uniform rate of percolation through every part of the charge.

The introduction of mechanical classifiers such as the Dorr, Esperanza, Bayliss, Colbath, Akins, etc., affords a very convenient and efficient method of applying this system.

In principle, of course, nothing could be simpler than a cone classifier, but in practice it needs constant attention to avoid irregularities in working. Bits of wood, waste, washers, nuts, and various other articles have a way of inserting themselves in the apex and if not at once seen and removed, throw that unit out of commission for the time being. Hydraulic cones have a tendency, too, to bank up gradually, even when the spigot discharge is free and unobstructed, and generally require emptying at intervals.

The proper working of the Caldecott cone, also, depends on a close adjustment of the spigot discharge to the amount of inflow-

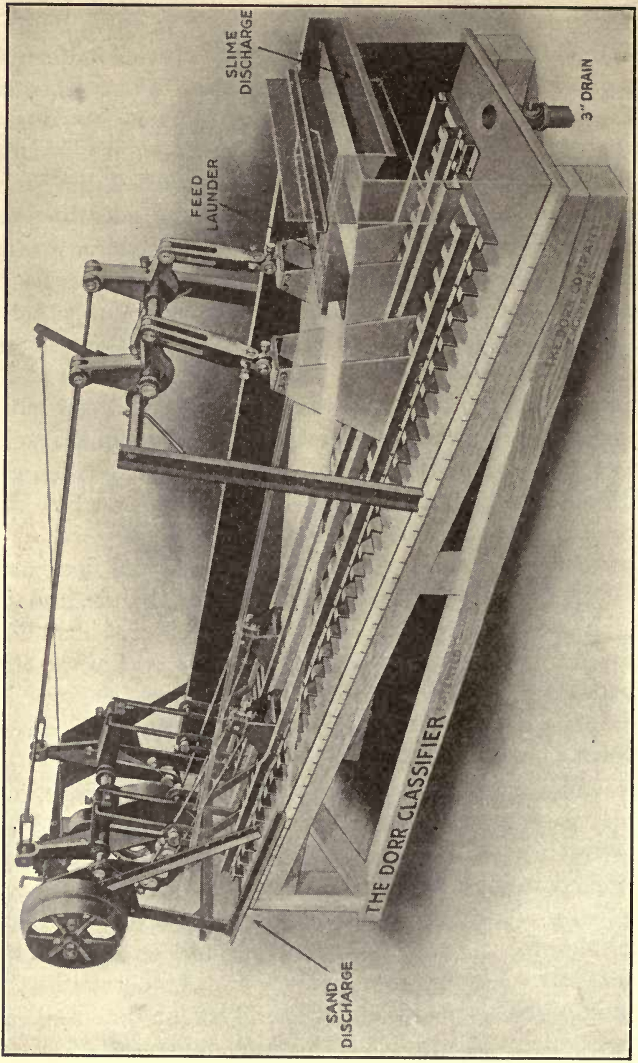

FIG. 9.—The Dorr Classifier.

ing pulp. Any falling off in the quantity of sand entering causes a lowering of the sand level within the cone and finally a "breaking through" or discharge of muddy water through the spigot, while any sudden increase in the pulp supply raises the sand level in the cone to a point where sand is carried over in the slime product. This fluctuation is not so important when working with large units which allow of a fair margin of variation, but with small units constant attention is needed in order to obtain satisfactory work.

The mechanical classifier, on the other hand, provided that it is not overloaded and that the ratio of liquid to solid is sufficiently large to allow of the disentanglement of the sand from the slime by the action of gravity, will perform its function efficiently regardless of wide variations in pulp flow, and with practically no attention.

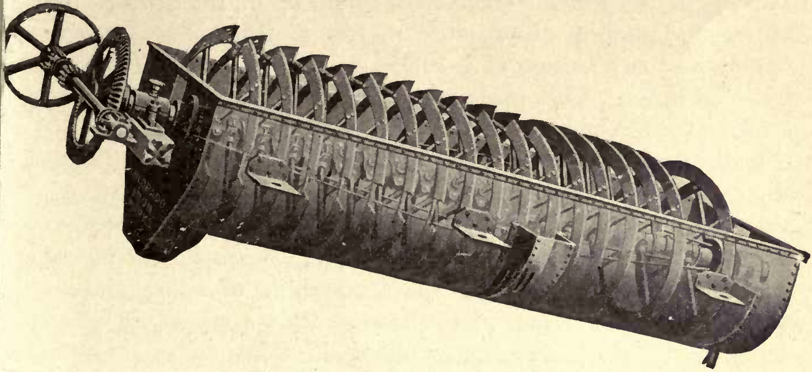

Fig. 10.—The Akins Classifier.

A very satisfactory system of sand leaching by direct treatment in use in at least one Mexican mill consists in separating the sand in a mechanical classifier, adding clear solution to the product and passing it to a second similar classifier, and then transferring from this by means of a stream of clear solution to a Butters distributor tank with peripheral overflow. When the tank is full of sand the leaching treatment is proceeded with and completed in the same tank, and the residue sluiced out of the dump.

A useful form of classifier for thoroughly washing the sand free from slime before discharging it into the Butters distributor for this system of collection would seem to be the Dorr washing classifier which combines three machines in one, the first compartment separating the sand from the slimey water or solution and discharging it successively into a second and third compartment containing clear solution which completes the removal of the entangled slime.

The Leaching Treatment.—Leaching is usually carried out in circular tanks of wood or steel with a false bottom covered with some filtering medium. A grid is first made on the bottom of the tank by laying wooden strips about 2 in. by 3 in. on edge, a foot apart, and nailing across and on top of these, smaller strips having a square section of 1 to 1½ in., and leaving 1-in. spaces between them. This grid is preferably made in sections sufficiently small to be handled by two men, as it has to be lifted periodically for the purpose of cleaning out accumulated sand and slime that may have worked through the filter. On top of the grid is laid a circular mat usually composed of ordinary cocoa fibre matting, and on this a cloth of a texture fine enough to prevent the passage through it of slime and fine sand. The best material to use for this purpose is closely woven jute cloth such as is used in Mexico and Central America for sacking coffee for export. The material commonly known as "burlap" is not suitable as a top covering because the texture is usually too coarse. Six- or eight-ounce duck is often placed above the cocoa mat instead of jute and answers the purpose well except that it tends to reduce the rate of percolation considerably.

The top covering, whatever it may be, is securely fastened in contact with the side of the tank, either by extending it a few inches up and tacking a flexible strip of wood over it, or in the case of a steel tank by driving a thick rope with a wooden caulking tool down between the grid and the side of the tank. For this purpose the grid is made 2 in. smaller in diameter than the size of the tank, and is enclosed by a continuous rim made from a flexible strip of wood, so as to give a solid backing for the rope caulking.

A pipe, 1½ to 2 in. in diameter, enters the tank through the bottom for drawing off the solution. The system of connecting the outflow pipes from the whole system of leaching tanks into a single header for conveyance to the precipitating house is not recommended, as it makes control of individual charges difficult both as regards sampling the effluent solution and also adjusting its rate of flow. If the precipitation house is at a distance from the leaching tanks, the individual pipes may discharge through regulating valves into a small tank or box near by, from which the solution may flow in a large common pipe line to the extractor house. If it is desired to circulate two different grades of solution, the small receiving tank may have two compartments, and the solution from any tank may be readily diverted to either compartment.

When the sand tank is filled and ready for treatment the solution may be run onto the surface of the charge and allowed to percolate downward, or it may be introduced under pressure beneath the filter and caused to percolate upward. The latter method is chiefly valuable where the charge contains a large admixture of fine material and slime, such as results when a dam of accumulated tailings is dried and coarse and fine mixed together for treatment of the whole by percolation. Such a charge might pack so densely under the influence of gravity percolation, especially if drained dry after a wash, that no subsequent washes would pass through, whereas by continuous upward percolation and removal of the effluent from the surface a rapid rate of flow may be maintained throughout the treatment. The usual practice, however, is to apply the solution on top and let it gravitate downward through the charge. It is often found a good plan to fix a stand pipe of about two inches in diameter against the side of the tank, the upper end terminating at the top and the lower end extending down through the filter mat which is tightly caulked around it. This acts as a vent for the air driven down in front of the mass of descending solution, which would otherwise be forced to escape upward through the sand in the form of bubbles, thus forming channels and spoiling the uniformity of the subsequent percolation. The leaching cock is better kept closed until the whole charge is saturated and the solution stands permanently above the level of the sand. It may then be opened and regulated to any desired rate of flow.

When the cyanide process was first introduced a great point was made of letting the solution remain in contact for many hours before drawing off. This soaking procedure does not seem to have anything to recommend it, since diffusion is almost impossible during this stagnation period, and consequently the film of solution in contact with the particles would soon become saturated, and also denuded of its oxygen, with the result of delaying rather than hastening the dissolution of the precious metals. The idea underlying this practice was probably the desire to effect the dissolving with as small a bulk of strong solution as possible, both as a means of saving cyanide and also of obtaining a solution richer in the precious metals than would otherwise be the case. A method of accomplishing this result without retarding the dissolving action has been used with good results in some mills, notably at Kalgurli, and El Oro, Mexico.

This consists in attaching a small air lift to the outside of the tank, arranged so that as the solution percolates and reaches the space beneath the filter bottom it is at once returned to the surface of the sand. In this way a small bulk of strong solution is kept circulating through the charge for as long as may be deemed advisable, and in its passage is given an opportunity of taking up oxygen to replace that which has been absorbed by the ore.

There are two methods in use for applying the solution. The first is to pump it on as fast as it percolates, thus keeping the charge always saturated or even covered, and the second is to pump on enough to saturate the charge and cover it to a depth several inches and then to drain it off and allow the sand to stand dry for several hours before the next application of solution. The second method is almost always preferable, because, except in the case of highly oxidized gold ores, the circulating solution does not carry sufficient dissolved oxygen to satisfy the reducing action of the charge and needs to be supplemented by other opportunities for oxygen absorption. By draining dry between washes the air is drawn down into the interstices of the charge as the solution recedes, and finds ideal conditions for absorption. In the case of very fine sand it is useful to apply a vacuum under the filter to remove the excess moisture due to the increased action of capillarity. When dealing with exceptionally reducing ores compressed air may then be introduced beneath the filter, if necessary, as is done at the Homestake. The system of draining dry between washes as compared with continuous leaching has been known to result in an increased extraction of 25% under actual working conditions on combined gold and silver ore.

The rate of percolation is usually measured by the number of inches per hour that the level of the solution on top of the sand falls as the wash percolates downward and flows out through the discharge pipe. Generally speaking the finer the sand the slower is the rate of percolation, but Julian and Smart[1] state that "if an ore be crushed to pass through an ordinary assayer's sieve of 90 mesh and washed free of slime, a good leachable product may be obtained, but if this fine ore be mixed with coarser particles, percolation becomes slower than the average of the two sizes treated separately." The presence of slime, even when uniformly distributed, has a serious retarding action on the rate of percolation, especially when leaching fine grades of sand. The depth of the charge is also a factor to be considered, there being a slight retardation in percolation for every additional foot in depth. As regards standard rates of percolation 3 inches and over per hour is considered good, 1½ inches fair, and ¾ inch bad.

Retained Moisture.—The finer the sand the more moisture is retained in the pores of the charge, but even with comparatively coarse sand there is a zone at the bottom that will never drain even approximately dry. This zone or layer is usually about 6 inches in depth regardless of the depth of the tank, and would therefore suggest the superiority of deep tanks over shallow ones,

[1] Cyaniding Gold and Silver Ores, page 54 (Second Edition).

but against that is to be set the decreased rate of percolation as already noted and also the superior opportunities for aeration in a shallow charge. This wet layer at the bottom can be dried and aerated by the application of a vacuum under the filter after the solution has ceased to percolate by gravity. A vacuum used in this way is especially advantageous in the leaching of clean fine sand, which although it admits of rapid percolation by gravity yet retains a disproportionate amount of moisture which can only be displaced by pressure.

An important point in securing good leaching is to see that the mats and filter bottom are kept clean and thoroughly pervious to the solution. Sometimes the fine upper cloth may take on a deposit of lime carbonate, especially if aeration by compressed air is made use of, and in that case it will be necessary to clean it with dilute muriatic acid. Even under ordinary conditions the mats and the spaces underneath them tend in time to get blocked up with fine slimy material and require periodical cleansing.

The quantity of solution used during treatment varies from ¾ to 1½ times the weight of ore, according to circumstances, and has to be determined by the metallurgist from economical tests.

The solution strength is on the whole usually higher than is used in agitation processes, and various strengths are used on the same charge. When treating accumulated tailings, the common practice is to begin with one or two weak washes so that the cyanicides may expend themselves before the stronger solution is applied. With freshly crushed ore, especially if milling or collecting be done in cyanide solution, this is usually unnecessary and strong solution may be added at once, with the advantage of saving time and beginning the dissolving action as early as possible. The strength of this solution varies, but a concentration of 0.3% KCN or 6 pounds per ton of solution is probably used more often than any other.

The cyanide necessary to make up the stock solution to the desired strength is best added before the wash is applied to the sand tank. It is a common practice to hang cans containing lumps of cyanide in the stream of solution as it flows onto the sand, but this method is not recommended as it cannot produce a wash of uniform strength.

The most approved plan is to keep a solution tank expressly for strong washes. The solution therein is tested once or twice a day and the necessary quantity of cyanide placed in a basket hanging below the surface. When the lumps are all dissolved the contents of the tank are well mixed either by blowing compressed air into it for a short time or by circulating with a centrifugal pump. The resulting strong solution can then be pumped or run by gravity onto the sand charges needing it.

Draining and Washing.—When all the soluble precious metal has been dissolved the strong rich solution in the charge is displaced by repeated washes of weak barren solution and finally by a water wash. Julian and Smart's experiments[1] go to show that given a certain volume of wash solution available its displacing efficiency is considerably increased by dividing it up into a number of small washes rather than applying the whole of it at once.

The quantity of water given at the finish is determined by the amount of moisture held in the charge when drained and ready to be sent to the dump, because although it would be desirable to give sufficient to displace the whole of the cyanide solution held as moisture, this usually could not be done without increasing the total quantity of solution in stock, with the result that the surplus would have to be run to waste, consequently only such an amount of water is given as is just equal to the quantity of moisture sent out of the plant in the residue.

It is advisable to assay the effluent solutions frequently throughout the treatment, especially during the washing-out stage of the process so as to observe the progress of displacement of the value-bearing liquor and avoid losing dissolved gold in the residues.

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[1] Cyaniding Gold and Silver Ores, page 58 (Second Edition.).

Discharging the sand is done in various ways. Where water is plentiful sluicing is a cheap and effective method. If this is not available, the tanks are so constructed that cars may be run underneath and the residue shovelled into them through doors in the tank bottom. Where labor is expensive, the Blaisdell excavating machinery in conjunction with a conveyor belt and stacker is probably the most satisfactory way of disposing of the residue.

CHAPTER IV

SLIME TREATMENT

Many definitions of the word "slime" have been attempted but the one that best expresses its meaning from a practical point of view is that given by Park[1] that it is that part of the pulverized ore that is not percolable on a commercial scale without the use of pressure. The consequences of this definition are that slime cannot be treated commercially by the same methods as sand and requires entirely different handling. The fundamental constituent of slime is the amorphous or clayey matter present in all ores to a greater or less extent and this is supplemented by varying amounts of exceedingly fine granular material produced in the process of crushing or pulverizing.

For some years after the leaching process of cyaniding was well established no feasible method was discovered for treating the slime which had been separated from the percolable part of the pulp: consequently the object of the metallurgist was to avoid as much as possible the formation of slime in milling, and in so doing there was a tendency to produce a pulp whose maximum sized particles were too coarse to liberate the precious metals and expose them to the action of the cyanide solution. With the development, however, of successful methods of treating the slime, the practice in milling was aimed more and more at the production of a pulp fine enough to yield the maximum extraction by cyanide.

This principle developed rapidly until the process of "all-sliming" came to be considered as the panacea for all the ills of the cyanide process and in discussing the erection of a new mill it was considered almost heresy to suggest the possibility that "all-sliming" might not be the most commercially profitable method of treating the particular ore in question. The expres-

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[1] The Cyanide Process, page 129 (Fifth Edition).

sion "all-sliming" does not usually mean grinding everything to a "slime" in the sense of the definition already laid down, but merely grinding sufficiently fine to avoid mechanical difficulties in the treatment of the whole of it by slime methods. In the majority of instances the product of "all-sliming" contains from 5% to 35% of sand that will not pass a 200 sieve; moreover, a good proportion of the sand −200 sieve is perfectly leachable if properly freed from slime, as has been already explained, so it is fairly safe to say that of the usual "all-slime" pulp from 30% to 50% does not come under the terms of the definition of slime, that is, it would be perfectly practicable to leach it by gravity on a commercial scale.

In many cases it may be just as effective and more convenient to consider the ground product all as "slime" and deal with it as such, but the writer has shown instances where it was more profitable to make a separation and treat the percolable part by leaching. For example, in one "all-sliming" mill the plant capacity restricted the agitation period of each charge to 24 hours; this was sufficient to give a good extraction on the true slime, but the fine granular material required considerably longer. Here the general extraction was improved, the tonnage increased, and the costs lowered by erecting a subsidiary leaching plant and removing as a percolable product about 40% of the finished pulp.

The question of "all-sliming" any given ore should be determined solely by commercial considerations. Extraction tests will, of course, be made but it will not be sufficient for these to show an increased extraction; the amount of such increase must be weighed against the increased cost of treatment and an estimate made of the probable net gain, if any. Even when all-sliming is indicated it usually will not be profitable in practice to reduce the entire ore to accord with the definition of slime. In pulverizing an ore the first thing to separate as a finished product is the natural slime, next the most friable portions will yield to disintegration, and so on, until there is left a residuum of the toughest and most refractory material. This latter will circulate round and round the milling system, only a small portion being eliminated each time as finished product. The cost of reducing this residuum is disproportionately high and in many instances will not result in a final profit.

It sometimes happens too that this tough residuum contains a lower assay value than any other portion of the ore, making it still more unprofitable to carry the process to a conclusion.

The amount of oversize finally allowed to pass to treatment will determine the point as to whether it is worth while to separate a part to be leached or to treat the whole by agitation. As a rule when the amount of coarse material is not sufficient to interfere with the mechanical working of the agitators and yields its maximum extraction within the time available, it will pay better to treat the whole as if it were slime.

What has been said thus far deals with those ores on which all-sliming has been indicated as the ideal treatment. Statements may often be read to the effect that while gold ores can be treated advantageously by a combination of leaching and agitation, for silver ores all-sliming is the only way to obtain a satisfactory extraction. Like most generalizations, these statements are to be received with caution. While it is probably true to say that most silver ores require finer grinding than the majority of gold ores, yet in the writer's experience the silver ores that unquestionably need "all-slime" treatment are in the minority. Very many of them show no appreciable gain in extraction when the −100 mesh mark has been passed, and with some the limit may be put at −80 or even coarser. In cases such as these it is difficult to see any point in trying to make an "all-slime" product. If the costs are increased (as they undoubtedly will be) without resulting in any increased recovery, the commercial result will be a loss.

Even when the finer comminution does show a substantial increase in extraction the coarser grinding may result in a greater net profit on account of lower costs and larger output from any given crushing unit.

If the degree of comminution decided on results in a pulp 50% of which will remain on a 200 sieve, separation of the sand for leaching will almost always be advisable. Early in the development of the cyanide process agitation methods were tried for sand, but were abandoned on account of the expense and mechanical difficulties involved, and the same objections hold good today, though perhaps not quite to the same extent.

Coagulation of the Slime for Settlement.—Since the slime treatment is conducted at a much lower ratio of liquid to solid than that usually employed in the operations of milling and classification there must be a dewatering of the slime between the mill and the cyanide plant. Such dewatering is almost always effected by settlement and decantation but owing to the invariable presence in the pulp of amorphous or clayey matter, it is impossible to obtain a settlement accompanied by a clear effluent liquor without the presence of some substance which will cause a coagulation or flocculation of this clayey material. Such substances or the solutions resulting from them are known as electrolytes, in view of the electrical nature of the phenomena exhibited by finely divided matter in suspension.

Action of Electrolytes.—The substance most commonly used for this purpose is lime both on account of its cheapness and also because of its usefulness as an alkaline protector for the cyanide. Many other substances have a similar effect though varying in degree, such as alum, calcium carbonate, sulphuric acid, ferrous and ferric sulphate, and calcium sulphate. Lime usually forms a coarser grained and more flocculent curd with a more brilliantly clear supernatant liquor than any other electrolyte, but the precipitated mass does not settle as densely as that produced by some of the others. Caustic soda in some cases acts as a coagulant and while the liquor produced is not quite as clear as that resulting from lime, the curd has a finer grain and settles more slowly but much more densely. Sodium carbonate, on the other hand, tends to prevent coagulation and renders clarification almost impossible.

Addition of Lime.—When milling is done in cyanide solution the whole of the lime necessary is usually added to the ore before it enters the mill; by this means it gets pulverized with the ore and performs its function of protective alkali. Milling in water is (1919) seldom employed except when it is desired to pass the pulp over amalgamated plates before cyanidation. Even in these circumstances the lime is often added in the mill, notably in the Rand practice. When the method was first adopted there it was strongly objected to by the mill men on the ground that it hardened the amalgam. As soon, however, as they got accustomed to the changed conditions, it was found that no detrimental results followed; in fact, in some cases a marked increase occurred in the amount of gold recovered on the plates due to an improved amalgamation of the fine gold in the slime. As a consequence certain slimes which it had been designed to treat by cyanide were now found to be low enough in gold content to be run to waste. In other parts of the world the plan of amalgamating in lime water has not commended itself after fair trial and the necessary quantity of lime is added (usually in the form of milk of lime) to the ground ore after amalgamation, or even later on, as at the Homestake, after the separation of sand and slime.

The amount of lime added varies of course with the ore, but for satisfactory settlement of the slime a strength in the solution of 0.02% CaO is usually sufficient. When milling in cyanide the quantity of lime maintained in solution is regulated rather with a view to protective alkalinity and is usually considerably in excess of the bare amount needed for slime settlement.

Thickening the Pulp.—The slime pulp on leaving the sand classifier with its proper admixture of lime is first led to some form of thickening device if, as is usually the case, the ratio of liquid to solid is too large for economic handling in the cyanide treatment. The form of thickener almost universally used in the early days consisted of a large spitzkasten composed of a number of pointed boxes arranged as a unit in such a way as to give a continuous superficial area.

At the Minas Prietas plant of Chas. Butters and Co., capable of treating 9000 tons of mixed sand and slime tailings per month, the slime spitzkasten was 36 ft. by 42 ft. composed of 42 pointed boxes 6 ft. square and 4′ 9″ deep; at the apex of each box was a nipple and hose discharging into a common launder. The slime pulp entered by a launder stretching from end to end along the

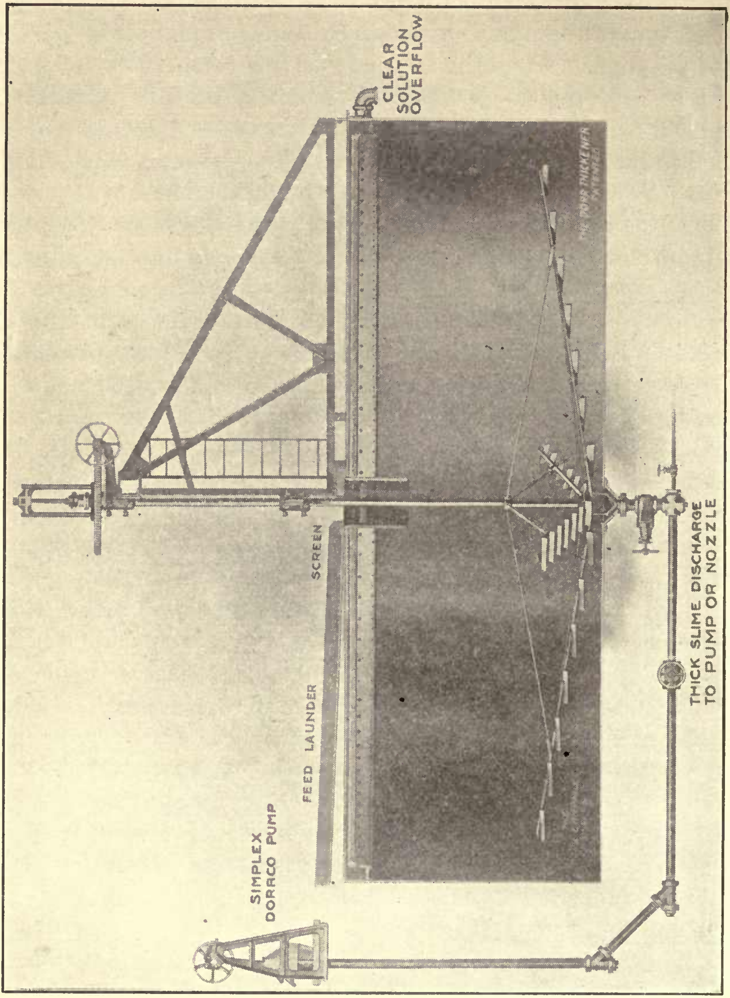

Fig. 11.—The Dorr Thickener.

centre line with small holes bored at intervals for the exit of the pulp, and the clear solution overflowed a weir at both sides, so that on each side of the feed launder there were three rows of seven boxes for the settlement of the slime. The overflow weirs were raised sufficiently to submerge the partition walls of the various compartments to a depth of nine inches, giving a continuous surface area of solution of 1512 sq. ft. and a depth of 5′ 6″ from the apex of each compartment to the surface of the solution. The apex discharges of the two rows nearest the intake were allowed to flow continuously, the discharges of the second rows were closed off part of the time and the third rows were only opened for a few hours each day, the general regulation being so arranged as to give as thick a pulp as possible in the underflow while maintaining a clear overflow for return to the mill (or in this particular instance, to the pulping tanks).

This thickened pulp was at times further dewatered in a collecting tank, usually a large circular tank with peripheral overflow and fitted with mechanical stirring arms. The pulp enters at the centre and when the tank is full the slime settles and a clear liquor overflows into the ring-launder to be returned to the mill. The pulp continues to flow in until the tank contains sufficient slime to constitute a treatment charge, when the stream is diverted to another collecting tank.

A very efficient modification of this type of collecting tank is the Dorr thickener. The entry of pulp and overflow of clear liquor are the same but in the Dorr system the thickened product is continuously removed from the bottom by means of slowly revolving rabbles which convey it to the centre where it falls into a discharge pipe whose aperture can be adjusted to regulate the speed of outflow. The device works smoothly and continuously and has superseded all the older methods for thickening slime pulp.

Regarding the capacity of Dorr thickeners J. V. N. Dorr in a paper presented to the American Institute of Mining Engineers, August, 1914, says,

> "The capacity of Dorr thickeners has been found to be primarily a function of area, although the depth of the tank has an influence depending on the dilution of the feed and the dilution of the underflow

desired. With a given area and depth and a very dilute feed and underflow, the capacity depends on the amount of liquid that can be clarified; *i.e.*, additional solids, but no additional liquid, can be added to a tank already fed to capacity without overflowing slime. On the other hand, with a feed perhaps 8 of liquid to 1 of solids and a thick discharge of 2 to 1 or less, it will be found usually that additional liquid can be added to a thickener operating at capacity without overloading it, but any addition of solids will cause slime to overflow."

In actual practice the area per ton of solids thickened per 24 hours averages about 4 sq. ft. for an ordinary silicious ore slime, but may rise as high as 10 to 15 sq. ft. in the case of light argillaceous slime.

The subject of the rate of settlement of slimes and the calculation of the area and depth required has been developed in considerable detail in a paper by H. S. Coe and G. H. Clevenger.[1] Their investigations show that slime pulp in process of settling passes through two main phases, that of "free settling" and that of "compression," the two zones being separated by a transition zone.

By the phase of "free settlement" is meant that condition of density in which the flocks or curds of colloidal matter that are produced by the electrolyte are sufficiently dispersed to settle by gravity unhindered through the liquid in which they are suspended, such settlement taking place at a constant or gradually diminishing rate until the critical point is reached, at which point the phase of compression begins. The "hindered settling" or compression phase is that in which the flocks have drawn so close together that no further settlement can take place without a compression of the flocks themselves, the liquid displaced in the process having to force its way out of the compression zone by forming channels therein leading up into the higher zones. It is thus apparent that in calculating the cubic capacity of tank necessary for any given slime these two separate factors have to be taken into account and separately determined.

In regard to the determination of the critical point dividing

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[1] *Transactions* A. I. M. E., Sept., 1916.

free settlement from compression, it is not always easy to reach a conclusion. Indications of the latter phase are a marked retardation in the settlement rate and the formation of channels of clear solution through the mass of solids. Referring to the characteristics of the pulp while still in the free settlement zone the authors of the paper referred to say "ordinarily the latter point is indicated by the evenly flocculated appearance of the pulp surface, without channels of fluid coming through, and by the uniform texture of the pulp as seen in a glass cylinder."

For determining the settling area necessary the Dorr Company gives the following outline instructions adapted by members of its staff from the paper referred to. It should be remembered, however, that the conditions of different ores are so varied that it is difficult to give a short and concise method of procedure applicable to all cases. Two rules should be carefully observed: one is that the sample should not be allowed to dry either by heat or by atmospheric evaporation between the time it is taken and the time the tests are made: the other is that to insure representative results it is not sufficient that the ore should be crushed to the desired fineness but that it should also be subjected to conditions of agitation similar as regards time and method to those which are to obtain on the milling scale.

"When a mixture of liquid and solids is fed to a Dorr Thickener at a certain dilution say 5 : 1 (liquid : solids) and the discharge at the bottom contains 50% moisture or a dilution of 1 : 1, the pulp in settling from 5 : 1 to 1 : 1 passes through all the intermediate dilutions 4 : 1, 3 : 1, 2 : 1.

"At different depths in the Thickener there will be different dilutions or zones of pulp. One of these zones will have a smaller capacity and so require a larger settling area than any of the others and so the amount of solids fed to a given tank cannot exceed the capacity of that slowest zone and the area must not be smaller than the area necessary for the slowest zone.

"To test for the size of a Thickener unit then, pulps are made up of different dilutions between the dilution of feed and the dilution of discharge. These pulps are put in 1000 cc. graduates and shaken. The rate of settlement of each pulp for 10 minutes is noted and these observations checked by shaking and settling again.

"Having obtained these dilutions the area required per ton of solids is calculated by the formula

$$A = \frac{1.333\,(F - D)}{R}$$

"Where A is the area in sq. ft. per ton of solids per 24 hours, F is the ratio of solution: solids which settle at the rate of R feet per hour and D is the ratio of solution: solids in the discharge required from the Thickener. 1.333 is $2000 \div (62.5 \times 24)$ which is a factor necessary in order to get the desired units.

"F and R are determined then for each graduate and the area required for each dilution calculated. The largest calculated area will be the unit area necessary and this multiplied by the tons of solids to be thickened per 24 hours will give the total settling area necessary."

(1) Take 1000 cc of pulp feed to Thickener.

A. Put above in a 1000 cc graduate.

Shake.

Allow to stand and note readings between pulp line and clear solution.

At end of shaking.............................. 1000 cc
After 2 minutes.............................. —— cc
After 4 minutes.............................. —— cc
After 6 minutes.............................. —— cc
After 8 minutes.............................. —— cc

B. Decant 150 cc of solution.

Shake.

At end of shaking.............................. 850 cc
After 2 minutes.............................. —— cc
After 4 minutes.............................. —— cc
After 6 minutes.............................. —— cc
After 8 minutes.............................. —— cc

C. Decant 150 cc of solution.

Shake.

At end of shake.............................. 700 cc
After 2 minutes.............................. —— cc
After 4 minutes.............................. —— cc
After 6 minutes.............................. —— cc
After 8 minutes.............................. —— cc

Weight of dry solids in graduate ————

Number of cc corresponding to 1 foot of length of graduate ————

(2) Take 1500 cc of pulp feed to Thickener.

Allow this to settle and decant 500 cc of clear solution.
Put remainder in 1000 cc graduate.
Shake and on standing note readings below.

A. At end of shaking.......... 1000 cc
After 2 minutes.......... —— cc
After 4 minutes.......... —— cc
After 6 minutes.......... —— cc
After 8 minutes.......... —— cc

B. Decant 150 cc.

At end of shaking.......... 850 cc
After 2 minutes.......... —— cc
After 4 minutes.......... —— cc
After 6 minutes.......... —— cc
After 8 minutes.......... —— cc

C. Decant 100 cc.

At end of shaking.......... 750 cc
After 2 minutes.......... —— cc
After 4 minutes.......... —— cc
After 6 minutes.......... —— cc
After 8 minutes.......... —— cc

Weight of dry solids in graduate ——
Number of cc of dry solids corresponding to 1 ft. of length in graduate ——

(3) Take 2000 cc of pulp feed to Thickener.

Allow this to settle and decant off 1000 cc of solution.
Put remainder in 1000 cc graduate.
Shake and on standing note readings below.

A. At end of shaking 1000 cc
After 2 minutes.......... —— cc
After 4 minutes.......... —— cc
After 6 minutes.......... —— cc
After 8 minutes.......... —— cc

B. Decant off 150 cc of solution
Shake.

At end of shaking.......... 850 cc
After 2 minutes.......... —— cc
After 4 minutes.......... —— cc
After 6 minutes.......... —— cc
After 8 minutes.......... —— cc

C. Decant off 100 cc of solution
Shake.

At end of shaking	750 cc
After 2 minutes	—— cc
After 4 minutes	—— cc
After 6 minutes	—— cc
After 8 minutes	—— cc
Weight of dry solids in graduate	———
Number of cc corresponding to 1 ft. of length of graduate	———

It will be noted that the successive decantations of clear portions of the solution are a convenient method for making the pulps of different densities.

The ratio of liquids to solids at each density is found by calculation from the specific gravity of the dry slime which must of course be determined. For instance, in the case of (1) *A*: assume that when the reading is taken the line of pulp has settled to 800 cc and that the weight of dry solids present found by subsequently drying the whole is 190 grams. Then if the specific gravity of the dry slime be taken at 2.6 and that of the liquid as 1 the ratio of liquid to solid in the 800 cc of pulp in the cylinder will be found as follows: the 190 grams of slime being 2.6 times heavier than water will occupy the space of $\frac{190}{2.6} = 73$ cc of water in the cylinder: then 800 cc of water less 73 = 727 cc, so that at the 800 cc mark there are 727 grams of water and 190 grams of dry slime, the ratio of the two being 727:190::3.8:1.

Note.—After each shaking it is well to allow the cylinder to stand for from three to five minutes before beginning to time the settlement so as to allow the colloids time to assume the flocculated condition which allows settlement to begin. It will often be found that if the timing is started immediately after shaking the settlement during the second five minutes will be considerably more rapid than during the first five minutes, the second reading being therefore more representative of working conditions.

The second or compression phase of settlement cannot be determined by this formula so it is very important that the foregoing tests should not be carried beyond the critical point or the end of the phase of free settling.

Determination of *D* (the ratio of solution to solids in the discharge required from the thickener). There is a point of maximum density beyond which any given slime will not settle and it is of course useless to require a discharge of greater density than this. In the Dorr Company's instructions already quoted this point is determined as follows:

"The dilution of discharge "*D*" that can be expected may be determined by allowing a pulp of consistency about 3:1 to stand about 48 hours in a graduate stirring gently with a glass rod every four or five hours." For instance,

" Take 1000 cc of pulp at approximately 3: 1.
Shake.
Allow to stand.

At end of shake	1000 cc
After ¼ hour	—— cc
After ½ hour	—— cc
After ¾ hour	—— cc
After 1 hour	—— cc
After 1½ hours	—— cc
After 2 hours	—— cc
After 4 hours	—— cc
After 8 hours	—— cc
After 12 hours	—— cc
After 16 hours	—— cc
After 20 hours	—— cc
After 24 hours	—— cc
After 36 hours	—— cc
After 48 hours	—— cc

Note.—After each reading decant as much clear solution as possible.

Dry weight of solids in graduate ——

Number of cc corresponding to 1 foot of length of graduate ——

Then if the slime needs 36 hours to attain its maximum density or the density required, the cubic capacity of the thickener must be sufficiently large to retain any given portion of the pulp in that tank for the period ascertained as necessary by the test.

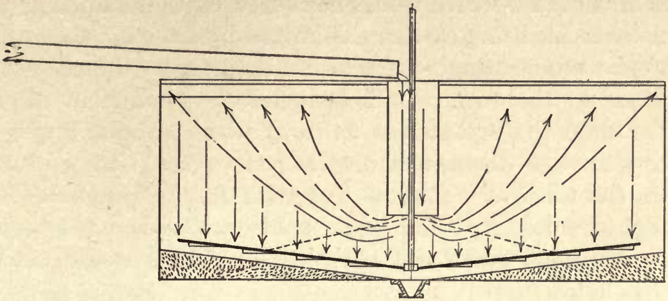
The following example is given by Coe and Clevenger on a pulp that required 4.31 sq. ft. of area per ton of solids per 24 hours for free settlement. A compression test was made in a cylinder 12 in. deep with a 3 to 1 pulp, giving these readings:

Time of thickening, hours	Consistency fluid to solid
2	1.70:1
4	1.59:1
9	1.35:1
14	1.20:1
19	1.12:1

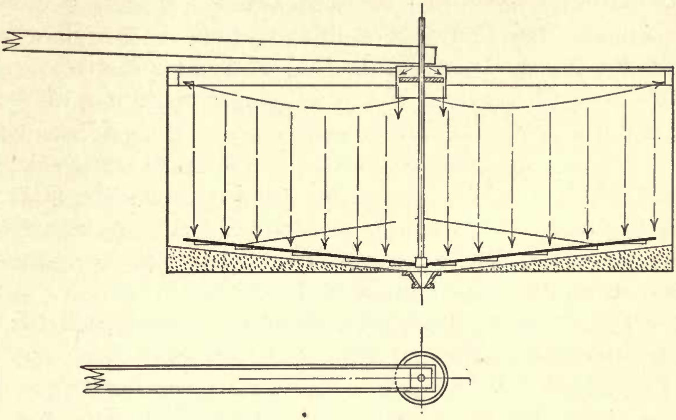
"Since an area of 4.31 sq. ft. is required per ton of solids per 24 hours the total solids per square foot retained in the thickening zone must be $\frac{19 \times 2000}{24 \times 4.31} = 367$ lb. or 19 hours supply per square foot. There is required a 5-hour supply of each of the pulp consistencies, 1.16 to 1, 1.275 to 1, 1.47 to 1, and a 4-hour supply of a 1.7 to 1 pulp.

"The solids per cubic foot in the above pulps are 43.2 lb., 37.6 lb., 33.7 lb., and 30 lb., respectively. The depth of each class of pulp would therefore be 2.23 ft., 2.57 ft., 2.87 ft., and 2.58 ft., or a total depth of 10.25 ft. To this depth must be added a foot for the loss due to the pitch of the drags in the thickener and 1.5 ft. for the depth of the feed, since the feed is thick and the volume will be proportionately low. The total calculated depth of the tank would be 12.75 ft. If proper allowance were made for storage capacity the tank might be inconveniently deep. In this case it would be better practice to make the tank 12 ft. deep and make various allowances for additional capacity by increasing the diameter of the tank to give a 30 per cent. increase in area."

In regard to the practical operation of Dorr thickeners the writer has observed in some instances a tendency on the part of the operator to extend the vertical intake cylinder down from ⅔ to ¾ of the depth of the tank, under the impression that the settlement would be helped thereby, the idea apparently being that if the slime is fed to the lower part of the tank it will stay there. This practice, however, defeats its own object and changes the apparatus from a settler into a classifier. The reason of this will be seen from the diagrammatic sketches, Figs. 12 and 13. Fig. 12 illustrating the incorrect arrangement shows how the deep cylinder destroys the tranquility of a large part of the tank



INCORRECT METHOD



CORRECT METHOD

Figs. 12 and 13.—Sketch of Dorr Thickener.

charge causing upward currents which tend to raise and carry off the more flocculent material, and reducing the available settling capacity of the tank. The action may be compared with that of a spitzlutte, wherein the further the baffle-plate is lowered the greater is the velocity and lifting power of the current in the lower part of the box. Fig. 13 shows the more approved arrangement in which the intake cylinder extends down only from 12 in. to 18 in. below the surface or just sufficient to prevent the surface disturbance which would be caused by the inflowing pulp, such disturbance being further minimized by allowing the stream to fall onto a deflecting board, either floating on the surface or fixed close to the surface. This deflector is made of smaller diameter than the cylinder so as to allow an annular space for the pulp to pass downward into the charge. It may also be made in the form of a shallow tray with holes bored all around the vertical sides, and this plan works well when a screen is used to intercept waste and other matter which would tend to block the holes.

Slime Extraction Treatment.—There are two different principles applied to the treatment of slime, (1) agitation, (2) leaching in thin cakes by pressure. Of these two the first is in almost universal use. One of the earliest attempts to cyanide slime was that of Charles Butters who devised a combination treatment whereby the pulp was agitated by stirring arms in a tank fitted with a filter bottom. When the dissolution of the gold was complete, the charge was allowed to settle, and while the clear solution was being decanted a vacuum was applied under the filter to reduce the moisture in the mass of settled slime. The latter feature was, however, soon abandoned and simple agitation, settlement, and decantation resorted to.

John R. Williams in his system of agitation dispensed with the stirring gear and substituted a method of transfer from tank to tank in contact with cyanide solution by means of centrifugal pumps. His tanks were made with a moderately conical bottom connected at the centre with a centrifugal pump. When a charge of thickened slime had been collected and the surplus water decanted the pump was started and at the same time a stream of weak solution was played on top of the charge diluting the pulp and washing it into the pump which delivered into a second similar tank. If this treatment was insufficient to dissolve all the gold, the operation could be repeated as often as necessary.

Both the Butters and Williams methods gave good results on the Rand and were developed simultaneously.

Aeration.—In the treatment of the accumulated slime from the storage dams on the Rand it was soon found that a very poor extraction was obtained by the methods that were perfectly satisfactory for the fresh or current product. This was traced by W. A. Caldecott[1] to the formation in the dams of products of partial oxidation, among which were ferrous sulphate, ferrous sulphide, and sulphuretted hydrogen: these being strong reducing agents abstracted all the free oxygen from the solution and checked the dissolution of the gold. When, however, they were thoroughly oxidized by blowing air through the pulp before it came in contact with cyanide, a normal extraction was obtained in the subsequent cyanidation.

To introduce the necessary air into the pulp H. T. Durant inserted a small automatic valve into the suction of the centrifugal pump so that by slightly choking the flow of pulp in the main suction line the little "snifter valve" would admit a continuous current of atmospheric air. This air in passing through the rotor of the pump was broken up and atomized in such a way as to render it very effective for the purpose in view.

In practice, the snifter value needed constant attention to secure a steady supply of air and for mechanically stirred tanks a widely adopted substitute, in cases where an additional amount of aeration is needed, consists in fitting a small air lift against the inner side of the tank. This gives excellent aeration, rarely gets out of order, and consumes very little power. One length of six-inch pipe discharging through a tee or ell just above the surface of the pulp is usually sufficient to aerate a charge of 80 to 100 tons of slime with a three to one dilution.

Agitating Devices.—Although the old form of mechanical stirring ag tator is (1919) rarely installed in a modern plant it is not to be despised being a simple and efficient device for agitating slime pulp. The main points to be observed in the construction are (1) to see that the gears and clutches are sufficiently heavy,

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[1] *Proceedings* Chem., Met. and Min. Soc. of S. A., July, 1897.

(2) that the paddles or arms are set sufficiently high above the bottom to be above the level of any mat of sandy material that may separate out and collect on the bottom when agitation is stopped for settlement and decantation, and (3) that the speed of the paddles is sufficient to pick up and pulp all such settled sand when the agitator is again set in motion. For this purpose the

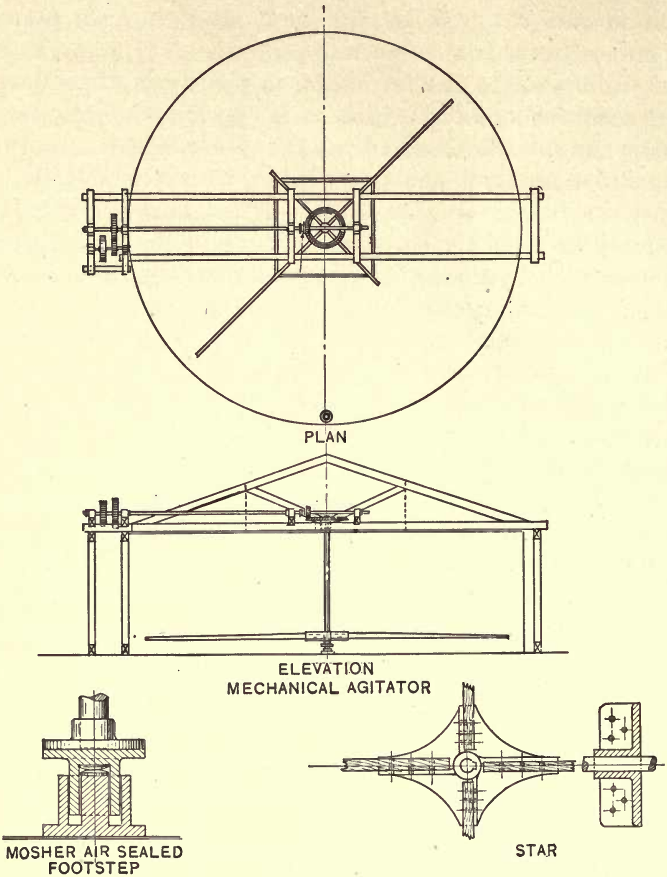

Fig. 14.—Original Mechanical Agitator.

speed should be not less than 700 feet per minute at the tip of the paddles, which should extend out as near as possible to the periphery of the tank. With these precautions the mechanical agitator will give good service with a minimum of trouble and repairs, even when treating pulp containing a considerable proportion of sandy material. The power consumption, including line shafting, works out at from five to seven h.p. for a 100-ton charge, and from ten to twelve h.p. for the first fifteen minutes after starting.

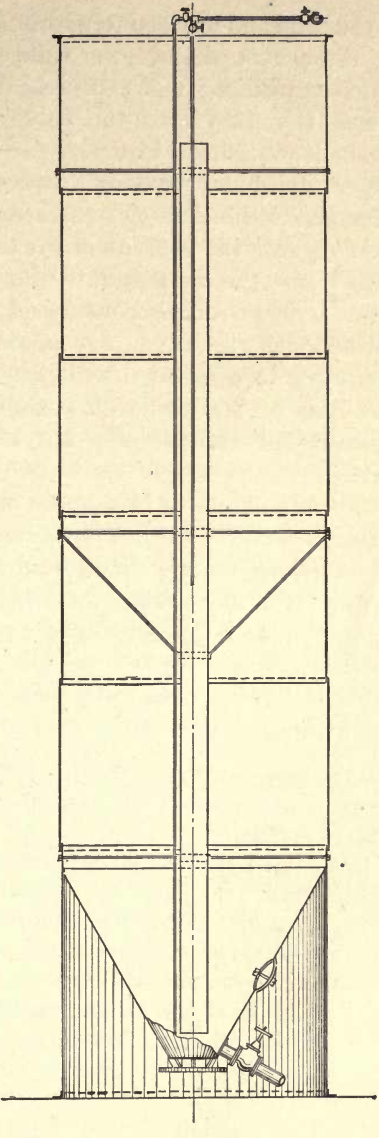

FIG. 15.—Brown or Pachuca Tank.

The next important development in methods of agitation was the Brown Agitator commonly known in America as the Pachuca tank. It consists of a vertical cylinder whose height is about five times its diameter and whose bottom is formed of a cone of 60 degrees slope. Braced vertically in the centre of this cylinder is a smaller one whose diameter is 1¼ inches for every foot diameter of the larger one. This small cylinder is open at each end and extends downward to within 18 inches of the apex of the cone and upward to within 18 inches of the top of the tank. At the centre of the lower orifice of this column is a small pipe which admits a jet of compressed air, forming an air lift which draws the pulp into the column and elevates it to the top, where it overflows into the surrounding space formed by the larger cylinder, causing a continuous vertical circulation throughout the time of treatment.

As regards the position of the air jet the most modern practice is to insert the air pipe into the top of the central column and carry it down only about ⅓ of the depth of the tank. This method has the advantage of reducing the initial air pressure necessary to operate the air lift and also eliminates trouble in blocking of the orifice, a trouble which in the original plan necessitated the use of a self closing ball valve or some such device.

In some plants the Pachucas are operated without the central column, but the method does not seem to have much to recommend it, as there is little doubt from figures obtained by the writer that the circulation and admixture of the pulp is less complete, and the consumption of air greater than when the column is used.

The following data are given by F. C. Brown in an article in the Mining and Scientific Press of September, 1908.

"The following figures relating to the air and power required are from actual working conditions extending over long periods, ranging from 6 to 18 months:

1. A tank 7 ft. 6 in. diameter by 37 ft. was used for treating slime, and required from 4 to 6 cu. ft. of free air per minute, at a pressure of 22 lb. per square inch. The charge of slime was 15 tons (dry weight), and the horsepower was from one-third to one-half.

2. The same size of tank, treating finely ground concentrate, requires 15 to 20 cu. ft. of free air per minute, at a pressure of about 26 lb. per square inch. This quantity of air gives a thorough agitation with a charge of concentrate of 35 to 40 tons (dry weight), with 1½ to 2 horsepower.

3. An installation of 10 tanks, each 10 by 39 ft., treating slime, requires 88 cu. ft. of free air per minute, at a pressure of 23 lb. per

square inch. Each tank holds a charge of 35 tons (dry weight), and requires ¾ hp. per tank.

4. Two tanks, 13 by 55 ft. treating slime, require 32 cu. ft. of free air at a pressure of 33 lb. Each tank holds 110 tons dry slime. The horsepower consumed is 1¾ per tank."

While the Brown tank is a simple and effective appliance and is capable of good work when its idiosyncrasies are understood, the disproportionate amount of head room needed has always been considered one of its drawbacks. Moreover, it is not suitable for settlement and decantation processes and when thickening of the pulp is necessary after agitation it is usually done in subsidiary thickeners. Another point that has always seemed to the writer a defect in the system is that since air is the sole instrument of agitation a certain minimum amount cannot be dispensed with regardless of whether such amount is necessary or desirable for the dissolving treatment or not. In many cases the minimum quantity of air necessary for agitation is many times greater than that needed chemically and the surplus may act adversely in consuming cyanide and lime or may even produce undesirable changes in the ore itself. Still, in spite of all objections the Brown tank is a valuable development in the treatment of slime and its use (1919) has extended to all parts of the world. The loss of head room involved in using the Brown tank for individual charge treatment is obviated by the "continuous method" in which the pulp leaves the series of tanks only a few feet lower than the level at which it enters. This is described more fully in the section on continuous treatment.

A more recent agitation system and one that, perhaps, has less defects than any other so far devised, is the Dorr agitator. The main objection to it, if objection it be, is the fact that it is not suitable for the charge system of treatment and is almost exclusively employed for the "continuous method" which will be described later. It is essentially a development of the Dorr thickener, the difference being that the rakes revolve at from one to three times per minute instead of once every five or ten minutes. The central shaft consists of a pipe which is made to do duty as a small air lift, discharging the pulp into a pair of launders attached to the mechanism and revolving with it. These launders are perforated at intervals so that the pulp is deposited over the whole surface of the tank very much in the manner of the Butters distributor.

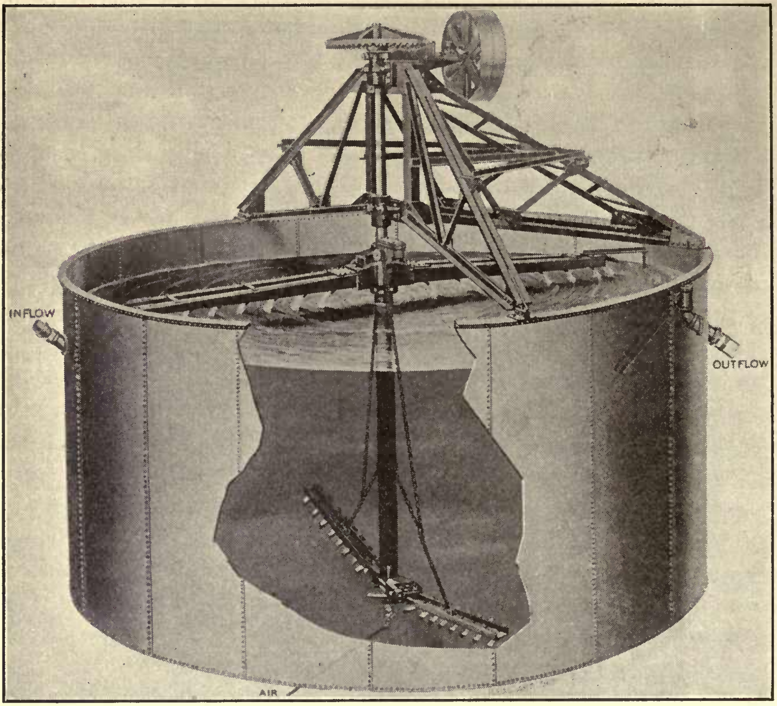

Fig. 16.—Dorr Slime Agitator.

The Dorr agitator is operated as follows: the pulp, of the required density, flows in continuously at a point in the periphery, and is led by a downcast launder almost to the bottom. There being no agitation of the charge as a whole, the solids tend to settle and distribute themselves over the bottom, where they are slowly scraped by the rakes to the centre and elevated in the form of fluid pulp to the revolving distributor launders and deposited on top of the charge, again sinking down to the bottom to be subjected to the same action. In this way a vertical circulation of the ore particles through the comparatively stationary solution takes place giving (at least theoretically) a more rapid individual motion relatively to the solution than occurs in systems where the whole pulp is in agitation simultaneously. While this is going on a continuous stream of pulp is flowing out at a point on the top of the tank opposite the inflow and maintaining a constant level of pulp in the tank.

Another feature claimed by the inventor is a "selective agitation" of the more sandy part of the pulp, the theory being that the larger and heavier particles, those therefore that require longer time for treatment, sink more rapidly and circulate more often than the lighter particles and also remain longer in the tank. The latter point is a little puzzling when it is remembered that just as much coarse material must be found in the outflow as there is in the inflow in order to prevent a gradual but continuous increase of sand in the tank, but this is not incompatible with there being continuously present in the tank a larger percentage of sand than is to be found either in the inflow or outflow. It is difficult to say how much practical importance is to be attached to this phenomenon but that it has an actual existence is claimed to have been sufficiently proved. According to Dorr the mechanical power taken varies from ¼ to 2 or 3 hp. according to the size of the tank, the speed, and the nature and dilution of the pulp handled. The air required is from 8 to 40 cu. ft. per minute at 10 to 20 lb. pressure.

Among other devices for agitation which have attained to some prominence are the Trent agitator, the Parral tank, the Hendryx agitator, and the Devereux agitator. The first of these consists of a submerged mechanism somewhat on the principle of the Butters distributor. A watersealed chamber on ball bearings is placed on the bottom of the tank at its centre and several pipes of different lengths radiate from it, whose ends are turned at a right angle. A centrifugal pump is placed outside the tank with its suction connecting with the pulp near the surface and its discharge entering the central chamber and finding its way out through the radial pipes, the tangential force causing the whole to revolve slowly. One of the chief objections lay in the inaccessibility of the mechanism in the event of stoppages or blocking of pipes, but this has been largely overcome by J. A. Carpenter who has devised a method of elevating the whole of the moving parts for repairs by means of a tackle and replacing them without emptying the tank.

The Parral tank, devised by Bernard MacDonald and named after the town in Mexico where it was first used, is the result of a desire to apply the good points of the Pachuca to a tank of more normal proportions and less head room consumption. In any tank of ordinary dimensions are placed 4 air lifts spaced at equal distances around the periphery and discharging tangentially through a tee or ell, so that the flow will be in the same direction. This action of the discharges imparts a rotary motion to the pulp which it is claimed is sufficient to hold in suspension an admixture of a considerable amount of sandy material. The centre being a dead point there is, of course, a tendency to build up a cone of solid material there and in more recent installations the tank is fitted with an artificial cone to take the place of this accumulation of solids.

The Hendryx agitator is somewhat like the Pachuca except that instead of using an air lift, the ascending motion is imparted to the central column of pulp by means of mechanically driven screw propellers.

The Devereux Agitator is operated simply by a comparatively small propeller traveling at a high speed. The tank used is of the usual type, but with a slightly conical bottom. The following dimensions are given in an article by Roscoe Wheeler in the Engineering and Mining Journal of July 15th, 1916; a 34-foot diameter tank with 5-ft. propeller revolving at 80 to 100 r.p.m. handles from 80 to 165 tons of dry slime at a charge with a dilution of 2 to 1. The maximum horsepower under these conditions is 12.

A 30-ft. diameter tank with 4-ft. propeller revolving at 80 to 100 r.p.m. at a maximum horsepower of 10 will handle 138 tons of dry slime at a dilution of 1.7 to 1.

It is claimed that there is no dead point in the tank and therefore no deposit of slime in any part of it and that there is no difficulty in getting the whole of the charge mixed and in circulation in a short time even after a prolonged shut down. An important feature is said to be the baffle which is used to break the vortex which would otherwise retard the velocity of the pulp and decrease the scouring action. Sufficient air is said to be drawn into the charge by the motion of the pulp to supply all the oxygen needed for chemical purposes, the amount being regulated by adjustments of the baffle. A subsidiary baffle for use

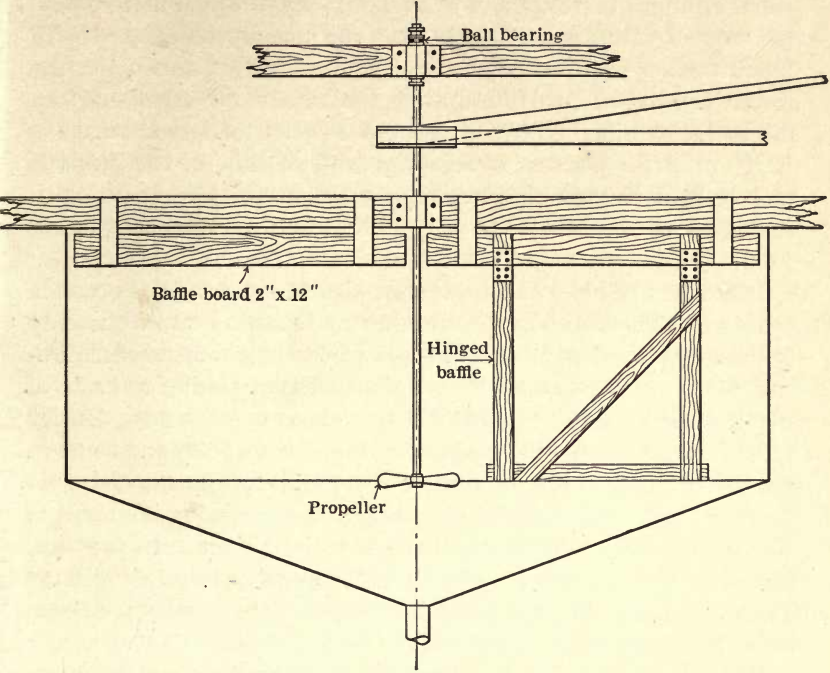

FIG. 17.—Devereux Agitator.

during charging is provided, the same operating on hinges and being drawn up out of the pulp as soon as the tank is full.

Cyanide Strength.—The best conditions under which to conduct the treatment can only be determined for each ore by careful experiment both in the laboratory and later in the working plant. Cyanide strength varies greatly; on the Rand the most usual strength is from 0.005% to 0.02% KCN or 0.1 lb. to 0.4 lb. per ton of solution. All-gold ores can usually be treated with much weaker solutions than silver-gold and silver ores. For the latter the writer has found that the maximum extraction can rarely be obtained with a treatment solution of less than 0.1% KCN or 2 lb. per ton of solution, while if time of treatment is unduly limited, even higher strengths are advisable. In Pachuca strengths of solution up to 0.4% KCN and over are frequently used.

Dilution of Pulp.—On this point also wide variations occur in practice. Theoretically the thinner or the more dilute the pulp is the more rapid will be the action on the precious metals. Experiments by Julian and Smart[1] showed that taking the rate of dissolution of gold in cyanide solution alone as 100 a pulp density of 6 of solution to 1 of slime gave a rate of only 58.6, and so on in gradually diminishing amounts till a 1 : 1 dilution showed only a rate of 20. They account for this by the retarding influence of viscosity on the movement of the K and CN ions. In practice, however, the increased cost of handling and precipitating large volumes of solution has to be offset against the increased extraction and an economic basis arrived at.

In any case the theory above stated does not always work out in practice, and at present the ratio of solution to ore in greatest favor lies between 1¼ and 2 to 1.

It may be noted that the pulp from an all-agitation mill, that is, a pulp with a high percentage of sandy material will call for a lower ratio of solution to ore than a slime pulp from which the granular material has been separated for leaching. A separated slime may need a ratio of 2 or 3 to 1 to give the best extraction,

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[1] Cyaniding Gold and Silver Ores, page 224 (Second Edition).

while a general mill pulp may give as good an extraction with less mechanical troubles if treated in a 1 to 1 pulp.

The usual practice where Pachuca tanks are in use is to make a dilution for the agitation treatment that does not require any subsequent thickening prior to the process in use for displacing the value-bearing solution, and such dilution is usually placed between 1 and 1.2 to 1.

Time of Agitation.—This may vary from a few hours in the case of free milling gold ores to several days for silver sulphide ores, and has to be determined by experiment. As already stated the time necessary may often be shortened by increasing the strength of the solution in cyanide.

Increasing the Dissolving Effect by a Succession of Washes.—It seems to be a very widely observed fact that an increase in extraction is often obtained by settling and decanting several times during treatment and substituting fresh barren solution each time as against conducting the whole period of agitation in the same solution. This is a point the writer has never been able to verify, personally. Even in some plants where the phenomenon has been reported as proved by experiment, he has been able to obtain the same extraction without changing the solution during agitation. In the majority of instances where this fact is observed the writer is inclined to believe that if an increased cyanide strength were used for the agitation a result could be obtained similar to that shown by a succession of washes. Even if this be so, it may happen in any particular case that a succession of weak washes may turn out to be more economical than a smaller amount of stronger solution, and this is a point to be decided by the metallurgist in charge.

Continuous Treatment.—The system of continuous treatment consists in allowing the stream of pulp to flow continuously through a connecting series of agitator tanks, as distinguished from the intermittent system where a charge is placed in a tank and agitated therein until the dissolution of the precious metals is complete. It may be applied to almost any form of agitator, whether the old stirring-geared kind, the Pachuca, or the Dorr.

In the case of all these the usual method is to introduce the stream to a tank through a downcast launder or pipe to a point near the bottom and allow it to make its exit on a level with the surface of the pulp.

For Pachucas an alternative system has been devised by Huntington Adams[1] at the Natividad Mine in Oaxaca. In each case the pulp stream enters at the surface but the outflow is obtained by cutting a section out of the stream issuing from the top of the air lift by means of a receiving box fitted with an adjustable sliding lid. By opening or closing this aperture the level of the pulp in each tank is regulated. Arrangements are also made for bypassing the pulp around any individual tank to allow of emptying for cleaning or repairs. The advantage of this system over the usual one seems to lie in ensuring a complete mixture of the inflowing untreated ore with that already in the tank and preventing the possibility of more than the normal proportion of ore passing into and out of any given tank untreated. At one mill known to the writer, however, where the system was tried the proper regulation of the stream cut from each tank was found so difficult that it had to be abandoned.

The advantages claimed for the continuous system are (1) an increase in the time of agitation for any given unit of capacity; (2) less labor and attention. These apply to all methods of agitation, but there is an additional advantage in the case of Pachuca tanks in that it obviates the loss of head due to their excessive height because the pulp makes its exit from the series only a few feet lower than the point at which it enters.

The only drawback to the continuous system seems to be the inevitable short circuiting of a portion of the pulp. The amount thus passing untreated through any given tank in the series may be considerable and will be impossible to estimate when the system of passes does not ensure a complete mixture of the incoming stream with the pulp contents of the tank before it can reach the exit.

Even when such perfect mixture takes place there will still be

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[1] *Engineering and Mining Journal*, Oct. 7th, 1911.

a definite proportion going through untreated. Assuming a series of 100-ton tanks and a flow of 10 tons per hour, then approximately 1 ton out of the 10 tons flowing into the first tank will pass out immediately untreated into the second tank of the series: one-tenth of a ton per hour of this original pulp will pass directly from the second to the third tank and so on in diminishing fractions to the end of the series.

As the action is continuous, however, the above does not represent the whole of the results of short circuiting. In order to demonstrate the modus operandi of this principle L. P. Hills in the Mining and Scientific Press of Feb. 8, 1913 makes the following calculations:

> "Assume a series of 100-ton tanks with a flow of 10 tons per hour. Consider the efflux from the first tank for any given hour. That 10-ton portion is composed of approximately:
>
> 0.909 tons of the influx of the given hour
> 0.825 tons of the first preceding hour
> 0.751 tons of the second preceding hour
> 0.683 tons of the third preceding hour
> 0.621 tons of the fourth preceding hour
> 0.565 tons of the fifth preceding hour
> 0.513 tons of the sixth preceding hour
> 0.466 tons of the seventh preceding hour
> 0.424 tons of the eighth preceding hour
> 0.386 tons of the ninth preceding hour
> 0.351 tons of the tenth preceding hour
>
> The remaining 3½ tons in diminished portions back to the initial inflowing 10 tons. The great bulk of the pulp receives exceedingly long treatment, the time of treatment of different portions varying between wide limits."

Another drawback to the continuous method exists in the case of Pachuca tanks. These tanks when kept full of pulp rapidly build up a deposit of hard slime around the sides thus seriously reducing the capacity available for pulp in motion, and proportionately reducing the time of treatment and increasing the fraction of the flow which has to pass untreated to the next tank in the series. It is therefore necessary to cut out each tank every two or three weeks for cleaning, and even so the available space for agitation is never at any time up to the maximum cubic capacity. Besides this permanent reduction in space there is the time lost in emptying and cleaning.

Loss in Extraction Due to Continuous System.—At one important mill in Mexico it was shown that the continuous system with Pachuca tanks as at that time being operated was responsible for a loss in extraction of 10 to 15 grams of silver per metric ton and the tails were reduced by that amount when the charge system was substituted for the continuous. The probability is, however, that the greatest contributary cause of this difference was due to the system of passes between tanks which allowed a straight flow from one tank to the next only a little below the pulp level. This while not attended with serious results when the tonnage and rate of flow were small yet was the cause of considerable short circuiting as the mill capacity, and therefore the volume of pulp, was increased, the velocity of flow carrying some of the pulp right across the tank to the exit before it had time to sink and become mixed with the rest of the charge. When instead of running the whole series of tanks in tandem they were connected into three or four separate series and the stream of pulp split up into an equal number of parallel streams the bad effect was reduced to a minimum, and in a case of this kind assuming that each tank is cleaned out frequently the added convenience of the continuous system would probably outweigh the remaining drawbacks.

Slime Treatment by Leaching in Filter Presses.—In New Zealand in the early days of slime treatment one of the first methods to be applied commercially was the use of filter presses and at first the dissolution of the gold was effected in the press as well as the subsequent washing out of the metal-bearing solution. This worked well with surface ores on which the action of the cyanide was rapid, but later it was found necessary to conduct the treatment by agitation, reserving the use of the filter for dewatering and washing only.

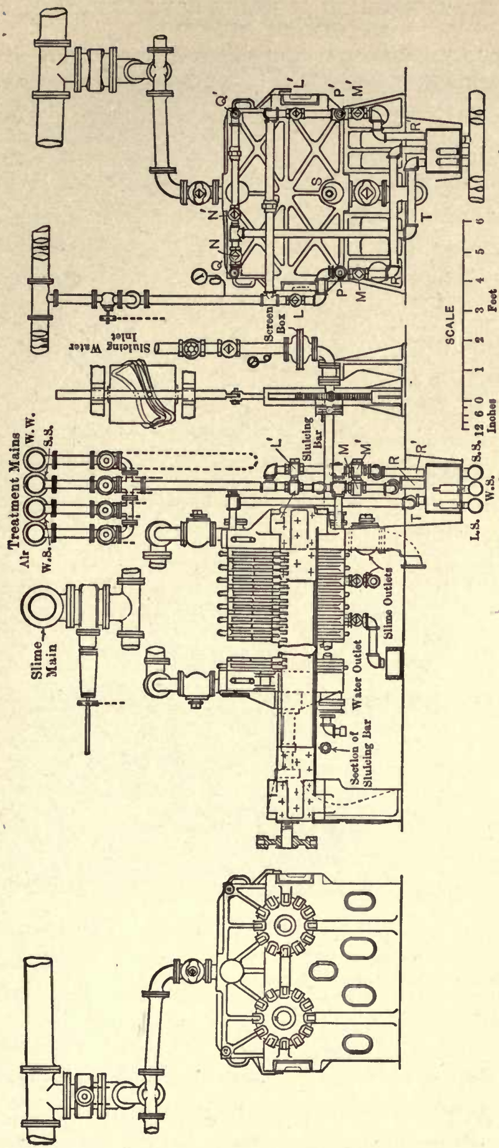

FIG. 18.—Merrill Slime Press as used at the Homestake Mill.

THE MERRILL AUTOMATIC SLUICING PRESSURE SLIME FILTER
METHOD OF SLUICING

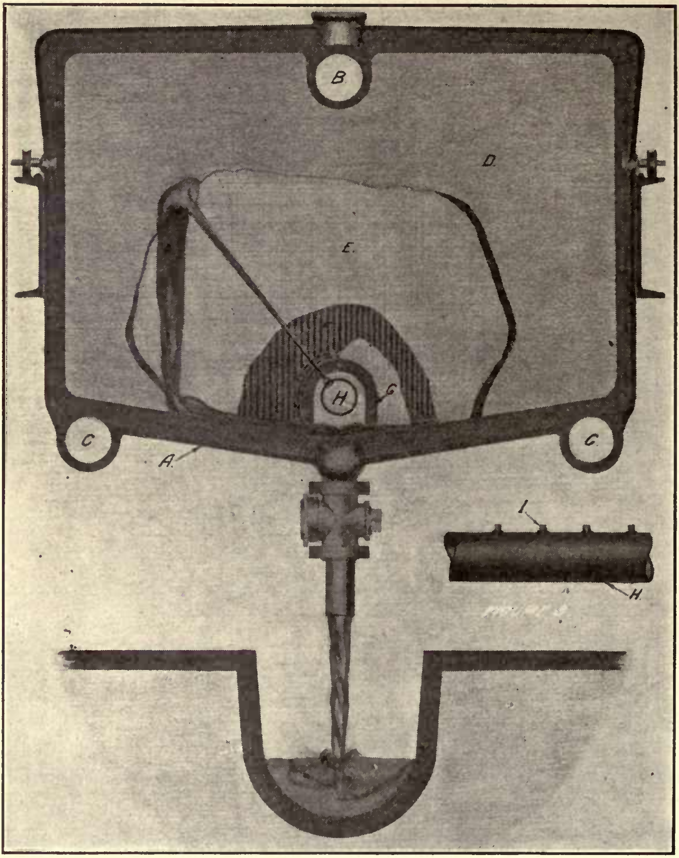

Fig. 19.—Merrill Slime Press. Method of Sluicing.

A. Standard container or frame.
B. Feed channel through which the slime pulp enters each frame.
C. Channels from which water or solution is drawn off during process of filtration.
D. Partially sluiced slime cake.
E. Filter cloth with portion removed showing corrugations of filter plate.
F. Filter plate.
G. Horseshoe clamp for holding filter cloth against filter plate.
H. Sluicing pipe, containing water under 60 to 90 lbs. pressure admitted at either or both ends.
I. Sluicing nozzles located at center of each container or frame.

Note.—The sluicing pipe is automatically revolved forward and back through an arc of about 180 degrees, by means of the mechanism shown at the head of the press. As the cake is washed away by the action of the jet of water playing upon it, the mixture of slime-residue and water flows into the channel underneath the sluicing pipe and through the outlet cocks to the waste conduit below.

At the present day probably the only mill of importance where the entire treatment of the slime is done in presses is at the Homestake.

> "The Merrill press is essentially of the ordinary rectangular flush-plate and distance frame pattern with internal channels, but equipped with the automatic discharging device which is the distinguishing feature of the press. A standard press, 28 of which are now in service, containing 92 4-inch frames and 91 plates, is 46 ft. long and holds somewhat over 600 cu. ft. or 25 tons of slime; under the usual treatment schedule and with average charges, its capacity is about 70 tons per day."[1]

The automatic discharging device referred to consists of a nozzle in the interior of each frame capable of being rotated vertically through an arc of 200 degrees. When it is desired to discharge the residue cake water is supplied to these nozzles at a pressure of 60 lb. per square inch and simultaneously the nozzles are rotated backward and forward breaking down the cake and sluicing it into the discharge channel in the lower part of the press through which it finally finds its way to the dump.

Homestake Slime Process.—The thickened slime in which the proper amount of lime has already been added is supplied to the press by gravity under 30 lb. pressure, the surplus water flowing off to waste until the frames are filled with firm slime cake, which takes about 70 minutes. Air is then passed through the cake in a horizontal direction, that is, entering from behind the cloth on one side of a cake and passing out through the cloth on the other side of the same cake. This aeration lasts one hour and is followed by a wash of weak precipitated solution for about 30 minutes. The cake is again aerated for an hour and then receives a wash of strong solution (0.1% KCN) for an hour, followed by weak solution and water wash

Clark and Sharwood[2] state that

> "The treatment time is varied according to the leaching rate of the charge and is calculated to pass approximately equal amounts of solution

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[1] Clark and Sharwood, *Proceedings* I. M. M., London, Nov., 1912.

[2] *Proceedings* I. M. M, London, November, 1912.

through the slime cakes, irrespective of the leaching rate. The leaching rate, as described here and referred to later on, is an empirical standard, being taken as the number of seconds required to discharge 5 U. S. gallons from the press under the standard wash water pressure. A charge is treated according to the leaching rate shown by the previous charge, a reasonable system in that the leaching rate is affected chiefly by the accumulation of carbonate of lime on the cloths, there being few sudden variations in the character of the slime. The working schedule now in use is given."

Working Schedule Homestake Filter Treatment.—The "Class of press" refers to the rate of leaching of the various presses in the system.

TABLE 4

Class of press	*A*		*B*		*C*	
Approximate per cent. of presses in this class	50		40		10	
Leaching rate	7 or faster		8 and 9		10 or slower	
	Hr.	Min.	Hr.	Min.	Hr.	Min.
Air	1	0	1	20	1	30
Weak solution	0	25	0	30	0	35
Air	1	0	1	0	1	20
Strong solution	1	0	1	20	1	30
Weak solution	0	50	1	30	1	40
Water	12 or 13 tons in all cases according to time available.					

Washing out of the Metal-bearing Solution after the Dissolving Treatment.—This operation constituted the chief problem in the earliest attempts to work out a slime treatment method. As early as 1893 James Park[1] patented a process whereby the slime was agitated in a filter bottomed tank, agitation and filtration proceeding simultaneously, the paddles being arranged in such a way that no solid cake was built up on the filter medium but a

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[1] The Cyanide Process, page 140 (Fifth Edition).

gradual and homogeneous thickening of the whole mass took place. When this thickening had proceeded to a suitable point the mass was diluted with more solution and re-thickened, the operation being repeated as often as necessary.

In South Africa it was very soon decided that a series of settlements, decantations, and re-dilutions was the only method devised up to date that would even approximately meet the local conditions, and this system has been the standard and universal practice there ever since, though recently vacuum filtration has been introduced in some mills.

While the decantation process was being worked out on the Rand the problem was being solved in Australia by the use of filter presses, the principal reason for the difference probably being the scarcity of water which would have rendered the decantation process almost prohibitive in some localities. The frame and plate type of filter press is so well known that it is unnecessary to give a detailed description of it here. The form most generally used in Australian practice is the Dehne. The advantages of the system are that the metal-bearing solution may be almost completely displaced from association with the solids in a short time and with a much smaller volume of wash liquor than is possible by decantation, and also with a minimum loss of moisture in the residue. The drawbacks consist in the comparatively high cost for labor and repairs, making it unsuitable for large tonnages of low-grade material.

To obviate the expense incidental to the use of the standard type of pressure filter, developments have taken place in three directions, the first being represented by the Merrill self-dumping frame-and-plate press already described, the second by various forms of vacuum filter whereby the pressure of the atmosphere is substituted for that of pumps or hydrostatic head, and third a combination of ideas gathered from both types whereby free filtering leaves or units are suspended in a closed receiver filled with slime pulp and mechanical pressure applied to the interior as a whole.

Group No. 2 consists of the Butters, the Moore, the rotary drum filters such as the Oliver and Portland, and the disc filter or "American Continuous Suction Filter."

The Butters filter consists of a number of filtering units or leaves suspended vertically in a containing box with hopper bottom. The leaves are usually constructed of a frame-work of pipe suspended from a 3″ × 9″ wooden support. The filtering medium consists of a cocoa matting foundation enclosed between sheets of canvas, the edges of which are so adjusted that only solution free from solid matter can enter into the interior of the unit. The pipe composing the lower part of the frame is perforated, so that when the extension of this pipe projecting above the level of the pulp is connected with a vacuum drum, the solution from the external pulp enters the unit through the canvas and passes away through the pipe system, leaving the solid matter on the canvas surface in the form of a cake. The containing box is first filled with pulp by means of a centrifugal pump and the vacuum applied to the leaves. When a cake of sufficient thickness has been built up on the canvas surface, the vacuum is lowered to a point just sufficient to keep the cakes adherent to their supports, the surplus pulp in the containing box is pumped out and replaced with barren wash solution or water, and the vacuum again raised to its maximum. The lowering of the vacuum during the time the cake is exposed to the air is very important and its purpose is to prevent excessive drying and shrinkage of the cake with formation of cracks which reduce the displacing efficiency of the subsequent wash and also tend to cause a premature dropping of the cake. When sufficient wash liquor has been forced through the cakes to displace the pregnant solution originally contained therein, the surplus remaining in the box is pumped back whence it came and the vacuum cut off the leaves. The cake is then dropped to the bottom of the box and discharged to the dump through doors at the apex of the hoppers. The dropping of the cake may be arranged to take place automatically by gravity, or pressure (either air or water) may be applied to the interior of the leaf as may be more convenient. Where possible it is usually better to drop the cake while the

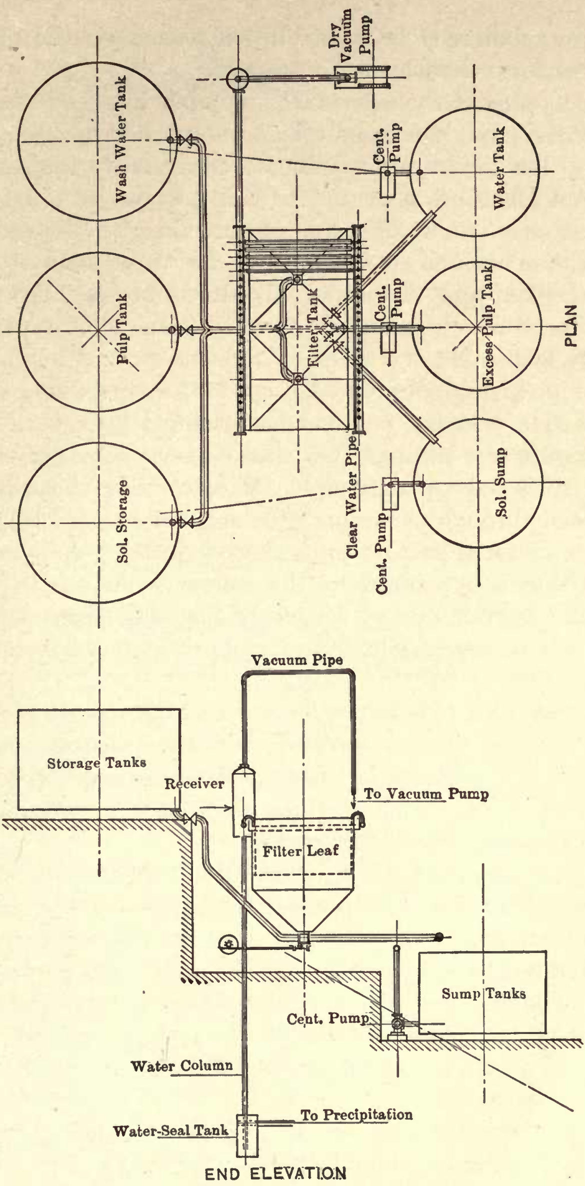


Fig. 20.—Butters Filter. Complete Gravity System.

leaves are submerged in water, but if necessary, the operation may be performed with the box empty.

This dropping of the cakes into an empty box, however, is not so satisfactory but can usually be accomplished by using a large volume of low pressure air suddenly admitted to the interior of the leaves. The use of water for this purpose does not answer since it at once throws off a line of cake along the bottom of the leaf and thus obtains an exit so that the water level never rises within the leaf and the upper two-thirds of the cake remains undetached. The best way to effect a discharge of the cake into an empty box is to use a specially constructed leaf having a perforated pipe along its upper edge inside the canvas and attached to the wooden header, one end being closed by a plug and the other passing out through the header at the opposite end from that of the usual vacuum pipe. For dropping the cake water is admitted through the upper pipe and compressed air through the lower or usual suction pipe, the result being that the water flowing from above moistens the canvas from top to bottom, acting as a lubricator and rendering the cake easily detachable by the compressed air acting uniformly over the whole interior of the leaf.

If the leaves are kept in good repair and thoroughly pervious to the solution, this filter gives good service not merely as a dewaterer but also as a means of removing the rich pregnant solution from the residue, so that with medium grade ores no washing by decantation is needed before filtration.

The principal drawback lies in the time and expense consumed in repeated transfers of pulp, solution, and water, into and out of the container. There is also a tendency to salt the barren-wash and water-wash storage tanks making such washes less effective for their purpose, but the latter drawback can be largely overcome by using what is known as the complete gravity system and having a separate pump and pipe line each for pulp, solution, and water wash.

At the mill of the Butters Divisadero Company owing to the excessive deposition of lime in the filter canvas and the high cost

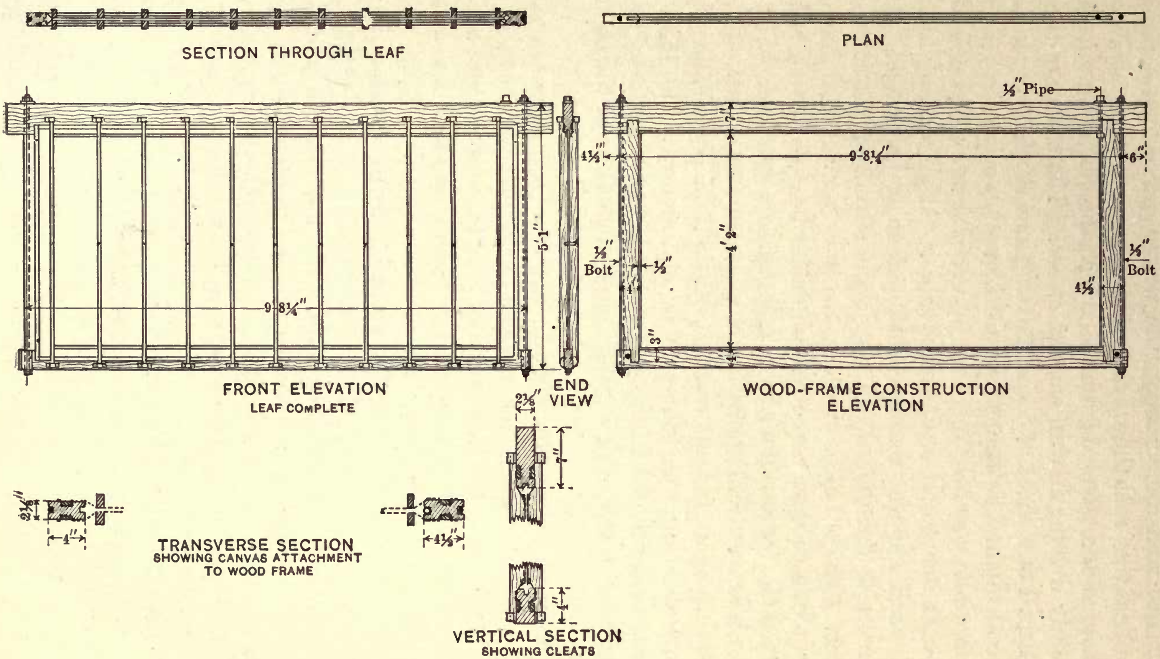


Fig. 21.—Bullock-Forth-Hamilton Leaf for Butters Filter.

of muriatic acid for cleaning, the writer introduced a different form of leaf that met the special circumstances very satisfactorily. It consisted of a wooden frame on which the cloths were fastened by merely driving in thin wooden cleats. The centre of the leaf was formed of a sheet of cocoa matting and the filtering medium was good quality twill cotton sheeting. No stitching was used but the whole was held together by vertical strips of wood about ten inches apart and easily removable. After running about a month each leaf was removed and instead of being acid treated was entirely dismantled, an operation requiring about twenty minutes. The old filter medium was thrown away and the cocoa mat put out to dry, to be subsequently beaten to free it from deposited matter and later to be inserted in a new leaf. The leaf in hand was then refitted with new sheeting and a clean mat and clamped up again for replacement in the filter box. The cost of the new sheeting was found to be considerably less than the former cost of acid for treating the canvas and in addition to this the leaf after renovation was far more efficient than the old acid treated leaf.

There seems to be an opening for the use of such a leaf in localities where muriatic acid is expensive or difficult to transport and also in cases where the deposition of lime is excessive and accompanied, as it was at Divisadero, by substances much less soluble than lime in muriatic acid.

The Moore filter is similar in principle to the Butters but in this case the leaves are made removable and are transferred successively from the pulp box to the wash solution and from there to the water wash and finally to the dumping hopper. This is done by aggregating a number of filter frames into a unit or basket and raising, transferring, and lowering by means of an electric overhead crane.

The Oliver Filter.—This consists essentially of a slowly revolving drum, the lower half of whose circumference is submerged in the pulp to be filtered. The filtering medium is applied all around the periphery, which is divided up into a number of separate segments, more or less isolated from one another.

The vacuum is applied through the axle bearing by means of a header valve which is so arranged that at a certain point in the revolution of the drum the vacuum connection is severed and

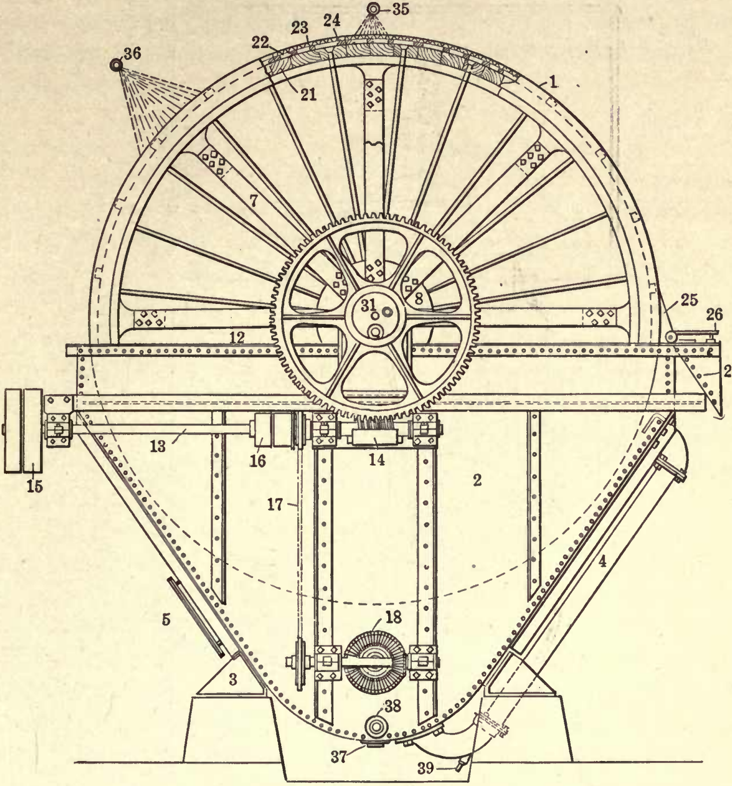


Fig. 22.—Oliver Filter.

compressed air admitted under the filter medium which serves to detach the cake and allow it to drop onto a metal deflector or scraper, whence it slides down to the residue discharge launder. As the newly formed and adherent cake rises from the bath of slime pulp it comes under the action of a number of atomizing spray nozzles by means of which the wash liquor is distributed

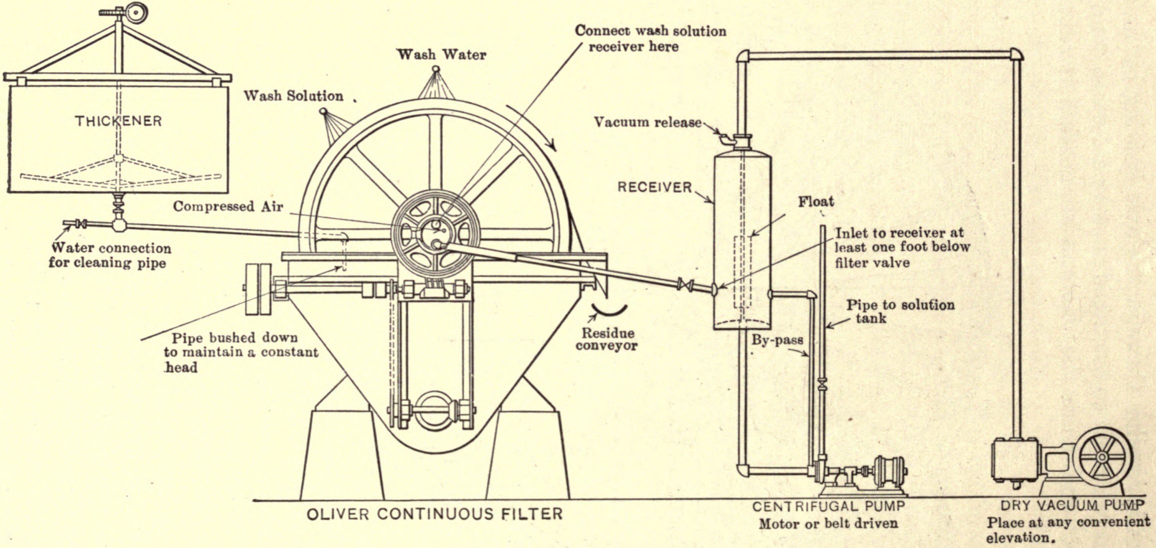


FIG. 23.—Oliver Filter. Showing General Arrangement.

over the surface and is at once absorbed, tending to act as a wash for displacing the value-bearing solution.

This filter is moderate in price, its running expenses are low, and it is continuous in operation, and while it cannot yet be considered a perfect solution of the filtration problem it seems to have the elements of more permanent usefulness than any other device at present in vogue.

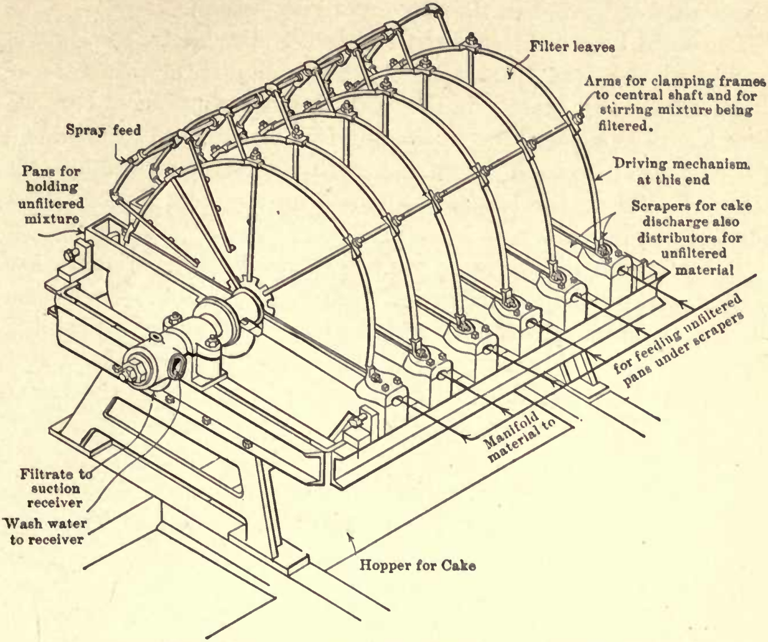


FIG. 24.—Disc Filter. ("American Continuous Suction Filter.")

Its principal defects are its small washing capacity, necessitating one or more decantation washes before filtration, and the mechanical difficulties involved in keeping the pool of pulp properly mixed and agitated in cases where there is a larger proportion than usual of granular or sandy material present. It shows itself at its best when employed as a dewaterer after several decantations have been given to remove pregnant solution, and when the material fed to it approaches most nearly to the definition of a true slime.

With this, as with all other vacuum filters, a thick pulp is essential to good work, both to ensure the homogeneity of the pulp in the container and also to obtain a high capacity per unit.

The Disc Filter is comparatively new, but it appears to be doing satisfactory work where it is in use. The principle of it is sufficiently indicated in the accompanying sketch.

The third type of filter is represented by the Kelly, the Sweetland, and the first Burt filter. The fundamental principle of these, as already stated, is the enclosing of a number of vertical filter leaves in a closed receptacle and forcing the pulp by mechanical pressure into such receptacle, with the result of building up cakes of solid on the leaves and expelling the clear liquor into a header pipe outside.

The methods of discharge vary. In the Kelly the leaves are withdrawn on a roller carriage before dumping. In the Sweetland the leaves remain stationary and the lower half of the containing shell or receptacle is removed, while in the Burt the cake is first blown off and then the end of the shell is removed: the latter being erected at a steep angle, the residue cake slides out as soon as the door is removed.

The later Burt filter is rather different from any of the foregoing. It consists of a revolving horizontal tube somewhat like a tube mill, and has a complete lining of filter material. The pulp is fed to it under mechanical pressure and a cake is built up in the form of a uniform lining inside the tube from 1 to 4 or 5 inches in thickness. When the supply of pulp is cut off, the pressure is maintained with compressed air so that filtration can be carried to a finish and no removal of excess pulp is necessary. For dumping, one end of the cylinder is removed and the revolving motion being restarted, the cake automatically falls off and discharges itself into the tailing launder.

This filter is said to be especially applicable in the case of exceptionally granular and porous pulps, and is in successful use at one or more mills in the El Oro district of Mexico, though it has not obtained much vogue elsewhere. The idea, however, seems capable of wider application.

The Dorr Continuous Countercurrent Decantation.—This is an interesting development of the old decantation process, devised by John V. N. Dorr, and while it contains the same principle of displacement by successive dilutions, the action is much more rapid, the same result is attained by the use of considerably less barren solution and it is continuous in action needing little power and practically no attention.

For its application a succession of Dorr thickeners is used, the series usually consisting of four. Each succeeding thickener is placed at a slightly higher elevation than the preceding one so that the clear overflow may be able to flow back by gravity. There are thus two separate currents of fluid running in opposite directions, the pulp after the dissolving treatment flows toward the residue discharge end of the series while the clear solution overflow runs backward from the discharge toward the treatment and milling end of the plant.

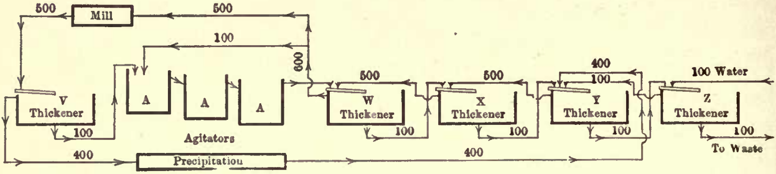
The system is usually, though not necessarily, supplementary to some system of continuous agitation dissolving treatment. The slime pulp from the milling department goes first to a thickener, the clear overflow from which being the highest in value of any in the plant forms the pregnant solution for precipitation. The thick underflow is raised by a diaphragm pump to the series of continuous agitators and, as it enters, is diluted to the desired degree with solution overflowing from No. 1 washing thickener. The pulp, when the dissolving treatment is finished, passes continuously into No. 1 washing thickener, being diluted on its way by the overflow solution from No. 2 washing thickener. The underflow from No. 1 washing thickener is raised by a diaphragm pump to the level of No. 2 where it meets the stream of solution overflowing from No. 3. The thick underflow from No. 2 is raised to the level of No. 3 where it is diluted with barren solution and also with the overflow solution from No. 4. The underflow from No. 3 is raised to No. 4 and diluted with water. The underflow from No. 4 goes to the dump but if necessary may be first passed through a dewatering filter.

The following typical flow sheet and calculations for dissolved value loss are taken from a pamphlet issued by the Dorr Co.

Regarding the assumption in these calculations that 75% of the whole metal dissolved goes into solution during milling, this is probably about correct for a gold ore, but for a silver ore it would in the writer's experience be nearer the facts to assume a dissolution during milling of only 15% to 25% of the total silver dissolved.

EXAMPLE NO. 1

Flow Sheet—Type "B"



All figures refer to solution tonnages.

Calculations for Dissolved Value Loss

Conditions Assumed:

(*a*) 100 tons of ore per day crushed in cyanide solution.
(*b*) Discharge from all Thickeners with 50% moisture.
(*c*) $10.00 value dissolved per ton of ore.
(*d*) 75% in mill and 25% in Agitators.
(*e*) 400 tons of solution from Thickener V precipitated to $0.02.
(*f*) Agitation with a dilution of 2 of solution to 1 of solids.
(*g*) Let V, W, X, Y and Z represent value in dollars per ton of solution discharged from the respective Thickeners.

Equating out of and into each Thickener:

(1) $100V + 400V = 500W + (0.75 \times \$10.00 \times 100)$.
(2) $100W + 600W = 500X + 100W + (0.25 \times \$10.00 \times 100) + 100V$.
(3) $100X + 500X = 100W + 500Y$.
(4) $100Y + 500Y = 100Z + 100X + (400 \times 0.02)$.
(5) $100Z + 100Z = 100Y$ + 100 tons of water value $0.00.

Simplifying:

(1) $V = W + 1.50$

(2) $W = X + 0.80$

(3) $X = Y + 0.16$

(4) $Y = 0.2Z + 0.048$

(5) $2Z = Y$

Solving:

$V = \$2.51332$

$W = 1.01332$

$X = 0.21332$

$Y = 0.05332$

$Z = 0.02666$

To check these figures:

The amount precipitated from 400 tons @ (\$2.51332 − 0.02)	= \$997.328
The amount lost in tailings, 100 tons @ \$0.02666	= 2.666
	\$999.994
The amount due to neglected decimals	= 0.006
	\$1,000.000

The amount dissolved 100 tons @ \$10.00 = \$1,000.00.

From the foregoing the following results are deduced:

Assay value of the pregnant solution, *i.e.*, value of V	= \$2.51332
Assay value of the discharged solution, *i.e.*, value of Z	= 0.02666
Loss of dissolved value per ton of ore,	= 0.02666

Dissolved value saved 99.7%.

Calculation for Mechanical Loss of Cyanide

Conditions Assumed:

(*a*) Neglect the Cyanide consumption throughout the system.

(*b*) Strength of Cyanide per ton of solution 1.0 lb.

(*c*) Let V, W, X, Y and Z represent the strength in pounds of Cyanide in Solution discharged from the respective Thickeners.

Equating out of and into each Thickener:

(1) $V = 1.0$.

(2) $100W + 600W = 100W + 100V + 500X$.

(3) $100X + 500X = 100W + 500Y$.

(4) $100Y + 500Y = 100Z + 400V + 100X$.

(5) $100Z + 100Z = 100Y$ + 100 tons of water.

Simplifying:

(1) $V = 1.0$

(2) $6W = 5X + 1$

(3) $6X = W + 5Y$

(4) $6Y = Z + X + 4$

(5) $2Z = Y$

Solving:

$V = 1.0$

$W = 0.9109$

$X = 0.8932$

$Y = 0.8898$

$Z = 0.4449$

$\frac{100}{100} Z = 0.4449$ lb. = Mechanical loss of Cyanide per ton of ore.

TABLE 5.—OTHER ASSUMED CONDITIONS AND CALCULATED RESULTS. TYPE "B" FLOW SHEET

Example No.	1	2	3	4	5	6	7	8	9	10	
Value dissolved	$6	6	6	10	10	14	14	18	18	10	ASSUMED
Per cent. dissolved in grinding department	75	75	75	75	75	75	75	75	75	50	
Per cent. moisture in Thickener discharge	50	50	37.5	50	37.5	50	37.5	50	37.5	50	
Ratio tons of solution precipitated to tons of ore	3 : 1	2 : 1	2 : 1	4 : 1	2 : 1	4 : 1	3 : 1	5 : 1	3 : 1	4 : 1	
Assay value of discharged solution	$0.031	0.074	0.036	0.026	0.056	0.033	0.030	0.026	0.035	0.035	CALCULATED
Loss of dissolved value per ton of ore	$0.031	0.074	0.022	0.026	0.034	0.033	0.018	0.026	0.021	0.035	
Per cent. of dissolved value saved	99.4	98.8	99.6	99.7	99.6	99.7	99.9	99.8	99.9	99.6	
Pounds of KCN per ton of solution	1.0	1.0	1.0	1.5	1.5	1.0	1.0	1.5	1.5	1.0	
Mechanical loss of KCN per ton of ore in lb.	0.43	0.41	0.26	0.67	0.39	0.44	0.27	0.69	0.41	0.44	

The two requisites for efficient working of the system are a sufficient volume of precipitated solution and a perfect precipitation yielding a tail solution that is for practical purposes barren. It is possible under these conditions to get a very satisfactory elimination of the dissolved precious metals from the residue.

Regarding elimination of the cyanide it is not quite so satisfactory because, as in all other washing processes, the quantity of final water wash available is limited by the daily tonnage of moisture eliminated from the plant, as otherwise the stock solutions would soon increase to an unmanageable bulk. In the case of gold ores where a weak cyanide solution can be used with effect the loss of cyanide in the residue is inconsiderable, but for silver ores where a treatment solution of 0.1% or over is employed it will often be found profitable to use a filter of some kind before discharging the residue from the last thickener to the dump.

It has often been observed, when working this process, that although the dissolving action in the treatment tanks may have apparently ceased before the pulp enters the washing thickeners, additional small amounts of precious metal seem to pass into solution during the washing out process, corroborating the theory already alluded to of the efficacy of successive washes for the dissolving process, a tentative explanation of which has already been given.

Another phenomenon, rather difficult to explain, is also sometimes in evidence in the use of this process. Even in cases where little or no dissolving action is observed in the successive thickeners in which cyanide solution is used as the diluent yet when the pulp reaches the last thickener and is diluted with water marked dissolution of the metals seems to take place. This may perhaps be due to the increased activity of the ions in a less viscous medium than is afforded by the stock solution. On the other hand it seems possible that the dissolving effect may be only apparent and that in reality the result may be due to a liberation of pregnant solution previously adsorbed by the colloidal constituents of the pulp. Such a liberation might conceivably be brought about by the changes in the properties of the electrolyte due to excessive dilution with water. One such change might be traced to a decrease of lime in solution sufficient to cause the curds or aggregations of clayey matter to disintegrate into their original units liberating soluble matter which had been entangled with them. If it be asked why a similar action is not displayed when making washed assay samples for the determination of these various points, the answer might lie in the fact that such washed assay samples are now almost universally prepared on laboratory vacuum filters, so that they are not subjected to the same conditions as obtain in the working of the plant.

CHAPTER V

MILLING IN CYANIDE SOLUTION

Milling in Cyanide Solution.—This was first tried on the Rand in 1892 but was soon abandoned, probably on account of difficulties arising out of the fact that there was at that time no method of treating the slime. It was introduced as a practical system in New Zealand in 1897 and in South Dakota in 1899. Since then the practice has grown until at the present time it may be considered standard. Almost the only exception to its use is in the case of mills where the ore is subjected to amalgamation before the cyanide treatment.

The reasons for this exception are two: first that the presence of lime hardens the amalgam. This is not considered an objection on the Rand, where amalgamation is usually carried out in a lime water pulp. Second, the cyanide solution is said to attack the copper plates through the coating of amalgam and gradually to destroy them. Clennell,[1] however, states that with solutions of .03 to .07 per cent. KCN the action on the plates is not serious, and E. S. Leaver (Mining & Scientific Press of Aug. 22, 1908) says that at the Nevada Goldfield Reduction works plate amalgamation had been in use one year while milling with 0.1% cyanide solution without any particular effects on the plates having been noted. E. H. Nutter (M. & S. Press, Dec. 14, 1907) says that he found the method satisfactory at Bodie and at the Liberty Bell though it was necessary to renew the plates twice a year. A third objection has been raised, especially on the Rand, to the effect that crushing in solution renders impossible a correct sampling of the ore head for assay. The statement is made that the only way to obtain a representative assay of the ore entering the mill is to sample the pulp after it leaves the mortar boxes, and this is of course rendered difficult if not impossible if cyanide

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[1] The Cyanide Handbook, page 342 (Second Edition).

solution is used in place of water. It would seem, however, that if the system has advantages as a money saver (and this class of objector if often willing to admit the fact), it would be more profitable to use it and put up with a degree of uncertainty about the head assay, if it is really impossible to get a correct sample of the ore except at the lip of the mortar boxes.

The advantages of milling in cyanide solution are mainly two: The first is that the ore is brought into contact with the solvent at the earliest possible moment, thus reducing the time needed for the dissolving treatment proper and decreasing the size of plant necessary. It is no uncommon thing to find from 70% to 80% of the gold in an ore dissolved during milling and before the pulp reaches the treatment tanks, though silver does not respond to the same extent, the amount dissolved from a silver ore during milling being usually from 5% to 25%.

The second advantage is that the introduction of large quantities of water into the solution stock is avoided. It is obvious that, considering any cyanide plant as a whole, no more water can be taken into the system daily than is withdrawn from that system in the same period; consequently, if, with every ton of slime, a ton of water be taken into the plant, then for every ton of slime residue discharged a ton of solution must be got rid of in some way or other. Some do this in the form of moisture in the slime residue; and others, who prefer to displace part of this solution moisture by water, run to waste an equivalent quantity of solution from the tailing of their precipitation-boxes, so as to preserve the balance in the system. This difficulty is much accentuated where the slime residue is treated in filter-presses or vacuum-filters; it is to some extent mitigated where double filtering is practised, that is, where the excess water is extracted from the slime before cyanide solution is added, though even here, when the cake contains 25% moisture, for every three tons of slime a ton of water is introduced into the system and a corresponding ton of solution has to be eliminated at some point, rendering it difficult or impossible to give a displacing water-wash to the residue cake without running to waste weak solution. It is bad enough to have to throw away solution carrying cyanide even when its gold content is nil or a "trace," but in many instances that so-called "waste" solution may be the vehicle of serious gold-losses in addition to the loss of the contained cyanide.

The only correct method of adding water for maintaining the bulk of stock solution is to use it as a final wash for the tailing before discharging, and when milling in cyanide the only water of necessity introduced into the plant otherwise than this is that in the form of the natural moisture in the ore, consequently it is possible to apply practically the whole of the daily quota of water as a means of removing the bulk of the cyanide and the last traces of dissolved gold and silver from the tailing.

CHAPTER VI

DISCREPANCIES BETWEEN ACTUAL RECOVERY AND THEORETICAL EXTRACTION

SECTION I

Everyone connected with cyanide plants has at some time or another been confronted with a real or apparent shortage of bullion in the clean-up, that is, the bullion recovered has been less than the amount expected when calculated from the assays of head and tail and the tonnage treated.

Such a shortage will often occur at the end of the first month's run of a mill and is easily accounted for. First, there may be an absorption of metal-bearing solution by the new woodwork and filtering medium; second, there may be mechanical losses by leakage, due to joints in wooden containers not yet having closed under the influence of the moisture, and also to run-overs which may easily happen during the process of getting a new plant and its shiftmen settled down to their working routine. If precipitation is done by zinc shavings, a large quantity of the precious metals first dissolved will be needed to coat the surface of the zinc in the boxes before a nonadherent deposit is formed which can be cleaned-up and melted.

Again, the relation between the amount of bullion found and the amount calculated as due for the first month is complicated by the difficulty of assigning to a given month its exact quota of ore and pregnant solution. For instance, take the case of a sand leaching plant where the treatment of a charge extends over 10 days; it will be obvious that if the collecting of a sand charge begins on the first of the month and one charge is collected every 24 hours, then by the end of the month, 31 charges of, say, 100 tons each, will have been collected, but only 21 of them will have been completely treated and discharged from the plant; the other 10 will be in various stages of advancement. No. 22 will perhaps be just finished and ready to go to the dump, 23 will be in the water wash stage, 24 in the weak solution wash stage, until when we reach No. 31 we find it has had no treatment at all and has as yet yielded up nothing of its valuable contents. It is thus evident that if the calculation for recovery is made on the tonnage of ore milled and collected during the first month, the bullion cleaned up will fall very far short of the amount called for.

Now suppose that to obviate this difficulty it is decided to calculate the amount of bullion due by taking account only of such charges as have already finished their treatment and been discharged from the plant. In this case the bullion cleaned up should be in excess of that calculated because the charge collected on the last of the month will not have finished its treatment till the 10th of the following month, consequently the first month's clean-up will contain not only all the precious metal contents of its own thirty-one days, but all of that belonging to the charge of the first day of the following month, practically all of that of the second, most of that of the third and so on in diminishing amounts to the 10th day when the clean-up has taken place. Thus whatever method of computation be adopted it is almost impossible to ascertain the true quantity of bullion that ought to be expected as the result of the first month's work. The same principle may be applied to slime treatment, whether continuous or intermittent.

It may be laid down then as a general principle that no importance whatever need be attached to the relation between calculated and actual recovery during the first month's run of a mill.

When a mill is well under way the high officials of the Company usually expect the bullion returns at the end of each month to check with the calculated amount due, but here again there are legitimate reasons why this may not always happen. One such reason lies in the computation of the tonnage treated. This is most often arrived at by weighing or estimating the amount of ore fed to the mill bins during the month. As there is nearly always a carry-over of ore from one month to another, this method involves an estimation of the amount remaining in the bins on the last of the month, which may or may not be correct. Another method of obtaining the tonnage is to determine it by specific gravity of slime pulp collected or discharged, or in the case of sand, by cubic capacity. Both these methods are open to small errors even when great care is exercised, but they get away from the necessity of guessing the amount of ore held over in the bins.

Again, the difficulty observed during the first month of milling is apparent in a less degree in the case of subsequent individual months, namely that of drawing a clear line between the tonnage, head assay, tail assay, and gold and silver precipitated, belonging to any two adjoining months. With ore of uniform assay value the resulting difference would be small but with ore whose contents are of a variable and patchy nature, it may be considerable.

If zinc shavings be used as the precipitant the quantity of bullion held over in the boxes may be a variable quantity, due to differences in the method and thoroughness of dressing them. If the quantity of zinc is allowed to become low toward the end of a month, there will be a greater recovery of bullion than would be the case if the boxes were well packed with new zinc a few days before the clean-up.

It will thus be seen that it may often legitimately happen that the bullion recovered at the end of a given month may be less than the amount called for by a calculation of tonnage and assays. If, however, there is nothing radically wrong with the methods of computation, such losses on individual months should be balanced by corresponding gains in other months, and at the end of a year or, say, six months, the total actual recovery should tally very closely with the calculated one. Should this not be the case, it will be obvious that all is not right, and that investigations are called for.

When such a discrepancy really exists there are six possible ways of accounting for it: (1) Theft, (2) carelessness, *e.g.*, losses of solution by run-overs and losses of precipitate by careless handling in cleaning up and melting, (3) errors in tonnage computation, (4) errors in sampling of ore heads, (5) errors in sampling of residues, (6) errors in the assaying.

Of these various possibilities (1) is probably comparatively rare; (2) should be inconsiderable if proper supervision is exercised, though there must be always a small unavoidable loss in cleaning up and melting; (3) and (4) are in the writer's experience the most fertile causes of serious discrepancy; (5) should not be a factor if ordinary care and intelligence are exercised; and (6) needs careful watching.

CHAPTER VII

DISCREPANCIES BETWEEN ACTUAL RECOVERY AND THEORETICAL EXTRACTION

SECTION II

TONNAGE COMPUTATION

In some mills every car or the belt conveyor if such be in use, passes over an automatic weighing and recording scale and this method comes nearest of any to giving a correct determination of the tonnage, but even here an error may creep in due to faulty correction for moisture. The percentage of moisture in the current ore is usually taken as a fixed quantity, and the amount is more often guessed at than ascertained. Even when an honest attempt is made to obtain a correct average figure for moisture, the determination is by no means easy. Ore from different parts of the mine is liable to vary over wide limits in its moisture content, and the season of year may make a difference in the ore from any given stope. And if there is an opening for discrepancies in tonnage under the weighing system, what is to be said of the more common practice of taking the buckets or cars at a fixed and arbitrary weight and multiplying by the number passed into the mill during each day?

Probably the best way to estimate average moisture in the ore fed to the mill is to weigh a whole bucket load at intervals and at once dry it on a flat-sheet supported on bricks, over a slow fire, weighing it again when dry. If two or three buckets are treated in this way every second day for a week or ten days a fairly good idea may be obtained of what correction ought to be made, and an average thus arrived at. Similar determinations should be made both in the dry and the wet seasons. If ore in different parts of the mine varies greatly in its water content, separate estimations of each kind of ore should be made and the number of cars of such ore during the day or month noted. If a sample of this dried ore be taken and tested for moisture after being pulverized in the laboratory, a further reduction in weight may often be recorded amounting to 1% or over.

Another method commonly made use of for arriving at the monthly tonnage is to estimate it from the ore pulp in the cyanide plant, the sand (if leaching is in use) being calculated from the cubic contents of each charge, and the slime by the specific gravity and measurement of the slime charges. If these operations are performed with proper precautions and due care, a close approximation to the correct tonnage may be obtained but it is very easy for serious errors to creep in. Regarding the sand charges the weight of a cubic foot of sand varies over wide limits according to the method of filling and the specific gravity of the ore The same ore may vary from 19 to 30 cubic feet per ton according to (1) whether the charge is measured in the tank in which it was collected or after being transferred to another tank, (2) (if measured after transferring) whether such transfer is made by shovelling or mechanical devices, (3) whether such measurement of transferred sand is made before or after applying solution, (4) (if measured in the collecting tank) whether it be collected by one or another of the various methods described, (5) whether the sand be charged dry or moist. Thus, in order to avoid error the weight per cubic foot should be carefully and frequently determined.

Another source of error may arise in the measurement of vertical depth of each charge owing to the fact that the surface is never level as a whole, nor is it free from small irregularities at the point of measurement making it difficult to decide within an inch or so at what point to begin measuring.

In the slime treatment it is difficult to estimate tonnage with the continuous system but in the charge system it may be done very closely, if the following precautions are observed:

1. The specific gravity of the dry slime must of course be accurately determined.

2. In ascertaining the ratio of liquid to solid in any given charge the sample taken must be representative. As a rule the thicker a pulp is, when in agitation, the more homogeneous it is. From a slime charge with a dilution ratio of 3 or 4 to 1 in an ordinary mechanical stirring tank, samples may be taken at different points which show widely differing ratios of liquid to solid.

3. Correct measurements of the cubic content of the charge must be made. This cannot be done while the charge is in motion. Moreover, allowance has to be made for permanent solid material occuping space in the tank, especially hard rings or cones of slime, sand, etc., and also (in the case of flat bottom tanks) a certain depth of treated pulp which it is impossible to remove each time the tank is emptied.

Tonnage of Slime Pulp, as Ascertained by Density Determination.—This method depends on the cubic capacity of the slime charge and its density as determined by specific gravity test. Specific gravity may be explained as the ratio between the weight of a given substance and the weight of the volume of distilled water at 4 degrees C. displaced by it. The first requisite is to find the specific gravity of the dry ore-pulp. For this purpose dry some of the powdered material over a water bath or at a temperature not exceeding 100 deg. C. (212 F.), and at once weigh up carefully 250 grams and place in a dry litre measuring flask which has been previously weighed and the weight recorded. Then add about 500 cc of distilled water and mix well. The dry ore nearly always contains a large quantity of air adhering to the particles which is very difficult to detach and therefore Clennell[1] recommends immersing the flask in boiling water and keeping it at that temperature for from one to two hours in order to expel all the air. It is then cooled, and distilled water added up to the mark, after which the flask with contents is weighed and the weight of the empty flask deducted, the object being to ascertain the weight of the water which has been displaced by the ore.

To do this the weight of a similar volume of distilled water

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[1] The Cyanide Handbook, page 557 (Second Edition).

must be known. A litre flask contains nominally 1000 grams of water but it is better to empty the ore pulp out of the flask and after washing well to refill with distilled water at the same temperature as was the water used for mixing with the ore sample, and after drying the exterior of the flask to weigh carefully. This figure less the recorded weight of the dry flask is then compared with the weight of the same volume of the ore-and-water mixture previously found.

Let a = weight of dry ore taken
b = weight of water in the flask when filled to the mark with water alone
c = weight of mixture of ore and water at the same mark
d = weight of water in the mixture
w = weight of water displaced by the dry ore
s = specific gravity of dry ore

Then $$c - a = d$$
and $$b - d = w$$
and $$s = \frac{a}{w}$$
or $$s = \frac{a}{a + b - c}$$

If great precision is required, the mixture of ore and water, and also the water alone, should be cooled to 4 deg. C. in the flask and weighed at that temperature, the level of course touching the mark in each case, but for ordinary purposes it is sufficient that the temperature at both weighings should be that of the atmosphere.

To determine the specific gravity of the wet pulp composing the slime charge it is more convenient to use a litre flask whose neck has been cut off about an inch above the mark, or better still to procure such a flask properly finished off by the manufacturer. This is weighed empty and then filled to the mark with the pulp to be tested. A representative sample should first be taken in a bucket and the ratio of water to ore should be low, that is, the pulp should be as thick as possible to ensure homogeneity. The sample is then thoroughly mixed and kept in agitation while portions are dipped out with a small beaker, and poured into the flask taking care that no settlement takes place in the beaker during the operation. This is continued until the level reaches the mark on the neck. The flask is then dried (if wet on the outside) and weighed, and the weight of the empty flask deducted from the figure found.

Then if W_1 = Weight of pulp.
W_2 = Weight of same volume of water.
$\frac{W_1}{W_2}$ = Specific gravity of the pulp.

For calculating the quantity of ore in the charge from this figure, Clennell[1] gives the following formula:

$$V^1 = \frac{32\,(S-1)}{S(P-1)} \text{ or } V = \frac{35.31(S-1)}{S\,(P-1)}$$

where V^1 = Number of cu. ft. of pulp to 1 short ton of dry slime.
V = Number of cu. ft. of pulp to 1 metric ton of dry slime.
S = Specific gravity of dry slime.
1 = Specific gravity of solution.
P = Specific gravity of pulp.

W. A. Caldecott's pulp table with one or two slight changes in the formulas is here given in full, based on a specific gravity of 2.6 for the dry ore. A second table follows, showing the changes in the various formulas in the preceding table for specific gravities from 2.5 to 3.5, so that a complete table may be drawn up for each mill according to the specific gravity of its ore.

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[1] The Cyanide Handbook, page 559 (Second Edition).

TABLE 6.—WEIGHTS & VOLUMES OF SLIME PULP PER TON OF 2000 LB.
Let sp. gr. of dry slimes = 2.6. Let per cent. of dry slimes = y.
Let sp. gr. of pulp = x.

Per cent. dry slimes in pulp	Sp. gr. of pulp	Weight of 1 cu. ft. of pulp		Cu. ft. of pulp to 1 ton	Cu. ft. pulp to 1 ton of dry slimes	Per cent. dry slimes in pulp	Sp. gr. of pulp	Weight of 1 cu. ft. of pulp		Cu. ft. of pulp to 1 ton	Cu. ft. pulp to 1 ton of dry slimes
		Tons	Pounds					Tons	Pounds		
$\frac{260(x-1)}{1.6x}$	$\frac{260}{260-1.6y}$	$\frac{62.5x}{2000}$	$62.5x$	$\frac{2000}{62.5x}$	$\frac{12.3(260-1.6y)}{y}$	$\frac{260(x-1)}{1.6x}$	$\frac{260}{260-1.6y}$	$\frac{62.5x}{2000}$	$62.5x$	$\frac{2000}{62.5x}$	$\frac{12.3(260-1.6y)}{y}$
0	1.00000	.031250	62.5000	32.0000	∝						
1	1.00619	.031443	62.8869	31.8031	3180.314	26	1.19047	.037202	74.4044	26.8802	103.385
2	1.01246	.031639	63.2788	31.6062	1580.309	27	1.19926	.037477	74.9538	26.6831	98.826
3	1.01878	.031837	63.6738	31.4101	1046.976	28	1.20818	.037756	75.5113	26.4862	94.593
4	1.02524	.032039	64.0775	31.2122	780.307	29	1.21723	.038038	76.0769	26.2893	90.653
5	1.03174	.032242	64.4838	31.0156	620.309	30	1.22641	.038325	76.6506	26.0925	86.975
6	1.03834	.032448	64.8963	30.8184	513.641	31	1.23574	.038617	77.2338	25.8954	83.534
7	1.04502	.032657	65.3138	30.6214	437.451	32	1.24521	.038913	77.8256	25.6985	80.308
8	1.05178	.032868	65.7363	30.4245	380.308	33	1.25482	.039213	78.4263	25.5016	77.278
9	1.05863	.033082	66.1644	30.2278	335.863	34	1.26459	.039518	79.0369	25.3046	74.425
10	1.06557	.033299	66.5981	30.0310	300.308	35	1.27451	.039828	79.6569	25.1076	71.736
11	1.07260	.033519	67.0375	29.8340	271.217	36	1.28458	.040143	80.2863	24.9109	69.197
12	1.07973	.033742	67.4831	29.6370	246.974	37	1.29482	.040463	80.9263	24.7138	66.794
13	1.08696	.033969	67.9375	29.4388	226.462	38	1.30522	.040788	81.5763	24.5170	64.518
14	1.09428	.034193	68.3925	29.2430	208.879	39	1.31579	.041118	82.2369	24.3199	62.359
15	1.10169	.034428	68.8556	29.0462	193.641	40	1.32653	.041454	82.9081	24.1231	60.308
16	1.10921	.034663	69.3256	28.8493	180.308	41	1.33745	.041795	83.5906	23.9262	58.356
17	1.11684	.034901	69.8025	28.6523	168.543	42	1.34855	.042142	84.2844	23.7291	56.498
18	1.12457	.035143	70.2856	28.4553	158.085	43	1.35983	.042495	84.9894	23.5323	54.726
19	1.13240	.035388	70.7750	28.2586	148.729	44	1.37131	.042853	85.7069	23.3353	53.035
20	1.14035	.035636	71.2719	28.0607	140.308	45	1.38298	.043218	86.4363	23.1384	51.419
21	1.14841	.035888	71.7756	27.8646	132.689	46	1.39485	.043589	87.1781	22.9416	49.873
22	1.15658	.036143	72.2863	27.6678	125.762	47	1.40693	.043967	87.9331	22.7446	48.393
23	1.16487	.036402	72.8044	27.4709	119.438	48	1.41922	.044351	88.7013	22.5476	46.974
24	1.17328	.036665	73.3300	27.2740	113.641	49	1.43172	.044741	89.4825	22.3507	45.614
25	1.18182	.036932	73.8638	27.0769	108.308	50	1.4444	.045139	90.2775	22.1539	44.308

WEIGHTS & VOLUMES OF SLIME PULP PER TON OF 2000 LB.—*Continued*

Per cent. dry slimes in pulp	Sp. gr. of pulp	Weight of 1 cu. ft. of pulp		Cu. ft. of pulp to 1 ton	Cu. ft. pulp to 1 ton of dry slimes	Per cent. dry slimes in pulp	Sp. gr. of pulp	Weight of 1 cu. ft. of pulp		Cu. ft. of pulp to 1 ton	Cu. ft. pulp to 1 ton of dry slimes
		Tons	Pounds					Tons	Pounds		
$\frac{260(x-1)}{1.6x}$	$\frac{260}{260-1.6y}$	$\frac{62.5x}{2000}$	$62.5x$	$\frac{2000}{62.5x}$	$\frac{12.3(260-1.6y)}{y}$	$\frac{260(x-1)}{1.6x}$	$\frac{260}{260-1.6y}$	$\frac{62.5x}{2000}$	$62.5x$	$\frac{2000}{62.5x}$	$\frac{12.3(260-1.6y)}{y}$
51	1.45740	.045544	91.0375	21.9569	43.053	76	1.87861	.058707	117.413	17.0338	22.413
52	1.47059	.045956	91.9119	21.7600	41.846	77	1.90058	.059393	118.786	16.8370	21.866
53	1.48402	.046376	92.7513	21.5630	40.685	78	1.92308	.060096	120.193	16.6401	21.333
54	1.49769	.046803	93.6056	21.3663	39.567	79	1.94611	.060816	121.632	16.4430	20.814
55	1.51163	.047238	94.4769	21.1692	38.489	80	1.96970	.061553	123.106	16.2462	20.308
56	1.52582	.047682	95.3338	20.9724	37.451	81	1.99386	.062308	124.616	16.0493	19.814
57	1.54028	.048134	96.2675	20.7755	36.448	82	2.01863	.063082	126.164	15.8524	19.322
58	1.55503	.048595	97.1894	20.5784	35.480	83	2.04402	.063876	127.751	15.6555	18.862
59	1.57005	.049064	98.1281	20.3815	34.545	84	2.07006	.064689	129.379	15.4585	18.403
60	1.58536	.049543	99.0850	20.1847	33.641	85	2.09677	.065524	131.048	15.2616	17.955
61	1.60099	.050031	100.062	19.9876	32.767	86	2.12418	.066381	132.761	15.0647	17.517
62	1.61691	.050528	101.057	19.7908	31.921	87	2.15232	.067260	134.520	14.8677	17.089
63	1.63317	.051037	102.073	19.5938	31.101	88	2.18121	.068163	136.326	14.6698	16.671
64	1.64975	.051555	103.109	19.3969	30.308	89	2.21088	.069090	138.180	14.4739	16.263
65	1.66666	.052083	104.166	19.2001	29.539	90	2.24138	.070043	140.086	14.2769	15.863
66	1.68394	.052623	105.246	19.0031	28.793	91	2.27273	.071023	142.046	14.0803	15.473
67	1.70157	.053174	106.348	18.8062	28.069	92	2.30496	.072030	144.060	13.8831	15.090
68	1.71958	.053737	107.474	18.6092	27.367	93	2.33812	.073066	146.133	13.6862	14.716
69	1.73796	.054311	108.623	18.4123	26.685	94	2.37226	.074133	148.266	13.4893	14.350
70	1.75675	.054393	109.797	18.2155	26.022	95	2.40741	.075232	150.463	13.2923	13.992
71	1.77596	.055499	110.998	18.0185	25.378	96	2.44361	.076363	152.726	13.0953	13.641
72	1.79558	.056112	112.224	17.8215	24.752	97	2.48091	.077528	155.057	12.8985	13.297
73	1.81564	.056739	113.478	17.6246	24.143	98	2.51938	.078731	157.461	12.7015	12.961
74	1.83616	.057380	114.760	17.4277	23.551	99	2.55905	.079970	159.941	12.5046	12.631
75	1.85714	.058036	116.071	17.2308	22.974	100	2.60000	.081250	162.500	12.3077	12.308

TABLE 7

Shows changes in formula for ores of different specific gravities based on 2000-lb. ton.

Let x = specific gravity of pulp. Let y = per cent. of dry slime.

Sp. gr. of dry slime	Per cent. of dry slime in pulp	Sp. gr. of pulp	Cu. ft. pulp to 1 ton	Cu. ft. pulp to 1 ton dry slime
2.50	$\frac{250(x-1)}{1.5x}$	$\frac{250}{250-1.5y}$	$\frac{2000}{62.5x}$	$\frac{12.80(250-1.5y)}{y}$
2.52	$\frac{252(x-1)}{1.52x}$	$\frac{252}{252-1.52y}$	$\frac{2000}{62.5x}$	$\frac{12.70(252-1.52y)}{y}$
2.54	$\frac{254(x-1)}{1.54x}$	$\frac{254}{254-1.54y}$	$\frac{2000}{62.5x}$	$\frac{12.60(254-1.54y)}{y}$
2.56	$\frac{256(x-1)}{1.56x}$	$\frac{256}{256-1.56y}$	$\frac{2000}{62.5x}$	$\frac{12.50(256-1.56y)}{y}$
2.58	$\frac{258(x-1)}{1.58x}$	$\frac{258}{258-1.58y}$	$\frac{2000}{62.5x}$	$\frac{12.40(258-1.58y)}{y}$
2.60	$\frac{260(x-1)}{1.6x}$	$\frac{260}{260-1.6y}$	$\frac{2000}{62.5x}$	$\frac{12.31(260-1.6y)}{y}$
2.62	$\frac{262(x-1)}{1.62x}$	$\frac{262}{262-1.62y}$	$\frac{2000}{62.5x}$	$\frac{12.21(262-1.62y)}{y}$
2.64	$\frac{264(x-1)}{1.64x}$	$\frac{264}{264-1.64y}$	$\frac{2000}{62.5x}$	$\frac{12.12(264-1.64y)}{y}$
2.66	$\frac{266(x-1)}{1.66x}$	$\frac{266}{266-1.66y}$	$\frac{2000}{62.5x}$	$\frac{12.03(266-1.66y)}{y}$
2.68	$\frac{268(x-1)}{1.68x}$	$\frac{268}{268-1.68y}$	$\frac{2000}{62.5x}$	$\frac{11.94(268-1.68y)}{y}$
2.70	$\frac{270(x-1)}{1.7x}$	$\frac{270}{270-1.7y}$	$\frac{2000}{62.5x}$	$\frac{11.85(270-1.7y)}{y}$
2.72	$\frac{272(x-1)}{1.72x}$	$\frac{272}{272-1.72y}$	$\frac{2000}{62.5x}$	$\frac{11.76(272-1.72y)}{y}$
2.74	$\frac{274(x-1)}{1.74x}$	$\frac{274}{274-1.74y}$	$\frac{2000}{62.5x}$	$\frac{11.68(274-1.74y)}{y}$
2.76	$\frac{276(x-1)}{1.76x}$	$\frac{276}{276-1.76y}$	$\frac{2000}{62.5x}$	$\frac{11.59(276-1.76y)}{y}$
2.78	$\frac{278(x-1)}{1.78x}$	$\frac{278}{278-1.78y}$	$\frac{2000}{62.5x}$	$\frac{11.51(278-1.78y)}{y}$
2.80	$\frac{280(x-1)}{1.8x}$	$\frac{280}{280-1.8y}$	$\frac{2000}{62.5x}$	$\frac{11.42(280-1.8y)}{y}$
2.82	$\frac{282(x-1)}{1.82x}$	$\frac{282}{282-1.82y}$	$\frac{2000}{62.5x}$	$\frac{11.35(282-1.82y)}{y}$
2.84	$\frac{284(x-1)}{1.84x}$	$\frac{284}{284-1.84y}$	$\frac{2000}{62.5x}$	$\frac{11.26(284-1.84y)}{y}$
2.86	$\frac{286(x-1)}{1.86x}$	$\frac{286}{286-1.86y}$	$\frac{2000}{62.5x}$	$\frac{11.18(286-1.86y)}{y}$
2.88	$\frac{288(x-1)}{1.88x}$	$\frac{288}{288-1.88y}$	$\frac{2000}{62.5x}$	$\frac{11.11(288-1.88y)}{y}$
2.90	$\frac{290(x-1)}{1.9x}$	$\frac{290}{290-1.9y}$	$\frac{2000}{62.5x}$	$\frac{11.03(290-1.9y)}{y}$
2.92	$\frac{292(x-1)}{1.92x}$	$\frac{292}{292-1.92y}$	$\frac{2000}{62.5x}$	$\frac{10.95(292-1.92y)}{y}$
2.94	$\frac{294(x-1)}{1.94x}$	$\frac{294}{294-1.94y}$	$\frac{2000}{62.5x}$	$\frac{10.88(294-1.94y)}{y}$
2.96	$\frac{296(x-1)}{1.96x}$	$\frac{296}{296-1.96y}$	$\frac{2000}{62.5x}$	$\frac{10.81(296-1.96y)}{y}$
2.98	$\frac{298(x-1)}{1.98x}$	$\frac{298}{298-1.98y}$	$\frac{2000}{62.5x}$	$\frac{10.73(298-1.98y)}{y}$
3.00	$\frac{300(x-1)}{2x}$	$\frac{300}{300-2y}$	$\frac{2000}{62.5x}$	$\frac{10.66(300-2y)}{y}$

TABLE 7.—(*Continued*)

Sp. gr. of dry slime	Per cent. of dry slime in pulp	Sp. gr. of pulp	Cu. ft. pulp to 1 ton	Cu. ft. pulp to 1 ton dry slime
3.02	$\frac{302(x-1)}{2.02x}$	$\frac{302}{302-2.02y}$	$\frac{2000}{62.5x}$	$\frac{10.59(302-2.02y)}{y}$
3.04	$\frac{304(x-1)}{2.04x}$	$\frac{304}{304-2.04y}$	$\frac{2000}{62.5x}$	$\frac{10.52(304-2.04y)}{y}$
3.06	$\frac{306(x-1)}{2.06x}$	$\frac{306}{306-2.06y}$	$\frac{2000}{62.5x}$	$\frac{10.45(306-2.06y)}{y}$
3.08	$\frac{308(x-1)}{2.08x}$	$\frac{308}{308-2.08y}$	$\frac{2000}{62.5x}$	$\frac{10.38(308-2.08y)}{y}$
3.10	$\frac{310(x-1)}{2.10x}$	$\frac{310}{310-2.1y}$	$\frac{2000}{62.5x}$	$\frac{10.32(310-2.10y)}{y}$
3.12	$\frac{312(x-1)}{2.12x}$	$\frac{312}{312-2.12y}$	$\frac{2000}{62.5x}$	$\frac{10.25(312-2.12y)}{y}$
3.14	$\frac{314(x-1)}{2.14x}$	$\frac{314}{314-2.14y}$	$\frac{2000}{62.5x}$	$\frac{10.19(314-2.14y)}{y}$
3.16	$\frac{316(x-1)}{2.16x}$	$\frac{316}{316-2.16y}$	$\frac{2000}{62.5x}$	$\frac{10.12(316-2.16y)}{y}$
3.18	$\frac{318(x-1)}{2.18x}$	$\frac{318}{318-2.18y}$	$\frac{2000}{62.5x}$	$\frac{10.06(318-2.18y)}{y}$
3.20	$\frac{320(x-1)}{2.20x}$	$\frac{320}{320-2.2y}$	$\frac{2000}{62.5x}$	$\frac{10.00(320-2.2y)}{y}$
3.22	$\frac{322(x-1)}{2.22x}$	$\frac{322}{322-2.22y}$	$\frac{2000}{62.5x}$	$\frac{9.93(322-2.22y)}{y}$
3.24	$\frac{324(x-1)}{2.24x}$	$\frac{324}{324-2.24y}$	$\frac{2000}{62.5x}$	$\frac{9.87(324-2.24y)}{y}$
3.26	$\frac{326(x-1)}{2.26x}$	$\frac{326}{326-2.26y}$	$\frac{2000}{62.5x}$	$\frac{9.81(326-2.26y)}{y}$
3.28	$\frac{328(x-1)}{2.28x}$	$\frac{328}{328-2.28y}$	$\frac{2000}{62.5x}$	$\frac{9.75(328-2.28y)}{y}$
3.30	$\frac{330(x-1)}{2.3x}$	$\frac{330}{330-2.3y}$	$\frac{2000}{62.5x}$	$\frac{9.69(330-2.3y)}{y}$
3.32	$\frac{332(x-1)}{2.32x}$	$\frac{332}{332-2.32y}$	$\frac{2000}{62.5x}$	$\frac{9.63(332-2.32y)}{y}$
3.34	$\frac{334(x-1)}{2.34x}$	$\frac{334}{334-2.34y}$	$\frac{2000}{62.5x}$	$\frac{9.58(334-2.34y)}{y}$
3.36	$\frac{336(x-1)}{2.36x}$	$\frac{336}{336-2.36y}$	$\frac{2000}{62.5x}$	$\frac{9.52(336-2.36y)}{y}$
3.38	$\frac{338(x-1)}{2.38x}$	$\frac{338}{338-2.38y}$	$\frac{2000}{62.5x}$	$\frac{9.46(338-2.38y)}{y}$
3.40	$\frac{340(x-1)}{2.40x}$	$\frac{340}{340-2.4y}$	$\frac{2000}{62.5x}$	$\frac{9.41(340-2.4y)}{y}$
3.42	$\frac{342(x-1)}{2.42x}$	$\frac{342}{342-2.42y}$	$\frac{2000}{62.5x}$	$\frac{9.35(342-2.42y)}{y}$
3.44	$\frac{344(x-1)}{2.44x}$	$\frac{344}{344-2.44y}$	$\frac{2000}{62.5x}$	$\frac{9.3(344-2.44y)}{y}$
3.46	$\frac{346(x-1)}{2.46x}$	$\frac{346}{346-2.46y}$	$\frac{2000}{62.5x}$	$\frac{9.24(346-2.46y)}{y}$
3.48	$\frac{348(x-1)}{2.48x}$	$\frac{348}{348-2.48y}$	$\frac{2000}{62.5x}$	$\frac{9.19(348-2.48y)}{y}$
3.50	$\frac{350(x-1)}{2.5x}$	$\frac{350}{350-2.5y}$	$\frac{2000}{62.5x}$	$\frac{9.14(350-2.5y)}{y}$

Table 8

Shows change in formula for ores of different specific gravities based on the metric ton. The columns showing "per cent of dry slime in pulp" and "sp. gr. of pulp" will remain as in Table 7. For "cu. ft. of pulp to 1 ton" substitute the formula

$$\frac{2204.6}{62.5x}$$

For "cu. ft. pulp to 1 ton dry slime" substitute the following.

Sp. gr. of dry slime	Cu. ft. to 1 metric ton of dry slime	Sp. gr. of dry slime	Cu. ft. to 1 metric ton of dry slime
2.50	$\frac{14.128(250 - 1.50y)}{y}$	2.98	$\frac{11.852(298 - 1.98y)}{y}$
2.52	$\frac{14.015(252 - 1.52y)}{y}$	3.00	$\frac{11.773(300 - 2.00y)}{y}$
2.54	$\frac{13.905(254 - 1.54y)}{y}$	3.02	$\frac{11.695(302 - 2.02y)}{y}$
2.56	$\frac{13.796(256 - 1.56y)}{y}$	3.04	$\frac{11.618(304 - 2.04y)}{y}$
2.58	$\frac{13.690(258 - 1.58y)}{y}$	3.06	$\frac{11.542(306 - 2.06y)}{y}$
2.60	$\frac{13.584(260 - 1.60y)}{y}$	3.08	$\frac{11.467(308 - 2.08y)}{y}$
2.62	$\frac{13.480(262 - 1.62y)}{y}$	3.10	$\frac{11.393(310 - 2.10y)}{y}$
2.64	$\frac{13.378(264 - 1.64y)}{y}$	3.12	$\frac{11.320(312 - 2.12y)}{y}$
2.66	$\frac{13.278(266 - 1.66y)}{y}$	3.14	$\frac{11.248(314 - 2.14y)}{y}$
2.68	$\frac{13.178(268 - 1.68y)}{y}$	3.16	$\frac{11.177(316 - 2.16y)}{y}$
2.70	$\frac{13.081(270 - 1.70y)}{y}$	3.18	$\frac{11.107(318 - 2.18y)}{y}$
2.72	$\frac{12.985(272 - 1.72y)}{y}$	3.20	$\frac{11.037(320 - 2.20y)}{y}$
2.74	$\frac{12.890(274 - 1.74y)}{y}$	3.22	$\frac{10.969(322 - 2.22y)}{y}$
2.76	$\frac{12.797(276 - 1.76y)}{y}$	3.24	$\frac{10.901(324 - 2.24y)}{y}$
2.78	$\frac{12.705(278 - 1.78y)}{y}$	3.26	$\frac{10.834(326 - 2.26y)}{y}$
2.80	$\frac{12.614(280 - 1.80y)}{y}$	3.28	$\frac{10.768(328 - 2.28y)}{y}$
2.82	$\frac{12.525(282 - 1.82y)}{y}$	3.30	$\frac{10.703(330 - 2.30y)}{y}$
2.84	$\frac{12.436(284 - 1.84y)}{y}$	3.32	$\frac{10.638(332 - 2.32y)}{y}$
2.86	$\frac{12.349(286 - 1.86y)}{y}$	3.34	$\frac{10.575(334 - 2.34y)}{y}$
2.88	$\frac{12.263(288 - 1.88y)}{y}$	3.36	$\frac{10.512(336 - 2.36y)}{y}$
2.90	$\frac{12.179(290 - 1.90y)}{y}$	3.38	$\frac{10.449(338 - 2.38y)}{y}$
2.92	$\frac{12.095(292 - 1.92y)}{y}$	3.40	$\frac{10.388(340 - 2.40y)}{y}$
2.94	$\frac{12.013(294 - 1.94y)}{y}$		
2.96	$\frac{11.932(296 - 1.96y)}{y}$		

Possible Error Due to Density of Solution.—A source of error that may in certain circumstances assume more than negligible proportions lies in the fact that, in most of the calculations of slime density, the specific gravity of the solution is taken to be 1, as it would be if it were pure water, but it is obvious that the various substances in solution, such as lime, cyanide, zinc, and soluble constituents derived from the ore, will raise the specific gravity above that of water. In the case of simple clean ores, which require only weak solutions of cyanide and lime, the error due to this cause may be neglected but when dealing with more complex ores and more concentrated solutions it may materially modify the monthly ore tonnage as determined by this method. An article by G. H. Clevenger, H. W. Young, and T. N. Turner, in the Engineering and Mining Journal of December 19, 1914, deals with this subject in considerable detail.

A list is there given of the specific gravities of the solution from a number of mills, showing a variation between 1.002 and 1.01, and two instances are cited to show the difference due to this cause.

> "Taking the specific gravity of the dry slime as 2.66 and the weight of a litre of the wet pulp as 1260 grams (sp. gr. 1.26).
>
> (*a*) If sp. gr. of solution = 1.0, then the percentage of dry slime = 33.07%.
>
> (*b*) If sp. gr. of solution = 1.008, then the percentage of dry slime = 32.2.
>
> Showing a percentage of difference of 2.63.
>
> Again, taking sp. gr. of dry slime as 2.66 and the weight of one litre of the wet pulp as 1100 grams (sp. gr. 1.1).
>
> (*a*) Taking the sp. gr. of the solution as 1, the percentage of dry slime = 14.56%.
>
> (*b*) Taking the sp. gr. of the solution as 1.008, the percentage of dry slime = 13.47%.
>
> The percentage of difference in this case would be 7.49."

So, when the stock solution contains an unusually large quantity of soluble salts, thus raising its specific gravity considerably above that of water, the error due to this cause should not be over-looked. The above-named article gives the following table for ascertaining the correction to be made in each determination on account of the specific gravity of the solution in the pulp.

The authors thus describe its use:

> "Let the specific gravity of the dry slime = 2.66; the weight of one litre of pulp = 1150 grams; the specific gravity of solution = 1.005. Required: The number of cubic feet of pulp to 1 ton of dry slime in the charge. In Table 9, at the intersection of the vertical column headed 2.66 and the horizontal column headed 1150, is found 133.3, the number of cubic feet of pulp to make one ton of dry slime when the specific gravity of the solution is 1. In the adjoining column opposite 133.3 is found 0.87, the number of cubic feet to be added to 133.3 for each increase of 0.001 in the specific gravity of the solution: 0.87 × 5 = 4.35 (total correction): 133.3 + 4.35 = 137.65 cu. ft. of the pulp specified necessary to make one ton of dry slime."

Moisture Determination Preferred in Certain Cases.—When a thick pulp is aerated by compressed air it often happens that the amount of air entrapped in the form of minute bubbles renders the Specific Gravity determination quite unreliable. By allowing the flask of pulp to stand for a few hours the level may fall from 1 to 2 inches, due to the escaping air, but a large proportion of the entrapped air will be held permanently thus increasing the volume to that extent. In such cases the only remedy seems to be to make a moisture determination by evaporation. It is not enough, however, to determine the relative weight of ore and moisture in a sample of given weight, because the contents of the tank are to be estimated by volume and that volume is occupied by a mixture of ore, moisture, and air, so the weight of pulp taken for the moisture sample should have its volume also determined in terms of cu. ft. in order that the weight of ore in a cubic foot may be accurately determined.

For instance:

1 Litre = 61.031 cu. inches or 0.03531 cu. ft.
1 Cu. ft. = 28.3174 litres

So if 1 litre of pulp be weighed and dried the weight of ore and water in a cu. ft. of capacity can be found.

TABLE 9.—NUMBER OF CUBIC FEET OF WET

Formula: $\dfrac{2000}{62.42\,a\,\dfrac{P}{100}}$

Specific Gravity of Dry Slime

Weight in Grams of One Litre of Wet Pulp	2.50	Correction	2.52	Correction	2.54	Correction	2.56	Correction	2.58	Correction	2.60	Correction
1010	1923		1933		1943		1953		1963		1972	
1020	961.4	94.9	966.3	95.4	971.4	95.9	976.4	96.4	981.2	96.9	985.8	97.3
1030	640.9	31.41	644.2	31.59	647.6	31.76	650.9	31.93	654.1	32.09	657.2	32.26
1040	480.6	15.60	483.1	15.69	485.6	15.78	488.1	15.87	490.5	15.95	492.9	16.03
1050	384.5	9.29	386.5	9.35	388.5	9.40	390.5	9.45	392.4	9.50	394.3	9.55
1060	320.4	6.15	322.0	6.19	323.7	6.23	325.4	6.26	327.0	6.29	328.5	6.33
1070	274.7	4.36	276.1	4.39	277.5	4.42	278.9	4.44	280.3	4.46	281.7	4.48
1080	240.3	3.25	241.5	3.28	242.8	3.30	244.1	3.32	245.3	3.33	246.4	3.35
1090	213.6	2.51	214.7	2.53	215.9	2.54	217.0	2.55	218.1	2.56	219.1	2.58
1100	192.3	1.99	193.3	2.00	194.3	2.02	195.3	2.03	196.2	2.05	197.2	2.06
1110	174.8	1.62	175.7	1.63	176.6	1.64	177.5	1.65	178.4	1.66	179.2	1.67
1120	160.2	1.34	161.0	1.35	161.9	1.36	162.7	1.37	163.5	1.38	164.3	1.38
1130	147.9	1.12	148.7	1.13	149.4	1.14	150.2	1.14	151.0	1.15	151.7	1.16
1140	137.3	.96	138.0	.96	138.8	.97	139.5	.97	140.2	.98	140.8	.99
1150	128.2	.82	128.8	.83	129.5	.84	130.2	.84	130.8	.85	131.4	.85
1160	120.2	.71	120.8	.72	121.4	.73	122.0	.73	122.6	.74	123.2	.74
1170	113.1	.63	113.7	.63	114.3	.63	114.9	.63	115.5	.64	116.0	.64
1180	106.8	.56	107.4	.56	107.9	.57	108.5	.57	109.0	.57	109.5	.57
1190	101.2	.49	101.7	.49	102.2	.50	102.8	.50	103.3	.50	103.8	.50
1200	96.14	.44	96.63	.44	97.14	.45	97.64	.45	98.12	.45	98.58	.45
1210	91.56	.394	92.03	.398	92.51	.401	92.98	.404	93.44	.406	93.89	.409
1220	87.39	.356	87.85	.359	88.31	.361	88.76	.363	89.20	.365	89.63	.368
1230	83.59	.323	84.03	.325	84.47	.327	84.90	.329	85.32	.331	85.73	.333
1240	80.11	.293	80.53	.296	80.95	.298	81.36	.300	81.76	.302	82.16	.304
1250	76.91	.266	77.31	.269	77.71	.272	78.11	.274	78.50	.275	78.88	.277
1260	73.94	.246	74.32	.248	74.71	.250	75.09	.252	75.47	.253	75.83	.255
1270	71.20	.225	71.57	.227	71.94	.229	72.31	.231	72.67	.232	73.02	.234
1280	68.66	.207	69.01	.209	69.37	.211	69.73	.212	70.08	.214	70.41	.216
1290	66.30	.190	66.64	.192	66.99	.194	67.33	.195	67.67	.197	67.99	.198
1300	64.09	.177	64.42	.179	64.76	.180	65.09	.181	65.41	.183	65.72	.184
1310	62.02	.165	62.33	.167	62.66	.168	62.98	.169	63.30	.171	63.60	.172
1320	60.09	.151	60.40	.153	60.71	.154	61.02	.155	61.33	.157	61.63	.158
1330	58.26	.143	58.56	.144	58.87	.145	59.17	.146	59.47	.148	59.75	.149
1340	56.54	.134	56.83	.135	57.13	.136	57.42	.137	57.71	.138	57.99	.139
1350	54.93	.124	55.21	.125	55.50	.126	55.78	.127	56.06	.128	56.33	.129
1360	53.41	.116	53.68	.117	53.96	.118	54.24	.119	54.51	.120	54.77	.121
1370	51.96	.109	52.23	.110	52.50	.111	52.77	.112	53.04	.113	53.29	.114
1380	50.59	.103	50.85	.103	51.12	.104	51.38	.105	51.64	.106	51.88	.107
1390	49.29	.097	49.55	.098	49.81	.099	50.06	.099	50.31	.100	50.55	.101
1400	48.06	.091	48.31	.092	48.56	.093	48.81	.093	49.05	.094	49.29	.095
1410	46.89	.086	47.13	.087	47.38	.088	47.62	.088	47.86	.089	48.09	.090
1420	45.78	.081	46.02	.082	46.26	.083	46.49	.083	46.72	.084	46.95	.085
1430	44.71	.077	44.94	.078	45.17	.079	45.40	.079	45.63	.080	45.85	.081
1440	43.69	.072	43.92	.073	44.15	.074	44.37	.074	44.59	.075	44.81	.076
1450	42.73	.068	42.95	.069	43.17	.069	43.39	.070	43.61	.070	43.82	.071
1460	41.79	.065	42.01	.066	42.23	.066	42.44	.067	42.65	.067	42.86	.068
1470	40.91	.061	41.12	.061	41.33	.062	41.54	.063	41.75	.064	41.95	.065
1480	40.06	.058	40.26	.058	40.48	.059	40.68	.060	40.89	.060	41.08	.061
1490	39.24	.055	32.44	.056	39.65	.057	39.85	.057	40.05	.058	40.23	.059
1500	38.45	.053	38.65	.053	38.85	.054	39.05	.054	39.25	.055	39.43	.056

PULP TO MAKE ONE TON OF DRY SLIME

2.62	Correction	2.64	Correction	2.66	Correction	2.68	Correction	2.70	Correction	2.72	Correction	2.74	Correction
1981		1990		1999		2008		2017		2026		2034	
990.6	97.8	995.2	98.3	999.8	98.8	1004	99.3	1009	99.7	1013	100.1	1017	100.5
660.3	32.42	663.4	32.58	666.5	32.73	669.5	32.89	672.4	33.10	675.3	33.2	678.2	33.30
495.2	16.11	497.5	16.19	499.8	16.27	502.1	16.35	504.3	16.43	506.4	16.50	508.5	16.58
396.2	9.60	398.0	9.65	399.9	9.69	401.7	9.74	403.4	9.79	405.1	9.84	406.8	9.88
330.1	6.36	331.7	6.40	333.2	6.43	334.7	6.46	336.1	6.49	337.6	6.52	339.0	6.55
283.0	4.51	284.3	4.53	285.6	4.56	286.9	4.58	288.1	4.60	289.4	4.62	290.6	4.64
247.6	3.37	248.8	3.39	249.9	3.40	251.0	3.42	252.1	3.44	253.2	3.45	254.3	3.47
220.1	2.60	221.2	2.61	222.2	2.62	223.2	2.64	224.1	2.66	225.1	2.67	226.0	2.68
198.1	2.07	199.0	2.08	199.9	2.10	200.8	2.11	201.7	2.11	202.6	2.12	203.4	2.13
180.1	1.68	180.9	1.69	181.8	1.70	182.6	1.71	183.4	1.71	184.2	1.72	184.9	1.73
165.1	1.39	165.8	1.40	166.6	1.41	167.4	1.41	168.1	1.42	168.8	1.43	169.5	1.43
152.4	1.17	153.1	1.17	153.8	1.18	154.5	1.19	155.2	1.19	155.8	1.20	156.5	1.20
141.5	1.00	142.2	1.00	142.8	1.01	143.5	1.01	144.1	1.02	144.7	1.03	145.3	1.03
132.1	.85	132.7	.86	133.3	.87	133.9	.87	134.5	.88	135.0	.89	135.6	.89
123.8	.75	124.4	.75	125.0	.75	125.5	.76	126.1	.76	126.6	.77	127.1	.77
116.5	.65	117.1	.65	117.6	.66	118.2	.66	118.7	.66	119.2	.67	119.7	.67
110.1	.57	110.6	.57	111.1	.58	111.6	.59	112.0	.59	112.5	.60	113.0	.60
104.3	.51	104.7	.51	105.2	.52	105.7	.52	106.1	.52	106.6	.53	107.1	.53
99.06	.45	99.52	.46	99.98	.46	100.4	.47	100.9	.47	101.3	.47	101.7	.47
94.34	.411	94.78	.414	95.21	.417	95.64	.420	96.06	.420	96.47	.42	96.88	.42
90.05	.371	90.47	.374	90.89	.376	91.30	.378	91.70	.381	92.09	.383	92.48	.385
86.14	.336	86.54	.338	86.94	.340	87.33	.343	87.71	.345	88.09	.347	88.46	.349
82.55	.306	82.93	.308	83.31	.311	83.69	.313	84.05	.315	84.41	.317	84.77	.318
79.25	.279	79.62	.281	79.99	.283	80.35	.285	80.70	.287	81.04	.289	81.38	.290
76.19	.257	76.54	.259	76.90	.261	77.24	.263	77.58	.264	77.91	.266	78.24	.267
73.37	.235	73.71	.237	74.05	.239	74.38	.241	74.70	.242	75.02	.244	75.34	.245
70.74	.217	71.07	.219	71.40	.221	71.72	.222	72.03	.223	72.34	.225	72.64	.226
68.31	.200	68.63	.202	68.95	.203	69.26	.204	69.55	.206	69.85	.207	70.15	.208
66.03	.186	66.34	.188	66.65	.189	66.95	.190	67.24	.191	67.53	.192	67.81	.193
63.90	.173	64.19	.175	64.49	.177	64.78	.178	65.06	.179	65.34	.180	65.62	.181
61.92	.159	62.20	.161	62.49	.162	62.77	.163	63.04	.164	63.31	.165	63.58	.166
60.03	.150	60.31	.151	60.59	.152	60.87	.153	61.13	.154	61.39	.155	61.65	.156
58.26	.140	58.53	.141	58.80	.142	59.07	.143	59.32	.144	59.58	.145	59.83	.146
56.60	.130	56.86	.131	57.13	.132	57.39	.133	57.63	.134	57.88	.135	58.12	.136
55.03	.122	55.28	.123	55.54	.124	55.79	.125	56.03	.125	56.27	.126	56.51	.127
53.54	.115	53.79	.116	54.04	.117	54.28	.118	54.51	.119	54.75	.120	54.98	.121
52.13	.108	52.37	.108	52.62	.109	52.86	.110	53.09	.111	53.31	.112	53.53	.113
50.79	.102	51.03	.103	51.26	.104	51.49	.105	51.71	.106	51.93	.107	52.15	.107
49.52	.096	49.75	.096	49.98	.097	50.21	.098	50.42	.099	50.64	.100	50.85	.101
48.32	.090	48.54	.091	48.77	.092	48.99	.092	49.20	.093	49.41	.094	49.62	.094
47.17	.085	47.39	.086	47.61	.087	47.83	.087	48.03	.088	48.24	.089	48.44	.089
46.07	.081	46.28	.082	46.50	.083	46.71	.083	46.91	.084	47.11	.085	47.31	.085
45.02	.076	45.23	.077	45.44	.078	45.64	.078	45.84	.079	46.04	.080	46.23	.080
44.03	.071	44.23	.072	44.44	.072	44.64	.073	44.83	.073	45.02	.074	45.21	.075
43.06	.069	43.26	.070	43.46	.071	43.66	.071	43.85	.072	44.03	.073	44.22	.073
42.15	.065	42.35	.066	42.54	.066	42.73	.067	42.92	.067	43.10	.068	43.28	.068
41.28	.061	41.47	.062	41.66	.063	41.85	.063	42.02	.064	42.20	.065	42.38	.065
40.43	.059	40.62	.060	40.80	.061	40.99	.061	41.16	.062	41.34	.063	41.51	.063
39.62	.056	39.80	.057	39.99	.058	40.17	.058	40.34	.059	40.51	.059	40.68	.060

TABLE 9.—NUMBER OF CUBIC FEET OF WET

Formula: $$\frac{2000}{62.42\ a\ \frac{P}{100}}$$

Specific Gravity of Dry Slime

2.76	Correction	2.78	Correction	2.80	Correction	2.82	Correction	2.84	Correction	2.86	Correction	2.88	Correction
2043		2052		2060		2068		2076		2084		2092	
1022	100.9	1026	101.4	1030	101.8	1034	102.3	1038	102.7	1042	103.1	1046	103.5
681.1	33.5	683.9	33.6	686.7	33.7	689.4	33.9	692.0	34.1	694.6	34.2	697.2	34.3
510.7	16.65	512.8	16.72	514.9	16.79	516.9	16.87	518.9	16.94	520.9	17.00	522.9	17.07
408.6	9.92	410.3	9.97	411.9	10.02	413.5	10.07	415.1	10.11	416.7	10.14	418.3	10.18
340.5	6.58	341.9	6.61	343.2	6.64	344.6	6.67	345.9	6.70	347.2	6.73	348.6	6.75
291.9	4.66	293.1	4.68	294.3	4.70	295.5	4.72	296.6	4.75	297.7	4.77	298.8	4.79
255.4	3.48	256.4	3.50	257.5	3.51	258.5	3.53	259.5	3.55	260.5	3.56	261.5	3.58
227.0	2.69	228.0	2.70	228.9	2.71	229.8	2.72	230.7	2.74	231.5	2.76	232.4	2.77
204.3	2.14	205.2	2.15	206.0	2.16	206.8	2.17	207.6	2.18	208.4	2.19	209.2	2.20
185.7	1.74	186.5	1.75	187.2	1.76	188.0	1.77	188.7	1.78	189.4	1.79	190.2	1.79
170.3	1.44	171.0	1.45	171.6	1.46	172.3	1.47	172.9	1.48	173.6	1.48	174.3	1.48
157.2	1.21	157.8	1.21	158.5	1.22	159.1	1.23	159.7	1.24	160.3	1.24	160.9	1.24
145.9	1.04	146.5	1.04	147.1	1.05	147.7	1.05	148.3	1.06	148.8	1.06	149.4	1.06
136.2	.89	136.7	.90	137.3	.90	137.9	.90	138.4	.91	138.9	.91	139.4	.92
127.7	.77	128.2	.78	128.7	.79	129.2	.79	129.7	.80	130.2	.80	130.7	.80
120.2	.68	120.7	.68	121.2	.68	121.7	.69	122.1	.69	122.6	.69	123.1	.70
113.5	.60	114.0	.60	114.4	.61	114.9	.61	115.3	.62	115.7	.62	116.2	.62
107.5	.54	108.0	.54	108.4	.54	108.8	.54	109.2	.55	109.6	.55	110.1	.55
102.2	.47	102.6	.48	103.0	.48	103.4	.48	103.8	.49	104.2	.49	104.6	.49
97.29	.43	97.70	.43	98.10	.43	98.49	.43	98.87	.44	99.24	.44	99.61	.44
92.87	.387	93.26	.389	93.64	.391	94.01	.393	94.37	.395	94.73	.397	95.08	.399
88.83	.351	89.20	.353	89.56	.355	89.92	.357	90.27	.359	90.61	.361	90.95	.363
85.13	.320	85.48	.322	85.83	.324	86.17	.326	86.50	.328	86.82	.330	87.15	.332
81.73	.291	82.07	.293	82.40	.295	82.73	.297	83.05	.299	83.36	.300	83.67	.302
78.57	.269	78.90	.271	79.22	.273	79.54	.274	79.84	.276	80.13	.278	80.44	.279
75.67	.247	75.98	.249	76.28	.251	76.58	.252	76.88	.254	77.16	.255	77.46	.256
72.95	.228	73.26	.229	73.56	.230	73.85	.232	74.13	.233	74.41	.234	74.69	.236
70.45	.210	70.74	.211	71.02	.213	71.31	.214	71.58	.215	71.85	.216	72.12	.217
68.11	.195	68.39	.196	68.66	.197	68.94	.199	69.20	.200	69.46	.201	69.73	.202
65.90	.182	66.17	.183	66.44	.184	66.71	.186	66.96	.187	67.21	.188	67.46	.189
63.85	.167	64.12	.168	64.38	.169	64.64	.171	64.88	.172	65.12	.173	65.37	.174
61.91	.158	62.17	.159	62.42	.160	62.67	.161	62.91	.162	63.14	.163	63.39	.164
60.09	.147	60.34	.148	60.58	.149	60.83	.150	61.05	.151	61.28	.152	61.51	.152
58.37	.137	58.61	.138	58.85	.139	59.09	.140	59.31	.141	59.53	.142	59.76	.142
56.75	.128	56.99	.129	57.22	.130	57.45	.131	57.66	.132	57.88	.133	58.10	.133
55.22	.122	55.45	.123	55.67	.123	55.89	.124	56.11	.125	56.31	.126	56.53	.127
53.76	.114	53.99	.115	54.20	.116	54.42	.116	54.63	.117	54.83	.118	55.04	.119
52.38	.108	52.60	.109	52.81	.109	53.02	.110	53.22	.111	53.42	.111	53.63	.112
51.07	.102	51.29	.103	51.49	.103	51.70	.104	51.89	.105	52.08	.105	52.28	.106
49.83	.095	50.04	.096	50.24	.096	50.44	.097	50.63	.098	50.82	.098	51.01	.099
48.65	.090	48.85	.091	49.05	.091	49.25	.092	49.43	.093	49.61	.093	49.81	.094
47.51	.086	47.71	.087	47.90	.087	48.10	.088	48.29	.089	48.46	.089	48.64	.090
46.43	.081	46.62	.082	46.81	.082	47.00	.083	47.18	.084	47.35	.084	47.53	.085
45.40	.075	45.59	.076	45.78	.077	45.96	.078	46.13	.079	46.30	.079	46.48	.080
44.41	.074	44.59	.074	44.77	.075	44.95	.075	45.12	.076	45.29	.077	45.46	.077
43.47	.069	43.65	.069	43.82	.070	44.00	.071	44.17	.071	44.33	.072	44.50	.072
42.57	.066	42.75	.066	42.91	.067	43.09	.067	43.25	.068	43.41	.069	43.58	.069
41.70	.064	41.87	.065	42.03	.065	42.21	.065	42.36	.066	42.52	.066	42.69	.066
40.86	.060	41.03	.060	41.19	.061	41.36	.061	41.52	.062	41.67	.062	41.83	.062

PULP TO MAKE ONE TON OF DRY SLIME—(*Continued*)

2.90	Correction	2.92	Correction	2.94	Correction	2.96	Correction	2.98	Correction	3.00	Correction	Weight in Grams of One Litre of Wet Pulp
2099		2107		2115		2122		2129		2136		1010
1050	103.9	1053	104.3	1057	104.7	1061	105.0	1065	105.5	1068	105.8	1020
699.8	34.4	702.3	34.5	704.8	34.7	707.3	34.8	709.7	34.9	712.0	35.1	1030
524.8	17.14	526.7	17.21	528.6	17.27	530.4	17.33	532.2	17.39	534.0	17.46	1040
419.9	10.22	421.4	10.26	422.9	10.30	424.4	10.34	425.8	10.38	427.2	10.42	1050
349.9	6.77	351.1	6.80	352.4	6.83	353.6	6.86	354.8	6.88	356.0	6.91	1060
299.9	4.81	301.0	4.83	302.1	4.85	303.1	4.87	304.2	4.88	305.2	4.90	1070
262.4	3.59	263.3	3.61	264.3	3.62	265.2	3.64	266.1	3.65	267.0	3.67	1080
233.3	2.78	234.1	2.79	235.0	2.80	235.8	2.81	236.6	2.82	237.4	2.83	1090
209.9	2.21	210.7	2.22	211.4	2.23	212.2	2.24	212.9	2.25	213.6	2.26	1100
190.9	1.80	191.5	1.81	192.2	1.82	192.9	1.82	193.6	1.82	194.2	1.83	1110
174.9	1.49	175.6	1.49	176.2	1.50	176.8	1.51	177.4	1.52	178.0	1.52	1120
161.5	1.25	162.1	1.25	162.7	1.26	163.2	1.26	163.8	1.27	164.3	1.28	1130
150.0	1.07	150.5	1.07	151.0	1.08	151.6	1.08	152.1	1.09	152.6	1.09	1140
139.9	.93	140.4	.93	140.9	.94	141.4	.94	141.9	.94	142.4	.94	1150
131.2	.80	131.7	.80	132.1	.81	132.6	.81	133.1	.81	133.5	.82	1160
123.5	.70	123.9	.70	124.4	.71	124.8	.71	125.3	.71	125.7	.72	1170
116.6	.63	117.0	.63	117.5	.63	117.9	.63	118.3	.63	118.7	.64	1180
110.5	.55	110.9	.55	111.3	.56	111.7	.56	112.0	.57	112.4	.57	1190
105.0	.49	105.3	.49	105.7	.50	106.1	.50	106.5	.50	106.8	.51	1200
99.98	.44	100.3	.44	100.7	.45	101.0	.45	101.4	.45	101.7	.46	1210
95.43	.401	95.77	.403	96.11	.41	96.45	.41	96.78	.41	97.10	.41	1220
91.28	.365	91.61	.367	91.94	.369	92.26	.371	92.57	.373	92.88	.375	1230
87.47	.333	87.79	.335	88.10	.337	88.41	.339	88.71	.341	89.00	.343	1240
83.98	.304	84.28	.306	84.58	.308	84.88	.309	85.17	.310	85.45	.312	1250
80.74	.281	81.02	.282	81.32	.284	81.60	.285	81.88	.286	82.15	.287	1260
77.75	.257	78.02	.259	78.31	.260	78.58	.261	78.85	.262	79.10	.263	1270
74.96	.237	75.23	.238	75.50	.239	75.77	.241	76.03	.242	76.28	.243	1280
72.38	.218	72.64	.219	72.91	.221	73.17	.222	73.42	.223	73.66	.224	1290
69.98	.203	70.23	.204	70.48	.206	70.73	.207	70.97	.208	71.20	.209	1300
67.71	.190	67.95	.191	68.20	.192	68.44	.193	68.67	.194	68.90	.195	1310
65.61	.175	65.84	.176	66.08	.177	66.31	.178	66.54	.179	66.76	.180	1320
63.62	.165	63.84	.166	64.08	.167	64.30	.168	64.52	.169	64.73	.170	1330
61.74	.153	61.96	.154	62.19	.155	62.41	.156	62.62	.157	62.82	.158	1340
59.98	.143	60.19	.144	60.41	.145	60.62	.146	60.83	.147	61.02	.148	1350
58.31	.134	58.52	.135	58.73	.136	58.94	.137	59.14	.138	59.33	.139	1360
56.73	.128	56.93	.128	57.14	.129	57.34	.129	57.54	.130	57.73	.130	1370
55.24	.120	55.44	.120	55.64	.121	55.83	.122	56.02	.123	56.21	.123	1380
53.83	.113	54.02	.113	54.21	.114	54.40	.114	54.59	.115	54.77	.115	1390
52.48	.107	52.66	.107	52.86	.108	53.04	.108	53.22	.109	53.40	.109	1400
51.20	.100	51.38	.100	51.57	.101	51.75	.101	51.93	.102	52.10	.102	1410
49.99	.095	50.16	.095	50.34	.096	50.52	.096	50.70	.097	50.87	.097	1420
48.82	.091	48.99	.091	49.17	.092	49.34	.092	49.51	.093	49.67	.093	1430
47.70	.086	47.87	.086	48.05	.087	48.22	.087	48.38	.088	48.54	.088	1440
46.65	.081	46.81	.081	46.98	.082	47.15	.082	47.31	.083	47.47	.083	1450
45.63	.078	45.79	.078	45.96	.079	46.12	.079	46.28	.080	46.43	.080	1460
44.66	.073	44.82	.073	44.98	.074	45.14	.074	45.30	.075	45.45	.075	1470
43.74	.070	43.89	.070	44.06	.070	44.21	.071	44.36	.071	44.50	.071	1480
42.84	.067	42.99	.067	43.15	.067	43.30	.069	43.45	.069	43.59	.069	1490
41.99	.063	42.14	.063	42.29	.064	42.43	.064	42.58	.065	42.72	.066	1500

CHAPTER VIII

DISCREPANCIES BETWEEN ACTUAL RECOVERY AND THEORETICAL EXTRACTION

SECTION III

SAMPLING OF MILL HEADS AND RESIDUES

Sampling of Heads.—This is the second of the more fertile causes of failure to check on bullion recovery. When milling in water the most reliable sample of ore heads is probably that taken from the pulp after it leaves the mortar boxes. This may be done by catching a portion in a cup or other receptacle at regular intervals, and if care is taken that the whole depth and width of the stream issuing from the screens is included, that the cup is not allowed to overflow, and that the intervals are regularly timed and sufficiently frequent, the resulting sample is usually reliable.

A better way is to place an automatic sampler in the stream of pulp, thus eliminating the personal factor, but great care is necessary to see that the sampler used is correctly designed to cut a perfect section of the stream both in breadth and depth, because a stream of pulp flowing in a launder is not homogenous, and moreover the effect of a slight error in the product of a mechanical sampler is cumulative, that is, it is invariably in the same direction and is not subject to the correction which may automatically occur in other circumstances where errors in excess and defect may neutralize one another over a period and result in a representative average reading.

An automatic sampler that satisfactorily withstood very severe and searching tests on the Rand is that invented by James Higham and described by him in the Journal of the Chem., Metall. & Min. Soc. of S. A. for July, 1903.

It will be seen from the illustration that it is essentially a short length of curved and slotted pipe that is made to pass across the stream of pulp by means of the screw mechanism which is power-driven, and the direction of travel automatically reversible. A break is made in the pulp launder and the stream in dropping from the upper to the lower level encounters the slotted pipe which cuts out a section of it and delivers it to a suitable receptacle placed underneath. The normal size of sample taken is about

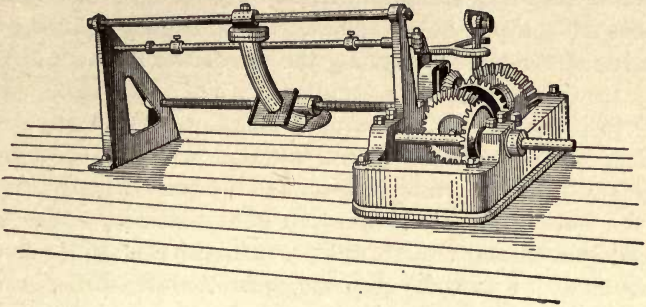

Fig. 25.—Higham Pulp Sampler.

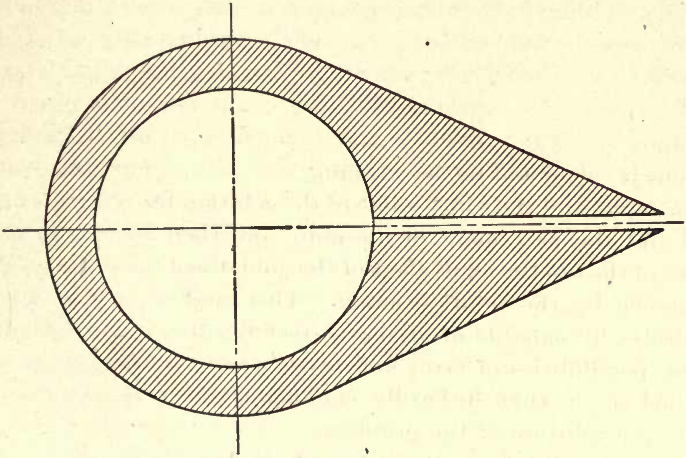

Fig. 26.—Section of Cutter Pipe, Higham Pulp Sampler.

2000 lb. per day from a 100 stamp mill, or 1 ton from a daily tonnage of presumably about 1000 tons of ore.

The only objection raised to this sampler has been the unwieldy size of the resulting sample but this may be obviated by inserting a second sampler below the first to reduce the quantity taken to convenient dimensions, or by the use of the Higham Dividing Box, a trough fitted with a number of knife edged slots, at right angles to it, acting on the same principle as the Jones dry ore sampler. The trough is placed parallel with the direction of movement of the sampling pipe and as the outflowing pulp passes over the slots a definite proportion of the stream enters and is received in a second trough underneath, the remainder or reject flowing away to join the main stream. Any machine, however, will answer the purpose that is so designed as to cut a perfect section of the stream both horizontally and vertically without appreciable acceleration during any part of the stroke, and with a minimum of splash and diversion of the line of flow.

When milling in cyanide solution it is difficult to obtain a head assay from the pulp owing to part of the gold passing rapidly into solution. Methods have been suggested on the Rand and are said to have been carried out with a certain measure of success, whereby the true value of a pulp sample taken under these conditions is calculated by ascertaining the ratio of solution to ore in the sample, and the assay value of the solution before entering the mill and also as present in the pulp, and then adding on to the assay of the ore the equivalent of the gold dissolved during milling as shown by the solution assay. This method, while it seems theoretically capable of giving correct results, yet is open to so many possibilities of error and needs so much care and time expended on it that it hardly commends itself as a practicable everyday solution of the problem.

The only alternative is to sample the ore before it comes in contact with the solution, and if properly done, this method may be made to yield reliable working results. It is possible to set about it in two ways, first, to install a complete sampling plant in the mill, cutting out a section of the stream of broken ore going to the bin and subdividing it by machines interspersed with crushing devices for reducing the size of the particles after each quartering until a small but representative assay sample remains. This, while satisfactory, is costly, both to install and to run, and some important mills which adopted the system have since abandoned it as not yielding any adequate compensation for the expense involved.

The second method is to cut out small individual samples of the ore periodically at some point after it leaves the breakers. If means can be found to do this regularly and fairly, a good representative sample may be obtained. The most reliable system is of course an automatic sampler if it is practicable to insert one somewhere between the rock breaker and the intake of the mortar box, but hand samples faithfully and regularly taken are on the average dependable. An important point to be observed is that as long as the ore is coarsely broken, the aggregate sample for the day or shift must be a large one or it will be of no use. When cutting out, at regular intervals of half an hour, samples of a two-inch rockbreaker product in a mill crushing 100 tons a day, an aggregate sample of 1000 lb. for the twenty-four hours is not any too much, and 2000 lb. would be safer. This should then be all crushed to about ¼ inch size before being quartered down to a smaller bulk. The mixing and quartering may be done by hand shovelling or one of the various mechanical devices may be used. When the bulk is reduced to about 100 lb. this should then be crushed to pass ⅛ inch screen and quartering continued till the sample weighs about 20 lb. when it should be ground to pass 10 mesh; in this condition it may be reduced in bulk to one or two pounds and then sent to the assay office where the process is continued until the necessary assay sample of 80 or 100 mesh is obtained. This may seem an unnecessarily expensive and laborious process, but it cannot be shortened with safety if the resulting assays are expected to furnish a basis on which to figure bullion returns.

An unusually significant example of this contention, drawn from the writer's own experience, is here given by way of illustration. The ore in question was a gold-silver ore in which the gold, more especially, occurred in a notoriously "spotted" manner. The assays given are combined gold and silver results. The sample for the twenty-four hours was taken by hand from the battery feeder, a portion being caught every hour in a specially shaped scoop as it fell off the edge of the disc. The total weight was about 200 lb. and maximum size of particles from one to one and a half inches. In the course of an investigation into a continued and serious shortage of bullion as compared with the amount expected from a consideration of head and tail assays, a special series of tests was made on the method of quartering down the daily ore head sample.

Each day's sample for a number of days in succession was either mixed and quartered direct or the whole was crushed to pass ¼-in. mesh before mixing and quartering. Duplicate samples were made at the first coning-up and quartered down and assayed separately, forming checks on the method of quartering. The results are summarized below.

TABLE 10

	"*A*" Quartered direct		"*B*" Crushed to ¼ in. before quartering	
	First half assay, total High	Second half assay, total Low	First half assay, total High	Second half assay, total Low
Nov. 12	$43.95	$37.77		
Nov. 14	$52.14	$31.04		
Nov. 16	.....	.....	$37.51	$35.53
Nov. 17	$50.91	$32.62		
Nov. 18	$30.14	$20.74		
Nov. 20	.....	.....	$25.05	$24.11
Nov. 21	.....	.....	$31.78	$30.16
Nov. 22	.....	.....	$24.79	$23.17
Nov. 23	.....	.....	$45.91	$40.48
Nov. 24	.....	.....	$37.23	$35.82
Nov. 25	.....	.....	$52.72	$51.77
Averages	$44.28	$30.54	$36.42	$34.43

The above indicates a possible error of about $14 a ton when the original sample is not crushed fine before quartering, while in the other case the difference shown is only $2.00.

It may be asked, in regard to "*A*," why the error shown should be supposed to account for a uniformly high result in the long run, instead of neutralizing itself over a long period of time, and it should be noted in reply that the source of the error is not likely to operate on the side of defect, or only to a slight extent, for this reason: since it is known that the ore is spotted in value and that a small stone may assay several hundreds of dollars right alongside of quite low-grade rock, let us suppose that a certain battery sample is fairly uniform but contains one stone of "high-grade" value; if, in coning and quartering the sample, that stone happens to be in the rejected quarter, the sample taken for assay will be representative, whereas if it happens to fall in the quarter saved the assay sample will be thrown up to the high limit of error shown. Thus, it is not so much a question between excess and defect (neutralizing one another) as it is between excess and mean, yielding an average in excess, over a period.

It is obvious that the greater the number of pieces into which that stone of "high-grade" is broken before the sample is quartered, the better the chance of its being distributed equally between the portion rejected and the portion saved, but on the other hand assuming the sample to have been broken sufficiently fine for safety at this stage, in continuing to quarter down this same sample the original danger appears again and increases in proportion as the sample gets smaller so that it will soon become necessary to subdivide the particles still further in order to obtain an even distribution of the high-grade ore throughout the mass.

A. Harvey (Mining and Scientific Press, Jan. 30, 1904) gives the following table as a guide to the margin of safety in size of ore pieces.

TABLE 11

Diameter of largest pieces in sample, in.	Minimum weight of sample, lb.	Diameter of largest pieces in sample, in.	Minimum weight of sample, lb.
5½	79,300	⅔	256
4	69,109	⅓	32
3½	44,958	⅙	4
2½	16,384	1/12	½
1¼	2,048	1/25	1/16

These methods of sampling the heads are necessarily expensive and as it is not always easy to point to any material gain in monthly profits resulting from their adoption, and as moreover, their use does not invariably result in the establishment of an accountant's balance between the bullion debited and credited to the mill, many companies have asked themselves whether they were not wasting time and money in trying to ascertain the true value of their ore heads and have finally decided to be content with a nominal value for the ore, obtained by adding the amount of bullion actually recovered to the aggregate amount of gold going to the dump in residues and dividing the sum of these by the tonnage for the month. The degree of satisfaction to be derived from this solution of the difficulty is a matter of individual opinion, and while admitting that it is not possible to ascertain head and tail assay values and tonnage with sufficient accuracy to furnish a check on small losses through carelessness or petty pilfering, yet the present writer confesses to a preference for making the best attempt possible to ascertain the real figures in each case, so as to act as a check against serious losses and afford an indication of any radical defects in the operation of the plant.

Sampling of Residues.—This is comparatively easy as the pulp at this stage is finely comminuted and usually homogeneous, so that only ordinary care and common sense should be needed to obtain a reliable sample.

In the case of sand the charge may be bored from top to bottom in a number of places after having leached dry preparatory to discharging, or if the residue be discharged into trucks each load may be bored as it passes to the dump. In boring a tank charge care should be taken when using the ordinary split tube sampler not to take too much at a time, since if the rod be worked down from top to bottom without being withdrawn and emptied there is danger that the lower strata may escape sampling. It is better to drive down about 12 inches, then withdraw and empty the pipe, and then after carefully inserting in the hole without breaking down the sides, sample the next 12 inches and so on to the bottom. If on account of the method of filling or any other reason it seems probable that the charge is not homogeneous it is better to divide the area of the tank into concentric rings containing an equal tonnage and bore in each ring an equal number of holes so that the quantity of sand taken from each ring will be representative of the tonnage contained therein.

For slime residues the use of the automatic sampler is probably the most satisfactory method where the pulp is discharged in a homogeneous fluid condition.

The sample of filter cake is not so easy to obtain. One method is to cut a section of the cake from a leaf of each charge or in the case of the rotary filters to take a grab sample from the scraper periodically. It sometimes happens, however, that a sample dipped out of the discharge launder as the watery cake is flowing to the dam does not give an assay check on the sample cut from the cake before discharging, the latter being usually lower. The reason for this is probably that, assuming the sample taken from the launder to be a correct average, there may be with almost any type of filter a loss of a small amount of valuable solution which passes out with the cake and in that case the sample taken from the leaf does not represent the whole of the lost metal going to the dams.

Sampling Slime Pulp at Different Stages of the Treatment.—If this is to be of any value at all it is absolutely necessary that the action of the cyanide on the ore should be stopped immediately the sample is taken. This may be ensured in two ways: (1) A sample taken every hour, for instance, at a certain point in the treatment may be at once placed on a laboratory vacuum filter and filtered and water washed, the several cakes composing the twenty-four hour sample being dried, mixed, and quartered down for assay as one sample; (2) or a method devised by H. A. White[1] may be used. The sample as soon as taken is placed in an enamelled bucket and enough saturated solution of potassium permanganate added to maintain a permanent pink coloration in the solution during the whole period of sampling. This destroys the free cyanide and stops further action on the ore. At the end of the period of sampling, usually twenty-four hours, the contents of the bucket are well mixed, washed by decantation or filtration and sent for assay. The reaction between the permanganate and the cyanide takes place without precipitation of the gold but whether this is true of the silver content of the solution the writer has never determined as he has been accustomed to use the immediate filtration and washing method.

Discrepancy in Assay of Slime Residues.—In the case of gold ores there is a possible source of error in the assays which will tend to show a higher extraction than has actually taken place, and therefore an apparent shortage of gold in the clean-up. W. A. Caldecott[2] has shown that when a sample of slime containing potassium auro-cyanide solution as moisture is dried for assay it is almost impossible in the subsequent fusion to collect in the lead button all the gold present in this form probably on account of its being deposited in such minute particles as to approach the molecular state, there being therefore not enough globules of reduced lead to come in contact with every molecule of gold. It was found that the finer the litharge in the flux was ground the higher was the resulting assay in gold, but that under the most favorable conditions the gold content found was lower than that obtained by washing the pulp thoroughly and assaying the ore and washings separately. Whether the above reason be the correct one or whether the loss of gold be due, as is maintained by some assayers, to a deposition of the gold bearing solution on the walls

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[1] *Journal Chem., Met. and Min. Soc. of S. A.*, Sept., 1911.

[2] *Proceedings* of Chemical, Metall. & Mining Society of S. A., Vol. II, page 372.

of the vessel used for drying the sample, the fact of the discrepancy has been well attested and it was probably in consequence of this investigation that the Rand practice was developed of "fixing" or precipitating the gold in the slime residues by the copper sulphate method used for solution assays before drying the pulp (see page 52).

A. Whitby[1] states that his practice is to add to the slime residue first a little cyanide solution and then small quantities of solutions of copper sulphate, sodium sulphite and sulphuric acid. After mixing thoroughly the pulp is turned out into an enamelled dish and dried, care being taken to avoid spitting. The sample is then passed through a 100-mesh screen and assayed. It is the custom on the Rand to take exceptionally large charges for the assay and for low-grade pulps it is a necessary precaution especially where a large mill tonnage will multiply an error of a few cents in the assay to a serious sum total of gold in the month's tonnage. From five to ten assay tons of pulp are usually taken for an assay of residue, distributed over two or more crucibles. The flux used by Mr. Whitby for the Rand residues has the following proportions:

Ore	5 A.T.
Litharge	4 A.T.
Sodium carbonate	5 A.T.
Borax	1 A.T.

Reducer, sufficient to give a button of about 45 grams.

That the error due to neglect of these precautions may be serious is shown by an experience of the writer with a high-grade gold-silver ore in Mexico. It had been the custom to estimate the assay of ore passing to the cyanide plant by assaying the pulp as thickened in the cyanide solution, prior to agitation, the argument being that as no part of the mill solution was withdrawn for precipitation the amount of gold and silver circulating in that medium might be taken as fairly constant and that therefore an assay of the thickened pulp with its normal burden of dissolved

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[1] *Journal Chem., Metall. and Min. Soc. of S. A.*, March, 1906.

gold and silver would truly represent the value of the pulp leaving the mill, and this value plus that contained in the concentrate would represent the assay of the mill head ore. In practice, however, there was a persistent plus recovery of gold over that shown by assays.

On precipitating the pulp before drying for assay it was found that the gold value as shown by the usual method was much too low though the silver did not appear to be affected. A comparison run daily for 26 days with and without preliminary "fixation" of the gold showed an average gain in the assay of $1.40 per ton due to "fixation," the figures being $13.90 by the usual method and $15.31 by the preliminary "fixation." Of course this is an extreme instance, but in the case of residues a persistent error in the assay of only 10 c. a ton on the low side would make a serious difference between actual and theoretical recovery.

CHAPTER IX

ORES PRESENTING SPECIAL DIFFICULTIES

Telluride and Mispickel Gold Ores.—Gold associated in this way is difficultly soluble in ordinary cyanide solutions and special treatment is necessary for its extraction. There are two methods in use in such cases.

1. *Roasting.*—The ore is ground dry to about 30 mesh and roasted; it is then ground to a slime in water or cyanide solution, usually in grinding pans, utilizing the opportunity simultaneously to amalgamate any gold that may be amenable, and finally treated by agitation in cyanide solution in the ordinary way.

2. *Treating Raw with Bromocyanide.*—This process was used at the Deloro mine, Ontario, Canada, on a mispickel ore, and was developed in Australia by Dr. Diehl for the treatment of the sulpho-telluride ores of the Kalgurli district. The process essentially consists of grinding exceedingly fine, and then agitating with cyanide solution to which bromide of cyanogen is added at intervals. In some mills preliminary amalgamation and concentration are in use, the concentrate being roasted and then cyanided. During bromocyanide treatment the protective alkalinity is kept at the lowest possible point owing to the instability of the reagent, lime sufficient for settlement being added after the treatment is finished.

Julian and Smart[1] state that at Kalgoorli when dealing with slimes assaying from 1 to 3 oz. of gold a wash of cyanide solution containing about 4.5 lb. of KCN to the ton of ore is first given, and after an hour or two's agitation cyanogen bromide is added at the rate of about 1 lb. BrCN per ton of ore, and agitation continued for twenty-four hours. The quantities of free cyanide and bromocyanide are varied according to the assay value of the

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[1] Cyaniding Gold and Silver Ores, page 260 (Second Edition).

ore, any additional bromocyanide needed being added at intervals of several hours. Shortly before agitation is finished sufficient lime is added for the settlement of the slime.

The usual method of making the reagent for laboratory use is to add a strong solution of cyanide to bromine (and not vice versa), until the brown color is just discharged.

$$KCN + Br_2 = KBr + BrCN$$

The quantity of BrCN may be determined in a working cyanide solution by acidifying with hydrochloric acid, adding excess of potassium iodide, and titrating the liberated iodine with decinormal sodium thiosulphate. Clennell gives the following reactions:[1]

$$BrCN + HCl = HCN + ClBr$$
$$ClBr + 2KI = KCl + KBr + I_2$$
$$1 \text{ cc } \frac{N}{10} \text{ thiosulphate} = 0.00529 \text{ grm. } BrCN.$$

The bromocyanogen is added to the cyanide solution to be used for extraction purposes in the proportion of about 1 of BrCN by weight to 4 of KCN.

Since BrCN is rapidly decomposed by alkali it is important that free alkali should be almost entirely absent during the treatment, the lime necessary for settlement being added at its conclusion. The reaction is usually illustrated thus:

$$BrCN + 2KOH = KBr + KCNO + H_2O$$

The following is an extract from a paper by E. W. Nardin, of the Hannan's Star Mill, Kalgoorli, published in the Mining and Scientific Press of October 24th, 1908.

"The bromo-cyanide solution is made according to the following equation:

(1) $2KBr + KBrO_3 + 3KCN + 3H_2SO_4 = 3BrCN + 3K_2SO_4 + 3H_2O\cdot$

Its action in the treatment vat is supposed to be as follows:

(2) $BrCN + 3KCN + 2Au = 2KAu\,(CN)_2 + KBr.$

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[1] Chemistry of Cyanide Solutions, page 100 (Second Edition).

"The first two quantities in equation 1 are contained in the mixed salts supplied by the London-Hamburg Co., having about 40 to 44% Br as KBr, and 20 to 22% Br as $KBrO_3$; the proportion of Br as bromide being about twice that of Br as bromate. A 30-lb. charge is usually made up, and for this the following weights are taken:

	Lb.
H_2SO_4	50.0
KCN	20.0
Mixed salts	36.8

The KCN is 93%, and the H_2SO_4 63% (chamber acid) strength. The solution is made in a closed wooden vessel, stirred by rotating arms, holding about 200 gal. In making up a charge, a portion of the water and all the H_2SO_4 are first mixed, and allowed to cool to normal temperature. The KCN, which is dissolved in a separate vessel in sufficient water to fill the mixing vessel, is then run in, and at the same time the proper weight of 'mixed salts' is gradually added. The whole is then agitated for 6 hours before being used, and in a closed vessel it will retain its strength for some days. The cost of a 30-lb. charge of BrCN is about $21.88, made up as follows: 50 lb. H_2SO_4 at 4 c., 20 lb. KCN at 17 c., 36.8 lb. salts at 44 c. From each charge mixed, a dip-sample is taken and tested with a standard $Na_2S_2O_3$ solution, using potassium iodide as an indicator.

$$\underset{(106)}{BrCN} + 2KI = KBr + KCN + \underset{(254)}{I_2} \tag{3}$$

$$\underset{(316)}{2Na_2S_2O_3} + \underset{(254)}{2I} = 2NaI + Na_2S_4O_6 \tag{4}$$

Thus BrCN corresponds to $2Na_2S_2O_3$.

"The standard solution is made so that 1 cc corresponds to 0.02 gm. BrCN, and for this about 93.6 gm. of ordinary photographic crystals, $\underset{(248)}{Na_2S_2O_3 \cdot 5H_2O}$, are dissolved in one litre of water."

"A solution of copper sulphate is used for standardizing the above.

$$\underset{(126.8)}{2\,CuSO_4} + 4KI = Cu_2I_2 + 2K_2SO_4 + \underset{(254)}{I_2} \tag{5}$$

126.8 Cu liberates 254 I, so that it corresponds to 106 BrCN and 496 $Na_2S_2O_3 \cdot 5H_2O$. The solution is made by dissolving one gram of pure copper foil in acid, converting to sulphate, and dissolving in 100 cc water. Hence 10 cc contains 0.1 gm. Cu, equivalent to 0.0836 BrCN; then 1 cc hypo solution corresponds to 0.02 gm. BrCN, and 4.18 hypo solution to 0.0836 gm. BrCN."

"In testing BrCN solutions, 5 cc are usually taken, Na_2CO_3 solution is added till alkaline, and then acetic acid till acid. A few crystals of KI, and some starch solution, are then added, and the whole titrated with the standard $Na_2S_2O_3$ solution."

"EXAMPLE: If 5 cc BrCN solution took 3.2 cc hypo, then

$$\frac{3.2 \times 0.0836}{4.18} \times 20 = 1.28\%$$

"This method of testing BrCN solutions is different to that used at other mines, where it is customary to titrate direct with $Na_2S_2O_3$ without first neutralizing the H_2SO_4.

The following is a series of tests made on a number of 30-lb. BrCN solutions by direct method, *A*, and, after neutralizing, *B*.

TABLE 12

Charge	Pounds	*A*, Per cent.	*B*, Per cent.
668	30	0.36	1.20
670	30	0.44	1.20
671	30	0.60	1.28
672	30	1.20	1.24
673	30	0.36	1.36
674	30	0.52	1.00

"These show that by direct titration the test is usually low, and also that it gives irregular results, depending on the amount of free H_2SO_4 present."

"There seems to have been some doubt as to whether a BrCN solution increases in strength after, say, one hour's agitation, but tests which were made show that it does increase up to about 8 hours, from 0.56 to 1% BrCN."

"It was generally known that if the solution became too alkaline, either through a change in the ores, or the addition of too much lime either to the ore before crushing or after the bromo-cyanide treatment. the extraction by BrCN fell considerably, and for this reason an occasional test of the plant-solution was made for alkalinity, but not until several high tailing-discharges had been observed."

"Under the old system the KCN did not have sufficient time by itself; the gold in the ore and the KCN residue of each vat were not known, and the amount of BrCN that should be added was more or less a guess. When it is remembered that every 5 lb. of BrCN added to a 50-ton vat represents a cost of 8¢ per ton of ore, that the action of the BrCN can be made just as effective after long KCN treatment, and that excess of BrCN gives no advantage, it will be seen how important it is that the value of the KCN tailing should be known after sufficient agitation (say 12 hr.) and the condition of the vat tested as to alkalinity before adding the BrCN. In any case, the action of the BrCN is of short duration, not exceeding 4 hours, so that if 20 hr. total agitation can be allowed, it is better to give 16 hours with KCN, and then add the BrCN. It is preferable, however, to keep the vat under KCN treatment until the KCN residue is known, then correct the alkalinity, and add the BrCN. This could easily be done with extra vat-capacity."

He concludes:

"The results of these experiments have suggested certain improvements in bromo-cyaniding, some of which have been adopted, as follows:

1. The daily ore sample should be taken in the morning, and assayed as soon as possible, so as to know the value of the ore passing to the vats in the previous 24 hours.
2. The pulp should have a long KCN treatment.
3. A vat should be kept under KCN treatment till the value of the KCN residue is known.
4. The alkalinity of the vat should then be determined and corrected to 0.01% by H_2SO_4 before adding BrCN.
5. The quantity of BrCN added should then be determined from the value of the KCN residue, the tonnage of the vat, and so forth.
6. The lime added to the ore during crushing should be varied according to the alkalinity-test after KCN treatment, so that the plant-solution tests about 0.02%.
7. Lime water should be made and added to the vats or to the solution from the presses, instead of adding lime to the vats.
8. Metallic iron should be kept out of the pulp as far as possible, as it is both a cyanicide and bromo-cyanicide."

Influence of Arsenic in the Cyanidation of Gold Ores.—W. B. Blyth[1] is of the opinion that the presence of arsenic in an ore does not necessarily cause that ore to be refractory to cyanide treatment. He thinks the idea arose from the fact that arsenic is so often accompanied by antimony, pyrrhotite or graphite, stating that "the two former minerals cause premature precipitation of the gold almost as fast as it goes into solution, if they are present in sufficient quantity."

The trouble in cyaniding semi-oxidized ores of arsenic he attributes rather to the presence of acid ferrous salts, and if he is right a preliminary water wash or possibly a weak acid wash ought to meet the case. (It should be noted here that the paper referred to does not deal with the cyanidation of silver ores containing arsenic.)

In detailing a number of experiments made on a certain antimonial gold ore Mr. Blyth states that by milling in water and running the ore into dams where it was settled, dried, and allowed to oxidize by cultivation of the surface, the pulp changed in color from a dark grey to a light yellow, and that whereas by immediate cyanidation the ore had only yielded 50% of its gold, after the oxidation treatment 93% extraction could be obtained. A similar result was later accomplished as far as the sand was concerned by milling the pulp without lime and then separating the sand from the slime. He says,

> "Owing to the solubility of antimony in alkaline cyanide it is not possible to add lime to the mills and achieve good results. Consequently the abraded iron which mainly goes with the slime is reduced to the ferrous state and independently of the antimony effectually upsets the extraction from this product when treated direct without lime. In connection with the sand, however, it is found that the residue rises proportionately to the lime used and the best results are obtained when no lime at all is used. About one-half lb. of lead nitrate is used per ton of ore and the sand is slimed in an acid solution. The consumption of cyanide is naturally heavy, but the good extraction counterbalances losses in this direction."

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[1] *Mining Magazine* (London), April, 1919.

In regard to the use of cyanide solutions without protective alkali the writer found with one Mexican gold ore that while in a solution with a normal protective alkalinity only about 68% of the gold could be extracted, in a solution showing no protective alkali the extraction was raised to 88%. The ore contained several per cent. of arsenic but no antimony was reported.

Manganese Silver Ore.—In Mexico and elsewhere there are large bodies of valuable silver ore containing manganese which present great, and hitherto insuperable, obstacles to a satisfactory extraction by cyanide. The proportion of the silver that is soluble in cyanide varies greatly and may be as low as 5% only. The mere presence of manganese in the ore is not invariably accompanied by refractoriness of the silver because the writer has known of an instance where ores of apparently similar composition and character and coming from adjacent mines behaved quite differently, one yielding only 50% of its silver and the other 90% under the same conditions. As a general rule, however, when oxides of manganese are present in a silver ore trouble may confidently be expected.

The manganese usually occurs as the dioxide but in what way, if at all, it is associated with the silver has never been determined. There are three possible theories on the subject. The first is that the refractory silver occurs as a silver-manganese mineral. Although no such mineral is described in the text books, there are certain facts which would seem to point to its existence. (*a*) The silver is insoluble in nitric acid, in connection with which it is to be noted that MnO_2 is also more or less insoluble in nitric acid. (*b*) If the oxides of manganese be removed by some solvent, such as hydrochloric or sulphurous acid, the silver is at once rendered soluble in cyanide solution.

On the other hand there are circumstances which seem to point rather to the idea that the silver may be free from the manganese but occurring in some unknown insoluble form, the result of oxidation. In favor of this theory may be mentioned the fact that (1) in the case of manganese ores it commonly happens that the ore in the upper levels is more refractory than that in the lower, though containing the same percentage of manganese. (2) An oxidizing roast applied to any silver ore usually renders the bulk of the silver insoluble in cyanide. (3) A preliminary reducing treatment given to a refractory manganese ore, as for instance agitation with an alkaline hydrosulphide or a roast with sulphur or crude oil in a reducing atmosphere, will increase the percentage of the refractory silver subsequently soluble in cyanide. (4) If a sample which has been through such a reducing treatment be subsequently subjected to an oxidizing effect, as for instance blowing air through the wet pulp, there is a tendency to throw the silver back into the insoluble form. It may be admitted as against this theory, that a simple silver oxide or salt insoluble both in nitric acid and potassium cyanide has hitherto been unknown.

The third hypothesis is that the silver is combined with some mineral other than manganese dioxide, but that the mineral combination never occurs except in company with the manganese though the manganese may occasionally occur unaccompanied by that particular silver mineral. None of these theories, however, seems to conform to all the known facts.

Following up the indications of the benefit of preliminary reducing treatment for these ores, a process has been patented in which the ground ore is roasted in an atmosphere of producer gas previous to cyanidation by which means the inventor claims to be able to obtain extractions up to 90% and over. It would seem, however, that as in the case of the chloridizing roast the ore would need to have an unusually high silver content to make this process possible.

Where the grade of the ore is sufficiently high to make the process commercially profitable a good extraction can usually be obtained by a chloridizing roast followed by cyanidation. The feasibility of this method will depend largely on the cost of fuel and salt at the mill as well as the assay value of the ore. There will also be a loss entailed by volatilization of a part of the gold if gold be present and also to a less extent of the silver. This could no doubt be corrected to a large extent by installing a Cottrell precipitator for treating the fume from the furnace though at the expense of increasing both installation and running costs. It would also be necessary to mill the ore dry, a method which is usually to be avoided whenever possible. Flotation has been tried for dealing with such ores with the result of producing a small quantity of high-grade concentrate, but a very low percentage recovery from the ore; moreover, generally speaking, the silver recovered by flotation is just that part of the silver content that is amenable to cyanide while the refractory silver remains in the tailing.

Probably the most hopeful chance of finding a solution of this problem lies in the process first suggested by the writer early in 1909 and mentioned in the Mining and Scientific Press of December 4th in that year. It consists in treating the ground ore in a water pulp with SO_2 gas, or its solution in water by which the manganese is dissolved and washed out leaving the silver free to be acted on by cyanide. Extractions up to 95% of the silver contents have been made on refractory ores by this process and while the extra expense is considerable it will in most cases be a good deal lower than that involved in roasting processes, and might under favorable conditions be found of payable application to ore of average grade. At the present time (1919) experiments are being conducted with very promising results along these lines in Pachuca using the concentrates from the sulphide ore to furnish sulphur gas for the treatment of the refractory ore. Walter Neal[1] appears to prefer the use of sulphuric acid and ferrous sulphate as a substitute for the sulphurous acid stating that he found the extraction due to these reagents somewhat superior to those from the SO_2 but the method would seem a good deal more expensive unless the solutions could be regenerated electrolytically. The writer has made some tests along these lines, but without very encouraging results.

For those who wish to experiment with this process it may be said that the most satisfactory method for laboratory use consists in the addition of varying amounts of a solution of SO_2

[1] Private communication.

gas of known strength which may be prepared by boiling strips of metallic copper in concentrated sulphuric acid, passing the gas through a washing bottle and thence into distilled water until saturated. The method of determining the SO_2 content of a solution is as follows:

Take from 50 to 200 cc of $\frac{N}{10}$ iodine solution according to the strength of the SO_2 solution and add to it from a pipette 5 to 10 cc of the SO_2 solution. Should the brown color of the iodine be dispelled or nearly so, it will be necessary to make a new test using more iodine since the iodine must be in excess. Then titrate the excess iodine with $\frac{N}{10}$ $Na_2S_2O_3$. The original amount of $\frac{N}{10}$ iodine less the quantity shown by titration gives the iodine consumed by the SO_2.

$$1 \text{ cc } \frac{N}{10} \text{ I consumed} = 0.0032 \text{ gr. } SO_2$$

Arsenic and Antimony Compounds of Silver.—Ores containing silver in such combinations offer considerable difficulties in cyanidation and it has usually been considered that the removal of these compounds by concentration was the only way to deal with them. G. H. Clevenger, however, when working on the ores of the Nipissing Mining Company of Ontario, Canada, found that by grinding exceedingly fine and treating with strong solution for four or five days a high extraction could be obtained. The silver occurred partly as sulphide, sulphantimonide and sulpharsenide and partly as metallic silver and dyscrasite, an alloy of silver and antimony in varying proportions.

To accelerate the dissolution of the antimony and arsenic compounds J. J. Denny introduced a process that had for its object their decomposition in a preliminary treatment. This process consists in grinding the whole of the ore to a slime and subjecting it to the action of metallic aluminium in a caustic soda solution, after which it is filtered and subjected to cyanidation.

He says[1]

"By the preliminary treatment the silver, and in part at least, the antimony and arsenic, are reduced to the metallic state, and are so found. The reduction is accomplished by the nascent hydrogen resulting from the action of caustic soda on the aluminium according to the following equation:

(1) $$2Al + 2NaOH + 2H_2O = Na_2Al_2O_4 + 6H."$$

"The probable reactions involved in complete reduction are indicated by the following equations:

(2) Argentite.

$$6H + 3Ag_2S + 6NaOH = 3Na_2S + 6H_2O + 6Ag$$

(3) Pyrargyrite.

$$6H + Ag_3SbS_3 + 6NaOH = 3Na_2S + 6H_2O + 3Ag + Sb$$

(4) Proustite.

$$6H + Ag_3AsS_3 + 6NaOH = 3Na_2S + 6H_2O + 3Ag + As."$$

"The reactions being reversible, probably the arsenic and antimony are not completely reduced to the metallic state in practice, and the investigation of the subject is rendered difficult by reason of secondary reactions by which the arsenic and antimony are possibly redissolved to form arsenates and antimonates by the excess caustic of the reducing solution, and the protective alkali of the cyaniding solution. The working solution shows the presence of these compounds, but in practice they are found to have no detrimental effect either in the reducing or the cyaniding treatments. The solution assays, antimony 0.0084% and arsenic 0.026 per cent."

In practice the difficulty has been to obtain adequate contact between the refractory mineral particles and the aluminium surface. Various expedients have been tried but so far the best results have been obtained by first passing the pulp through a tube mill loaded with small aluminium ingots instead of pebbles, and then sending it to an agitator tank lined with aluminium plates, in which it is kept agitating for about 12 hours. Even so the action is incomplete because if a sample of the pulp be taken after treatment is finished and divided into two parts, one of which is cyanided direct and the other subjected to further

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[1] Mining and Scientific Press, September 27th, 1913.

desulphurizing in a bottle and then cyanided, the latter will invariably yield a better silver extraction.

The dyscrasite is unaffected by this treatment, but will dissolve in the cyanide solution, given time and sufficiently fine comminution.

As regards costs the above mentioned article gives the following figures:

"In connection with the desulphurizing treatment, treating 7268 tons per month, the following data are available:

COLLECTING DESULPHURIZING AND TRANSFERRING OF PULP

	Per ton
Labor	$0.050
Supplies (aluminium, 0.81 lb.; caustic soda, 1.46 lb.; lime, 5 lb.)	0.347
Power	0.027
Workshop	0.008
Total	$0.432

ALKALI SOLUTION, FILTERING AND TRANSFERRING

	Per ton
Labor	$0.069
Supplies	0.006
Power	0.028
Workshop	0.002
Total	$0.105

"The desulphurizing treatment effects a saving of from one to four ounces per ton, depending on the amount of refractory minerals present, at a total cost of 54 c. per ton."

In the cyanidation of this ore, whether previously desulphurized or not, it was found that a serious deterioration occurred in the extractive power of the stock solution after repeated precipitation by zinc, an effect which was shown to be due to some obscure interaction between the zinc gradually accumulated in the solution and the arsenic and antimony derived from the ore. This was overcome by the substitution of aluminium for zinc as a precipitant, a method which proved so efficacious that the stock solution after the plant had been running twelve months was more active in dissolving power than a freshly prepared cyanide solution.

Cyaniding of Concentrate.—The question as to whether concentration shall be included in the treatment of a given ore will often depend on the possibility or otherwise of recovering the precious metals from the concentrate at the mine. It may be suggested that if the concentrate can be cyanided after being separated from the gangue, why can it not be equally well cyanided if allowed to remain in the pulp going to the cyanide plant? The answer is that a small bulk of refractory high grade material can be treated by special methods which would not pay if applied to the whole of the ore. For instance, a few tons a day may be profitably roasted or slimed or treated with extra strong solution or for extra long periods, all of which might be prohibitive if applied to the entire ore.

There are two ways of dealing with a concentrate in a cyanide mill; one is to regrind to the degree of fineness found to be necessary by experiment and then to discharge the resulting pulp into the stream of mill pulp flowing to the cyanide plant, where the mixture goes through the usual routine of the process; and the other is to keep it apart and treat it in a small plant specially designed for the purpose. The latter has the advantages of greater flexibility and of affording an opportunity to ascertain exactly what results are being obtained.

Roasting prior to cyanidation is often the only way to get a satisfactory extraction from refractory gold concentrates, and the resulting material may be either leached, or reground and treated by agitation. It is seldom found profitable to treat raw concentrates by leaching though this was sometimes done before the development of processes for cyaniding slimed products.

It is to be noted that desulphurizing-roasting of concentrate containing silver has the effect of rendering a large proportion of that metal insoluble in cyanide, so that for silver or silver-gold concentrates some kind of raw treatment must be devised, unless there are reasons for preferring a preliminary chloridizing roast.

The cyaniding of concentrate proceeds in the main upon similar lines to those of any ordinary ore and the method is worked out by a series of comparative tests in the ordinary way. In general it may be said that concentrates if they are to be treated raw need exceptionally fine grinding in order to yield a high recovery by cyanide; a long period of contact with the solution and extra cyanide strength are other factors that usually play an important part.

A word should be said here about cyanide consumption. This naturally has a tendency to be high considered in terms per ton of material because the concentrate contains most of the base mineral of the ore and also a high metal content that is soluble in cyanide. Part of this loss may sometimes be avoided and part recovered by regeneration.

One fertile source of cyanide consumption is the formation of ferrocyanide, and this may be prevented to a large extent by protecting the product from oxidation. Concentrates that are to be cyanided should be taken straight from the concentrators with the least possible removal of moisture and exposure to the air. When concentrate is exposed to the air in a porous moist condition an oxidation more or less rapid takes place with formation of ferrous and ferric sulphate and insoluble basic ferric sulphate. The first two can be removed by water washing but the latter cannot, and a plan was proposed in the early days of the Rand and has since been actually used on a commercial scale at the Goldfield Consolidated, Nevada, of giving a preliminary treatment with sulphuric acid, water washing, neutralizing and cyaniding. Removal of the iron sulphates in one or both of these ways reduces the formation of ferrocyanide to inconsiderable proportions.

Another plan which is sometimes very effective is to agitate with lime water and compressed air until all the ferrous compounds are oxidized to the ferric state. In one instance in the writer's experience a concentrate when treated direct, showed a cyanide consumption of 100 lb. per ton, 50 lb. of which was found to be combined in the form of ferrocyanide. By giving a preliminary treatment with lime water and compressed air agitation for 48 hours, the subsequent cyanide loss was reduced to 54 lb. per ton, the resulting solution showing only traces of ferrocyanide.

The remaining chief source of cyanide loss is due to metals soluble in the solution, among which may be named copper, zinc, and the precious metals. If the copper is soluble in weak acid, it might be profitable in some cases to give a preliminary acid treatment, but if the cyanide loss cannot be reduced by this means and is prohibitive in amount, it is probable that the cyanide process is not suitable for that material, though it might be worth while to experiment on the regenerative effect or electrolytic precipitation.

When zinc is present in the concentrate as a factor in cyanide consumption it can be removed from the solution with regeneration of the cyanide by means of sodium sulphide. This would be done, of course, after the precipitation of the silver as so to produce a zinc sulphide free from precious metal.

The amount of cyanide combining with silver in a rich silver concentrate will account for a large proportion of the apparent consumption. For instance, a concentrate assaying 400 oz. of silver per ton would need 32 lb. of cyanide (100% KCN) to form the compound $KAg(CN)_2$. In some cases the value of such an amount of cyanide would constitute the difference between profit and loss on the treatment. This can all be recovered and rendered available by using as precipitant either sodium sulphide or aluminium dust. The former has a limited sphere of usefulness since it does not precipitate gold, so that it would need to be supplemented by some additional process when gold is present. Moreover, if the silver were to be first thrown down with sodium sulphide and the gold afterward precipitated by zinc, the stock solution would become zinky with the result that when next the silver was precipitated the zinc would also be thrown down with it, yielding a base product for the clean-up.

Aluminium dust seems to offer an ideal method in such cases as it precipitates both gold and silver, regenerates all the combined cyanide, and yields a high grade metallic product easily put into the form of bullion.

Cyanidation of Flotation Concentrate.—At the time of writing very little work has been done along this line. There seems to be a widespread anticipation of difficulties arising due to the presence of the oil, but in the cases thus far investigated by the writer, the samples of flotation concentrate gave no evidence of being much different from concentrates recovered by mechanical methods, as regards their amenability to direct cyanidation. Some writers, however, have found considerable difficulty in cyaniding such material, and suggestions have been made as to the preliminary treatment of it with solvents for the oil such as caustic soda. There are indications that the kind of oil used in the flotation has an effect on the subsequent amenability of the concentrate to cyanidation, and there is a considerable field for investigation of the subject.

CHAPTER X

PRECIPITATION

Whatever method of precipitation be used it is of great importance that the solution should be absolutely clear and free from suspended matter, and in practice it is almost impossible to obtain solution from slime treatment, either by decantation or by means of the various types of slime filter, that is sufficiently clean to give the best results. It is therefore almost always advisable to clarify it by filtration before precipitation.

There are special presses on the market for this purpose, or a cheap and efficient filter may be made by using submerged filter leaves of the Butters type, operated either by vacuum in an open tank or by pressure in a closed receptacle. Perhaps the simplest and cheapest device is a sand filter, or better a pair of such filters for alternative use. They may be circular tanks 20 to 30 feet in diameter and 3 or 4 feet deep with a filter bottom such as is used for leaching tanks, covered with six inches of clean river sand of from 20 to 40 mesh. The suspended matter will collect in a thin film on top of the sand, and when percolation becomes too slow it is only necessary to drain the tank dry, skim off the surface with shovels, and sprinkle a little more clean sand over it, the whole operation taking only a few hours.

To ensure the maximum efficiency and minimum of trouble with these filters it is advisable to run the solution first to a settling tank of about the same size fitted with a baffle board extending right across it, and from the top downward about 18 or 20 inches; this baffle gives a tranquil surface and good settling conditions. The solution is introduced on one side of the baffle and flows out at the opposite side into the sand filter, the tank remaining all the time full of solution. Most of the suspended matter will settle down to the bottom of this tank which is periodically decanted and the collected slime hosed out either to the agitators or slime filter. By this device a minimum of work is thrown on the clarifying filter and under ordinary circumstances it should need skimming only about once a month and may even run two months without attention.

Precipitation by Means of Zinc.—The recovery of the precious metals from the cyanide solution is almost universally accomplished by precipitation with zinc, either in the form of fine threads or of dust (the condensed fume recovered in the process of retorting the metal).

The phenomena of precipitation are essentially electrical and may be traced to the action set up by two metals of different potentials forming a galvanic couple, with the cyanide solution as electrolyte.

The chemical changes theoretically accompanying this action are expressed by Park[1] in the equation

$$2KAg(CN)_2 + Zn = K_2Zn(CN)_4 + 2Ag$$

and Julian and Smart[2] apparently assume the same reaction though they do not give the equation. According to this 0.166 parts by weight of zinc should be needed to replace 1 part of gold, and 0.303 parts of zinc to replace 1 part of silver. Clennell,[3] however, prefers the reaction

$$\begin{gathered} KAg(CN)_2 + 2KCN + Zn + H_2O = \\ K_2Zn(CN)_4 + Ag + H + KOH, \end{gathered}$$

in accordance with which it would require 0.332 parts of zinc to 1 part of gold and 0.606 parts of zinc to 1 of silver. In favor of the former equation it may be noted that the normal practice in the Pachuca district of Mexico when using zinc dust shows a consumption of zinc of not over 1 pound per pound of bullion and often as low as 0.8 lb. of zinc, while with the Crowe Vacuum Process even lower zinc consumptions have been ob-

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[1] The Cyanide Process, page 180 (Fifth Edition).

[2] Cyaniding Gold and Silver Ores, page 139 (Second Edition).

[3] The Cyanide Handbook, page 123 (Second Edition).

tained. Now by Clennell's equation the theoretical proportions would be 0.6 of zinc to 1 of silver, and it would be rather surprising if the practical and theoretical figures were to approach so close to one another, especially when it is remembered that both the free cyanide and the free alkali in the solution tend to attack the zinc and dissolve it, and also that there is usually more or less unused zinc present in the precipitate after cleaning up. On the other hand Clennell's reaction would better agree with the admitted fact of the difficulty of producing precipitation without the presence of a fair amount of free cyanide and would also explain the observed rise in alkalinity after precipitation.

Park's equation does not show the liberation of hydrogen as being an essential to the reaction, though the dissolution of some of the zinc by free cyanide and free alkali may be sufficient to account for the almost universal generation of that gas in the well-known copious amounts.

$$4KCN + Zn + 2H_2O = K_2Zn(CN)_4 + 2KOH + H_2$$
$$2KOH + Zn = K_2ZnO_2 + H_2$$

Whatever be the exact terms of the reaction zinc replaces the precious metal in the solution and there is a tendency for this zinc to accumulate in the stock solutions after repeated use. This accumulation, however, in practice only proceeds to a limited extent and this fact has been the subject of speculation. Of course, a small quantity of the stock solution with its burden of zinc is continually passing out of the plant in the residue moisture but where the usual water washes are used this will not account for the whole effect noted. Some zinc is no doubt precipitated as sulphide by soluble sulphides formed from the ore during the dissolving treatment, but it would seem that both these causes need to be supplemented by something further. The writer is inclined to account for it by the effect of the lime added to the pulp upon the double zinc cyanide. It is almost universally admitted that there is a considerable rise in the free cyanide strength of the precipitated stock solution when it comes in contact with lime and this may be accompanied by a precipitation of the zinc in an insoluble form, either as calcium zincate,

$$K_2Zn(CN)_4 + 2Ca(OH)_2 =$$
$$2KCN + Ca(CN)_2 + CaZnO_2 + 2H_2O;$$

or, as Walter Virgo[1] suggests, in presence of carbonic acid, a double carbonate of zinc and calcium may be formed

$$K_2Zn(CN)_4 + 2Ca(OH)_2 + 2CO_2 =$$
$$ZnCa(CO_3)_2 + 2KCN + Ca(CN)_2 + 2H_2O$$

White Precipitate.—It often happens, more especially in the precipitation of solution containing less than 0.1% of free cyanide, that a white deposit is formed on the zinc shavings, generally in the shape of little excrescences adhering to the metal and in its extreme manifestations cementing the zinc together in solid white masses quite impervious to the solution. It is usually found to consist principally of zinc hydroxide with varying amounts of zinc cyanide and zinc ferrocyanide and sometimes calcium compounds. Its formation is sometimes said to be promoted by a high lime content in the working solution but the writer has never seen any evidence to support this view. This white precipitate is a nuisance because it both interferes with proper working of the zinc boxes and also yields a low-grade product which is difficult and expensive to melt.

As to the remedy it is not easy to make suggestions applicable in general apart from a consideration of the individual conditions. A higher strength of free cyanide and free alkali will usually be effective in preventing its formation, but circumstances may not always warrent such an increase. In that event the next best thing is, probably, to dress the boxes more frequently so as to detach the coating from the zinc and keep a clean metallic surface exposed, preventing also the formation of patches and layers impervious to the flow of solution.

Action of Oxygen in Precipitation.—In regard to this question some interesting facts have been noted in connection with the recently devised method for determination of oxygen in working

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[1] *Journal Chem., Metall. and Min. Soc. of S. A.*, August, 1903.

solutions already described in the chapter on "Testing and Analysis of Solutions," page 48. In the course of the article by H. A. White referred to and the discussion thereon it is shown that (1) the oxygen contained in the solution entering the zinc boxes is rapidly eliminated by the reducing action of the hydrogen formed and that the rate of precipitation increases as the oxygen content is diminished, and (2) that the formation of white precipitate seems to be a function of the amount of oxygen present, and that when the solution reaches a point where the oxygen has been practically eliminated, no white precipitate is formed even under conditions normally favorable to its production. The first observation tends *a priori* to demonstrate the advantage likely to be derived from a mechanical removal of the dissolved oxygen from the solution before precipitation while the latter suggests the possibility that by such a preliminary elimination of the oxygen the formation of white precipitate might be entirely prevented.

The Crowe Vacuum Process.—It is interesting to note that almost at the same time that the action of oxygen in precipitation was attracting attention on the Rand, Thomas B. Crowe at the Portland Mine, Cripple Creek, was perfecting a process for the mechanical removal of dissolved oxygen from solutions, arguing that precipitation being a reducing process the less oxygen present the more favorable would be the conditions for the reaction. The invention consists in spraying or otherwise disintegrating the solution in a vacuum chamber by which means the dissolved oxygen is removed before the zinc dust is added, resulting in a reduction of the zinc consumption and also in some instances a reduction in the free cyanide loss which accompanies zinc precipitation. The process which is at its best when used in combination with zinc dust has shown savings in zinc amounting to two-thirds of the former consumption. In the case of zinc shaving precipitation its benefit is not so marked on account of the extreme rapidity with which the solution re-absorbs oxygen, but even here in the few instances in which it has been tried it appears to have resulted in a saving of zinc coupled with the possibility of obtaining good precipitation from solutions too low in free cyanide to precipitate under normal conditions.

The Zinc-lead Couple.—The principle of coating the zinc with a film of lead to increase the activity of precipitation was patented by MacArthur in the early days of the process but it was not till the occasion arose for precipitating the exceedingly dilute solutions obtained from slime treatment on the Rand that the process attained to commercial prominence. When the decantation process of slime treatment was first mooted the impossibility of precipitating the solutions by zinc was assumed and the Siemens and Halske electrolytic process was considered indispensable. This had its drawbacks, however, and in the search for a substitute it occurred to W. K. Betty to try the zinc lead couple. He found that by its use, combined with a drip of strong cyanide solution into the head of the box he could precipitate satisfactorily solutions carrying as low as 0.005% in cyanide. The ordinary zinc shavings were dipped in a solution of lead acetate containing from one to five per cent. of lead, thus coating them with a deposit of spongy metallic lead. This zinc was then placed in the extractor box and covered with solution as soon as possible. A barrel containing concentrated cyanide solution was placed at the head of the box, and as soon as the pregnant stock solution was turned in a small drip of this strong solution was allowed to run in with it, and maintained for the first twelve hours. Julian and Smart[1] state that this feature was later on abandoned.

When working with this process even under the most favorable conditions it was not found possible to maintain effective precipitation for more than five or six days after starting, and when the action became sluggish the zinc was transferred to the sand plant precipitation system and replaced by new.

Solutions Containing Copper.—These sometimes give trouble by forming a continuous and adherent skin over the zinc and stopping further precipitation of the gold and silver. This effect is more often observed when the solution is low in free

[1] Cyaniding Gold and Silver Ores, page 139 (Second Edition).

cyanide, and in strong solutions copper interferes very little with the proper working of the boxes. When trouble occurs from this cause the zinc lead couple may be found a valuable aid in counteracting it. Park[1] states that its effect is to keep the copper in solution allowing only gold and silver to be precipitated but at the Creston Colorada mine in Sonora, Mexico, a system was in vogue, about the year 1900, devised by Mr. Higgins, the metallurgist, of precipitating the gold and silver with a minimum of copper, out of the strong and rich solution on ordinary zinc shavings and passing the weak and low grade solution through boxes in which the zinc was heavily coated with sponge lead, for the purpose of throwing out the copper and preventing its accumulation in the stock solution, which it was said to accomplish very satisfactorily.

Should the presence of copper be found an insuperable obstacle to precipitation by zinc shavings, the zinc dust method will usually not be open to the same objections. The electrolytic process also may sometimes be effectively employed, either for recovering the gold, silver and copper together in the form of base bullion or for cleaning the stock solution of its copper content after recovering the gold and silver by the zinc dust method.

There is an interesting process in use at Dayton, Nevada, for precipitating the solution resulting from the treatment of the accumulated coppery tailings from the old pan amalgamation mills of Virginia City. The solution after the necessary contact with the ore is heated to about 200 deg. F. and strengthened with fresh cyanide before the zinc dust is added. By this means not only is no trouble experienced about precipitating the silver, but a large proportion of the cyanide combined with copper is said to be regenerated.

Method of Working When Using Zinc Shavings.—The extractor box is a long narrow box made of wood or steel and divided up into from five to ten compartments by partitions, and baffles so arranged that the solution rises through the column of zinc in each compartment and descends through the narrow space between

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[1] The Cyanide Process, page 183 (Fifth Edition).

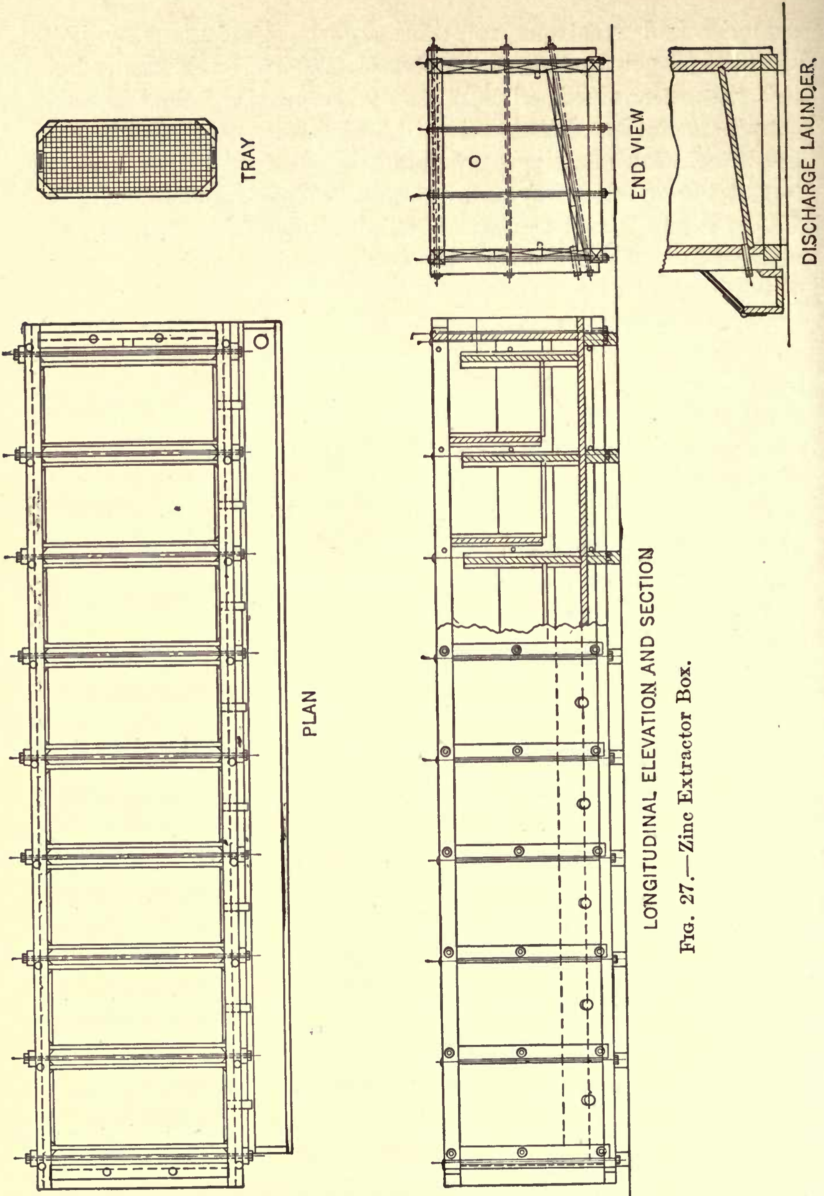

Fig. 27.—Zinc Extractor Box.

the partition and baffle. A tray whose bottom is composed of iron screening of about ¼ inch mesh rests on cleats six inches above the bottom of the box so as to allow space beneath for the collection of the gold and silver sludge as it becomes detached from the zinc.

Should the boxes be made of steel it is better to keep the interior surface well covered with P. and B. paint as it has been found that when the solution contains much caustic soda, a couple is formed between the iron and the zinc, resulting in a marked increase in zinc consumption. The same applies in a less degree to the use of iron screening on the zinc trays and some metallurgists have substituted screens made of fibre or some non-metallic material.

The zinc filaments or shavings are packed in on top of the tray almost to the height of the compartment, care being taken that the whole space is uniformly filled especially in the corners. A good method, devised by L. M. B. Bullock, is to wind the shavings off the lathe into wisps or hanks about three inches in diameter and an inch or two longer than the measurement of the compartment. When charging a box these hanks are doubled over at each end to the exact size and laid carefully side by side and close together forming a layer of one hank in thickness with all the threads parallel to one another. Another layer is then placed on this but transversely, and so on till the compartment is filled. The idea of making the hanks a little too long and then doubling the ends is to give increased compactness and density at the sides and corners where the greatest channelling occurs.

A fine thread-like shaving is usually preferred to the ribbon form and the thickness should be such as to give the maximum surface per unit of weight without making it so slender as to disintegrate rapidly into short pieces under the action of the solution. A width of from 1/500 to 1/800 of an inch is probably the most useful.

The zinc, especially in the first three compartments should be dressed frequently. If the solution is strong in cyanide and rich in metal this may need to be done every day and the lower compartments every four or five days.

In dressing and also in cleaning up the most approved practice is to move up the zinc from the lower compartments to replenish that which has been dissolved in the upper ones, filling the spaces thus left vacant with new zinc. By doing this the bulk of the precious metal is concentrated and maintained in the head compartments, minimizing the risk of precipitate being carried over the end of the box mechanically, and keeping a body of new clean zinc for stripping the last traces of metal from the solution. Moreover, it is only by this means that the partly consumed zinc can be completely used up. Even so there is always a certain amount that breaks into short pieces which tend to mat together and render the compartment impermeable to the solution.

When the amount of the broken zinc is not excessive it can be disposed of by sprinkling a thin layer between each layer of long zinc in the upper compartments, but if, as sometimes happens, the quantity grows to unmanageable proportions, screen trays may be placed on top of the head compartments just below the level of the solution and the short zinc laid in a thin layer on the screens, being removed each day and washed to detach the adhering precipitate and expose fresh surfaces to the action of the solution. Another useful device for working up these zinc shorts is a small agitator tank about three feet diameter and two feet deep with a circular overflow launder. Between this launder and the rim of the tank is placed a circular screen of 30- to 60-mesh wire cloth. The short zinc arising from each day's clean-up of the boxes is placed in the tank and a small stream of pregnant solution run in, the stirring gear keeping the whole charge in continuous agitation. As the rich solution dissolves the zinc and precipitates the gold and silver, the stream of solution continuously flows out through the circular screen into the annular launder, carrying with it the precipitate. This stream enters the head of one of the precipitation boxes and deposits its suspended material there, its soluble content being also completely precipitated as it flows on down through the zinc shavings.

In dressing the boxes and especially in cleaning up it is most important not to allow the zinc to remain exposed to the air, as it gets hot and oxidizes rapidly, resulting in a considerable increase in zinc consumption and also favoring a formation of the obnoxious "white precipitate."

Capacity of Zinc Boxes.—The total cubic capacity of zinc boxes necessary varies with the quantity of solution and its metal content and also with the method used for packing them with zinc shavings. The following were the conditions at Butters Copala Mines in 1906 where the boxes were packed by the Bullock method.

There were two boxes of eight compartments each; both were 24 inches deep but one was two feet wide and the other, one foot six inches; the width of the compartments was in every case fifteen inches.

Average value of solution before precipitation $1.01 in gold and 3.41 oz. in silver. Average value of solution after precipitation, at the rate of 15 tons per hour traces, at the rate of 25 tons per hour, $0.06 gold and 0.03 oz. silver.

Daily tonnage of solution	360 to 600
Gross capacity of the two boxes, cu. ft.	70
Cu. ft. in boxes for each ton of solution treated daily	0.19 to 0.12
Space actually occupied by zinc shavings, cu. ft.	49.5
Solution per day for each cu. ft. of zinc, tons	7.2 to 12.1

When the boxes are packed by the usual method of throwing the loose shavings in, armful by armful, and pressing them down, the space necessary may be three or four times that allowed at Copala and for low-grade gold solution especially those from slime treatment a cubic capacity ten times as great as that quoted may be necessary.

Zinc Dust Precipitation.—The method of applying the zinc to the solution in the form of dust or fume was first introduced by Sulman and Pickard but it did not attain to a wide commercial importance until C. W. Merrill devised his modification of it for the Homestake Mine. The special point about the latter process consisted in adding the dust to a flowing stream of solution instead of, as heretofore, to an agitating cone or tank. The dust is released from its receptacle by a feeder whose speed is adjustable and falls into a mixer or emulsifier where it is thoroughly wetted and then passes, usually, directly into the suction pipe of a triplex plunger pump, discharging into the precipitate press. The action is so rapid that it is practically complete by the time the emulsion reaches the chambers of the press but as a measure of precaution the delivery pipe is lengthened as much as possible, sometimes extending to 200 to 300 feet before reaching the press.

If desired, the special triangular press designed for the purpose by the Merrill Metallurgical Company may be used but the writer has found that the ordinary square type of press answers the purpose equally well, the only thing necessary being to ensure that the filtering area is large enough to allow the necessary rate of flow per hour to pass through it, such area being for practical purposes about 1 sq. ft. per ton per day.

In order to obtain a complete precipitation, that is, to produce an effluent assaying only 2 or 3 cents, a rather heavy feed of zinc dust is sometimes necessary resulting in a low-grade precipitate for melting. This is more especially noticeable in the treatment of low-grade gold ores and is less in evidence where the solution contains several ounces of silver per ton. At the Homestake it is often found advisable to run a drip of lead solution, either nitrate or acetate, at the point where the zinc dust enters the stream, in order to intensify the action. At this mill the precipitate is so low in grade that it is first acid-treated and then dried and fluxed with litharge preparatory to being finally cupelled. By this means a bullion 980 fine is produced.

In order to reduce the quantity of zinc dust used while at the same time producing a barren solution for residue filter washes J. S. Colbath introduced a combination system at the mill of the El Rayo Mining Company. This consisted in carefully adjusting the supply of zinc dust and maintaining it at the lowest possible point so as to precipitate the greater part of the precious metals and allowing the effluent solution to assay as high as 20 to 40 cents per ton. A part of this effluent solution, just sufficient in quantity to provide barren washes for the ore residues was diverted and passed through one or more boxes of zinc shavings to strip the remaining assay values and produce a strictly barren solution. By this means an unusually low consumption of zinc dust was produced and the resulting precipitate was considerably higher in value than it would otherwise have been. The cost of maintaining the box of zinc shavings was small as only part of the daily solution tonnage went through it and it needed very little attention on account of the small quantity of precious metal deposited therein. In recent years, however, the zinc dust process has been so developed and improved that it is now hardly worth while going to the additional trouble of the double system.

There are several advantages in the zinc dust method. (1) Zinc dust costs less per pound than zinc shavings. (2) Less zinc is usually consumed per unit of fine bullion than in the case of zinc shavings. (3) It is superior to shavings for precipitating coppery solutions because it does not afford the same opportunity for the copper to form an impervious skin over the zinc surface and stop further action by the solution. (4) The laborious and disagreeable task of dressing and cleaning up the zinc boxes is avoided, and (5) greater security from pilfering the precipitate is effected.

With clean solutions high in silver it is possible, even when the El Rayo system is not used, to produce bullion which by direct melting assays over 900 fine with a consumption of zinc dust not over one part to 1 of fine bullion, by weight; indeed in some of the Pachuca plants the zinc dust consumption has been less than 0.8 kilo per kilo of fine bullion.

Electrolytic Precipitation.—This was introduced on a commercial scale as an adjunct to the cyanide process on the Rand for precipitating the weak and low-grade solutions resulting from the treatment of slime by decantation, and was known as the Siemens Halske process. The anodes were of sheet iron enclosed in burlap to prevent short circuiting and the cathodes were formed by hanging four or five sheets of lead foil cut into strips upon a stiff wire, and shaking them till the strips separated and formed a bunchy mass. The anodes were placed about four inches apart and the cathodes occupied the entire space between. It will be noted that the cathode area was four or five times greater than that of the anode. The current density commonly used was 0.04 ampere per square foot of anode surface and about 16 cu. ft. of box capacity were required per ton of solution per 24 hours. The gold was deposited as a firmly adherent film or plating and was recovered each month by melting down the cathodes and cupelling the resulting pigs of lead. The litharge derived from this operation was reduced in a reverberatory furnace at the mine and sold to the Rand Central Ore Reduction Company who treated it by Park's process and then rolled the softened and barren lead into foil which was sold back to the mine companies.

The anodes slowly dissolved with formation of ferrous hydroxide and a little prussian blue, forming a sludge which often assayed high in gold. The sludge was either sold to the Rand Central Company or roasted and treated by cyanide at the mine. The process had its drawbacks, the chief of which were an uncertainty and variability in the results obtained, a lack of simplicity, and the loss and expense incidental to the handling of various by-products, and it was gradually supplanted by the development of the zinc lead couple.

At the Minas Prietas plant of Charles Butters and Company for the treatment of pan amalgamation tailing containing copper, H. T. Durant introduced an important modification of the South African practice. Owing to the large amount of copper that was deposited with the gold and silver the cupellation of the lead foil cathodes was impossible, so he conceived the idea of raising the current density sufficiently to cause the whole of the precipitated gold, silver and copper to be deposited as a non-adherent sludge which dropped off and fell to the bottom of the box from which it was removed each month in a manner similar to that used in cleaning up a zinc box. The detachment of the deposit from the cathode was assisted by periodically wiping down the plates with rubber-edged brushes without removing them from the box.

The anodes were made of lead sheets 3⁄16 in. thick, peroxidized by being placed in a bath of 1% potassium permanganate and subjected to the action of the current for one hour at a current density of about one ampere per square foot. Owing to the use of graphite in the rolling of the lead sheets it was found necessary to scour off the surface film so as to expose a clean metallic surface to the action of the solution. These plates had a life of from twelve to eighteen months under the conditions to which they were subjected. They were placed in the compartments parallel with the flow of solution and about seven inches apart from centre to centre. The cathodes which were made of commercial tin plate were set in the centres of the spaces between the anodes so that the distance between anode and cathode was from 3 to 3½ inches. Both electrodes were soldered to their respective copper leads and remained permanently in position. The current density varied in the different boxes from 0.25 to 0.56 amperes per square foot and the voltage across the electrodes was from 2.5 to 3.

There were six boxes, each 30 feet long and 10 feet wide, divided up into twelve compartments with baffle boards similar to those used for zinc precipitation. Each compartment contained 18 anodes and 17 cathodes. Four of the boxes were used for weak solution and two for strong.

WORKING RESULTS

Weak Solution.—Current density, 0.25 ampere per square foot; daily quantity of solution, 480 tons, total anode area 13,536 sq. ft.

Head assay		Tail assay		Extraction, per cent.	
Au	Ag, oz.	Au	Ag, oz.	Au	Ag
$1.10	1.45	$0.13	0.15	88	89.6

Strong Solution.—Current density 0.55 ampere per square foot; daily quantity of solution, 216 tons.

Total anode area 6950 square feet.

Head assay		Tail assay		Extraction, per cent.	
Au	Ag, oz.	Au	Ag, oz.	Au	Ag
$2.93	3.51	$0.26	0.29	91	91.7

Regeneration of Cyanide in Electrolytic Precipitation.—Theoretically it should be possible to regenerate a part at least of the cyanide combined with the metals dissolved from the ore but under the usual conditions obtaining in practice there is also a tendency to oxidize the free cyanide present, so that the loss and gain about balance one another, and in electrolyzing solution containing only the usual amounts of gold and silver little or no net gain in cyanide is observed. When copper is also present, however, the case is for some reason different and a marked rise in cyanide strength is noticed at the tail of the boxes. At Prietas this gain averaged for strong solution 0.0146% KCN or 0.292 lb. per ton of solution and for the weak solution 0.01189% or 0.236 lb. per ton. The actual total cyanide consumption in the plant was from 2.5 to 3 lb. per ton of ore, and the amount regenerated in precipitation worked out at 0.58 lb. KCN per ton of ore treated.

This Minas Prietas method is likely to prove useful when treating ores containing copper soluble in cyanide solution, both on account of the recovery of some of the cyanide and also of the avoidance of difficulties caused by the copper in zinc precipitation. Its chief drawbacks are (1) the low efficiency of the current. The best efficiency shown was only about 14% and the worst was 5%. (2) The large cubic capacity and floor space required. (3) The failure to regenerate all the combined cyanide. In treating clean silver ores no regeneration whatever is apparent and even in the case of the coppery tailing at Minas Prietas it was only about ¼ of the theoretically possible amount. (4) The disintegration of the anodes and cost of replacement. This perhaps may be overcome by the use of fused magnetite or passive iron.

Some interesting work in the development of electrolytic precipitation is detailed in a paper by Hugh Rose (Bulletin A.I.M.E., Aug., 1916), the experiments being carried out by A. W. Hahn at the Santa Gertrudis Mill, Pachuca, Mexico. The first noteworthy point is that an anode composed of lead alloyed with 6 to 9% antimony was found to give better service than any material heretofore tried. And the second point is that in investigating the destruction of free cyanide by the action of the current such destruction was located at the anode and it was found that this could be largely obviated by maintaining a high concentration of protective alkali in the neighborhood of the anode. A porous diaphragm such as canvas separating the anode and cathode assisted in maintaining an alkaline anolyte, circulation of anolyte and catholyte being accomplished individually. In these experiments which were made principally on barren solution with a view to a regeneration of the cyanide, a substantial recovery of cyanide was shown, the cathode deposit consisting chiefly of zinc. It would seem, however, that if electrolysis is to be used its proper application would be in recovering the gold and silver from solution instead of in removing the by-products of a previous chemical precipitation. But however that may be the data obtained in this investigation are highly suggestive and might well be made the starting point for further research.

Regarding electrolytic precipitation in general, its application in the present stage of its development has a very limited utility in the cyanide process and it has hitherto been unable to compete with other methods. It would seem, however, to offer the most reasonable and scientific process for dealing with the metal in cyanide solutions and it is not unlikely that with further study and research ways may be found to surmount the present difficulties and drawbacks.

Aluminium Precipitation.—The use of aluminium as a precipitant for gold out of cyanide solutions was patented by Moldenhauer in 1893 but he does not appear to have made any practical use of the process. H. F. Julian experimented with it, in the same year, but soon abandoned the idea. In 1910 S. F. Kirkpatrick introduced it at the Deloro Smelter in Canada, applying the metal for the first time in the form of a powder, and achieved a complete success. He later installed it at the O'Brien Mine in Cobalt.

When making tests on the cyaniding of the Nipissing low-grade ore it was found that the arsenic and antimony dissolved out of the ore by the cyanide formed a harmful combination with the zinc derived from zinc precipitation causing a serious loss of dissolving power in the stock solutions. Consequently an alternative method of precipitation was sought for, and that in use at the O'Brien mine naturally suggested itself. The experiments were so successful that the process was introduced in the new mill at its start and the results fully justified the choice, the stock solution at the end of twelve months being more active for dissolving silver than freshly made up solution.

The fact that aluminium does not replace the precious metals in the cyanogen compound renders necessary the presence in the solution of caustic soda. There are two possible equations to represent the reaction incident to precipitation.

The first is suggested by Moldenhauer:—

$6NaAg(CN)_2 + 6NaOH + 2Al = 6Ag + 12NaCN + 2Al(OH)_3$, the aluminium hydroxide at once dissolving in the excess of caustic to form sodium aluminate,

$$2Al(OH)_3 + 2NaOH = Na_2Al_2O_4 + 4H_2O$$

This would indicate that one part of aluminium should theoretically precipitate twelve times its weight of silver, but in practice the amount is about three times its weight only, so the following may more nearly represent the actual reaction,

$$2NaAg(CN)_2 + 4NaOH + 2Al = 4NaCN + 2Ag + Na_2Al_2O_4 + 4H$$

The method of working is somewhat similar to that for zinc dust but the action is slower in practice, and necessitates a receptacle in which to retain the emulsion under agitation until the action is complete. The dust also is not nearly so easily wetted as zinc dust and a special form of agitator is used to assist in effecting this. The reason why the action is so slow is probably a mechanical one, because if an excess of dust be used in a strongly caustic solution the reaction is rapid, about two minutes only being required for total precipitation, but in practice the amount of precipitant used has to be cut down to the lowest possible point for the sake of economy and consequently a proportionately longer time is needed to bring each molecule of the cyanogen compound in contact with the metallic surface of an aluminium particle. It follows from this, at least in the case of solutions of the average low metallic content, that the quantity of precipitant necessary depends more upon the volume of solution to be precipitated than upon its assay value, and therefore it is more economical to concentrate the precious metal into as small a bulk of solution as possible for precipitation rather than to precipitate large volumes of low grade solution.

One point in the chemistry of the process is vital to its practical success and that is that at the time of precipitation lime must be absent, as it decomposes the sodium aluminate to form insoluble calcium aluminate,

$$(Na_2Al_2O_4 + Ca(OH)_2 = CaAl_2O_4 + 2NaOH),$$

which deposits in the press with the precipitate, making a large bulk of low-grade product which it is almost impossible to melt into bullion. At the Nipissing this condition of absence of lime does not present any difficulty because the slime settles well for the final decantation in the cyanide liquor with only the merest trace of lime, but difficulties are likely to arise with other ores from this cause and to meet such cases the writer devised the modified process described later.

The reaction between the lime and the sodium aluminate is turned to useful account after precipitation in preventing an undue accumulation of the sodium aluminate in the stock solution, because when the barren solution next comes in contact with the lime added in the mill it is broken up, the aluminium being precipitated as the insoluble calcium compound with liberation of caustic soda.

There is usually a tendency to a formation of soluble sulphides as the result of precipitation by this process. It has been suggested that this is caused by a decomposition of the soluble sulphocyanates present, but laboratory tests made in weak solutions of KCNS (0.05% to 0.15%) have failed to show any signs of such a reaction. Sodium thiosulphate is easily decomposed by aluminium dust with formation of sodium sulphide, so where soluble sulphides are found after precipitation they are probably due to the presence in the stock solution of thiosulphates.

At the Nipissing mill the formation of soluble sulphides was considerable and a few drops of lead acetate added to a little of the effluent from the press in a beaker would turn it a dead black. As the addition of lead salts to neutralize this sulphide seemed to have a prejudicial effect on the subsequent extraction of silver from the ore, the expedient was adopted of installing an air lift in the barren sump, and a few hours agitation in this way served to remove every trace of soluble sulphide, rendering the solution perfectly efficient in dissolving power when next brought in contact with the ore pulp. At the Butters Divisadero mill only traces of soluble sulphide were ever found in the press effluent and none at all in the barren sump, and it was not found necessary to take any precautions to counteract their effect.

Method of Working.—The apparatus necessary is similar to that in use for zinc dust with the addition of two small agitator tanks to afford the additional time of contact of the solution with the precipitant. Owing to the difficulty of wetting the dust a special form of agitator is used. The tanks for treating 600 tons of solution a day may be 6 ft. in diameter and 4 ft. deep. In each is a central shaft resting on a bearing on the tank bottom and rotated at from 40 to 60 revolutions per minute by means of a horizontal shaft with crown and pinion or by a quarter-turn belt. Bolted vertically onto the central shaft on opposite sides of it are two boards 10 inches wide and 1 inch thick, extending from the bottom to within 12 inches of the top of the tank. This device imparts a peculiar boiling motion to the contents of the tank quite unlike that produced by the ordinary long paddles at the bottom of a slime agitator, and the fact that the boards extend upward so near to the top of the tank serves to break the smoothness of the surface of the liquid, breaking up at the same time the film of aluminium dust which would otherwise have a tendency to float and accumulate there.

The two tanks are connected in such a way that the emulsion in the first is maintained at a constant level and overflows into the second whose function is chiefly that of a sump from which to operate the pump. The latter is at the Divisadero mill fitted with a by-pass on the delivery, leading back into the sump, and the flow through the by-pass is regulated by an automatic float valve, the object being to maintain a constant head of solution over the suction and prevent the entry of air into the pump. A better plan than this, however, is to arrange the float valve to control the main flow of solution from the pregnant storage into the precipitating tanks as this avoids the additional aeration of the solution caused by the entry of the stream flowing from the by-pass.

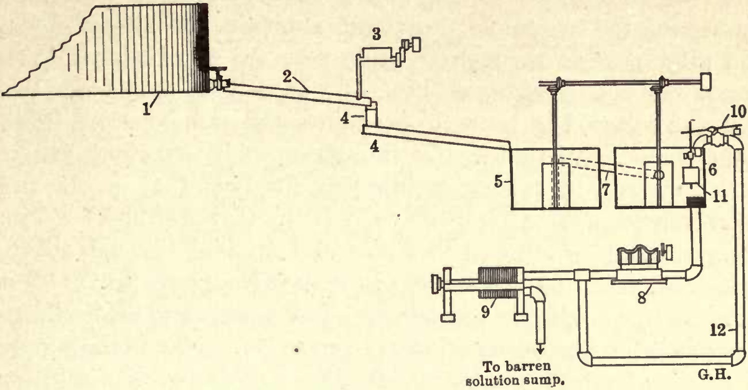

FIG. 28.—Aluminium Precipitation, General Arrangement of Plant.

The following data refer to an installation capable of precipitating from 500 to 600 tons of solution a day.

The solution after leaving the clarifying filter passes down through a covered launder to the precipitation tanks. In this launder is added the aluminium dust, through a worm-feeder whose speed is adjusted by means of a pair of cone-pulleys. When it nears the precipitation tanks the continuity of the launder is broken twice, allowing the solution a clear drop of 6 inches each time, to assist in wetting the dust.

Referring to Fig. 28 (1) is the clarifying filter, (2) launder, (3) the aluminium-dust feeder, (4) interruptions in the flow of solution, (5) and (6) the precipitation tanks. (7) is a 6-in. pipe connecting the two tanks; it extends almost to the bottom of (5), and after passing through the side near the top it slopes down somewhat and discharges into (6). It will be seen from this that the solution in tank (5) is maintained at a constant level, while the level of that in (6), though normally constant, can be varied at will, by raising or lowering the float (11) on the rod operating the automatic cut-off valve (10). (8) is a triplex 7 × 7 in. plunger pump running at 29 r.p.m. and drawing through a 5-in. pipe from tank (6) discharging into the filter-press (9). (12) is a 2½-in. by-pass line in the delivery of the pump, and its discharge is regulated by the automatic float-valve (10). Should the supply of solution from the clarifier be slackened off the level of solution in (6) would tend to fall, thus opening valve (10) and allowing the surplus solution passing the pump to return to the tank, in this way maintaining a constant level. The importance of this lies in the necessity of keeping the pump-suction water-sealed and preventing the entry of air. The operation of precipitation being a reducing one, the presence of air tends to retard it; moreover the silver precipitate formed by aluminium dust is so sensitive to re-solution influences that air in juxtaposition with cyanide solution will often cause the metal to re-dissolve in the filter-press itself.

There is also on the pump-delivery a pressure-gauge, and a relief-valve (not shown in the diagram) set at 80 lb., and discharging back into tank (6).

Hamilton's Process.—At the Nipissing mill the small amount of lime added in the battery is almost completely eliminated from the ore and solution by the time the latter is ready for precipitation, consequently no trouble is experienced by the deposition of calcium aluminate in the press; and no caustic soda is added before precipitation because more than sufficient for this purpose passes into the cyanide plant from the preliminary desulphurizing process. Neither of these conditions, however, is likely to be encountered in the majority of mills where the process might be introduced and to meet circumstances where the Nipissing routine is inapplicable the writer devised the following process, which was worked out at Butters Divisadero Mine with the assistance of P. H. Crawford, the resident metallurgist.

It consists in treating the solution before precipitation with soda ash, thus throwing out the lime as carbonate and substituting caustic soda therefor:

$$Ca(OH)_2 + Na_2CO_3 = CaCO_3 + 2NaOH$$
$$CaSO_4 + Na_2CO_3 = CaCO_3 + Na_2SO_4$$

It allows the use of all the lime necessary for neutralizing and settlement, it yields a lime-free solution for precipitation, and incidentally manufactures the caustic soda necessary for that operation, in accordance with the formula, given above.

The method of operation is as follows: The pregnant solution is pumped to a tank of 100 tons capacity, above the plant, fitted with a mechanical stirring-gear, and it is here that the soda ash is added. In order to find out exactly how much of the latter is needed, a sample test is employed. A standard solution of the soda ash is prepared by dissolving 6.25 grams in distilled water and making up to 1 litre, and is such that if 50 cc of pregnant solution be taken for the test, each cubic centimeter of soda ash solution will be equivalent to ¼ lb. of the dry salt per ton of cyanide solution. A portion of the solution to be tested is filtered, and 50 cc measured and placed in an Erlenmeyer flask or small beaker and heated over a flame until almost too hot to hold in the fingers. A measured quantity, say, 2 cc of the soda ash solution is then drawn from a burette and agitated for a few minutes to insure complete precipitation. The contents of the flask are filtered, and the filtrate divided into two parts: to the first part is added a few drops of potassium oxalate or sodium carbonate solution, and if any turbidity results it indicates an insufficiency of soda ash. In that event a fresh portion of the original solution is taken and the test repeated, increasing the soda ash to 2.5 or 3 cc, filtering, and testing as before. This operation is repeated, if necessary, until no turbidity is produced on adding oxalate. When a test is secured in which no lime is indicated, the second portion of the filtrate is tried for excess of soda ash, by adding a few drops of calcium chloride solution. If no turbidity is produced, the required quantity has been found, and the number of cubic centimeters of soda ash solution multiplied by 0.25 and by the number of tons of solution in the tank gives the amount of soda in pounds to be added to the charge. Should the addition of calcium chloride, mentioned above, indicate that there is an excess of soda present, the original test may be repeated, using 0.5 cc less of the standard soda solution.

A slight excess of soda ash seems to be unimportant, but it is essential that no lime should be indicated.

It usually takes from 10 to 15 minutes to ascertain the amount of ash needed for a charge, and when this has been done the required quantity is placed in a screened drum hanging under the inflow pipe, and the pump and agitator started. By the time the tank is filled to the required tonnage mark the reaction is complete and the charge is allowed to settle, the clear solution being then decanted to a sand-filter clarifier, whence it passes to the precipitation house. The turbid remainder is afterward stirred and run through a valve in the bottom into a smaller tank below fitted with a filter-bottom connected to a vacuum-pump; here the carbonate of lime settles and allows a further decantation of clear solution to the sand-filter. The result of several days' operations is allowed to accumulate in this filter-tank, and finally the vacuum is turned on and the precipitated lime filtered to a firm cake. After a water-wash, to remove pregnant solution, the deposit is shoveled into empty cyanide tins and allowed to dry in the air for a few days, and then charged back into the lime-kiln to be re-burnt and used over again. If desired the deposit of carbonate of lime may be allowed to remain in the soda tank for several days, and then after close decantation allowed to run out to the residue pulp filter storage.

The precipitate obtained by the use of aluminium dust is bulky and porous; it has a rich brown color, and usually melts readily with a minimum of flux though in some cases it appears to be somewhat more refractory than zinc precipitate. On an average it contains about 85% of the precious metals, and when melted yields bullion 980 fine.

As far as has been ascertained the process is not suitable for solutions containing only gold though the gold contained in solutions assaying a couple of ounces or more of silver per ton is almost completely precipitated. Copper is only precipitated to a slight extent even in the presence of a considerable amount of silver.

Advantages of Aluminium as a Precipitant.—The use of the process as a substitute for zinc precipitation when treating silver ores containing antimony and arsenic will usually result in an increased extraction, both of gold and silver, which in some cases has been as much as 10% of the original assay value, but apart from this its chief advantage will be in the saving of cyanide in the treatment of silver ores. The equation already given shows that the whole of the cyanogen combined with silver is regenerated in the form of alkaline cyanide, and this theoretical increase in cyanide strength is borne out by titrating the solution before and after precipitation. At the Butters Divisadero Mill, however, in the course of a comparative run between zinc and aluminium, taking three representative months on each process, the reduction in cyanide consumption actually shown during the period covered by aluminium precipitation, was more than double the amount calculated as having been in combination with silver, and the balance of the saving was probably due to the elimination of a source of loss mentioned under the head of zinc precipitation, that is, the direct combination of cyanide with zinc.

There is one drawback to the process that may be in evidence where filters are used for handling the pulp direct from the treatment tanks. The filter cake before washing will obviously contain solution high in lime and when the barren solution containing sodium aluminate strikes this soluble lime an insoluble precipitate of calcium aluminate is formed, as already noted. It happens however that the reaction is not instantaneous and therefore continues after the solution has reached the canvasses resulting in a deposition of calcium aluminate in the fiber and reducing the porosity and consequently the filtration rate very materially. Moreover this deposit does not readily dissolve in weak muriatic acid so that the consequences in loss of filtration efficiency may be serious. This was not in evidence at the Divisadero mill on account of the use there of the special leaf with replaceable filter medium already described (page 111). In ordinary cases, however, if the trouble is serious it could no doubt be remedied by adding sifted lime powder or milk of lime to the barren solution sumps which for this purpose would be fitted with some form of agitator. By this means the reaction between the lime and the sodium aluminate would take place before the solution entered the filter cake, thus rendering the solution harmless in this respect. It would probably not be necessary to settle or filter out the suspended matter from the barren solution before sending it to the filter as this would merely deposit in a thin film on the outside surface of the filter cakes and would thus continually eliminate itself. In cases where the Dorr counter current system of washing is in use the trouble would not appear at all even when filtration is resorted to for dewatering the product of the final thickener, because the reaction between the lime and the sodium aluminate would take place in the thickener and be complete before the pulp reached the filter.

As to relative costs of the two processes, the figures ascertained in the above-mentioned test were, zinc $0.076, aluminium, $0.071, soda ash, $0.036, showing that the aluminium process cost $0.031 per ton more than zinc but resulted in a saving of

$0.18 per ton in cyanide, giving a net profit of about 15¢ per ton of ore, and this on an ore containing only 7 ounces of silver per ton.

Sodium Sulphide Precipitation.—Sodium sulphide has from time to time been suggested as a precipitant for silver and it should be a highly efficient reagent where silver alone is to be considered. The difficulty is that it rarely happens that the amount of gold in a silver ore is small enough to be negligible and as gold is not precipitated by sodium sulphide the method would have to be supplemented by some other process in order to recover the gold. The advantages of sodium sulphide for silver precipitation are its low cost and its faculty of regenerating the combined cyanide, thus, $2KAg(CN)_2 + Na_2S = Ag_2S + 2KCN + 2NaCN$. Ordinarily it would produce a very pure precipitate because copper is not thrown down and about the only metals likely to be precipitated with the silver would be lead and zinc if dissolved from the ore. Its drawbacks are that it does not recover any gold that may be present and that the resulting product is not silver but the sulphide of the metal, which is not so readily converted into bullion. The process is now in use at the Nipissing Mill with apparently satisfactory results having displaced aluminium dust when the price of the latter reagent became prohibitive under war conditions. The resulting silver sulphide is reduced to a metallic slime by the Denny desulphurizing process which incidentally regenerates most of the sodium sulphide used in precipitation.

Charcoal Precipitation.—Park[1] states that this method is in use in many plants in Victoria for precipitating gold from cyanide solutions the procedure being to pass the solution through barrels packed with finely broken charcoal, which is afterward burned, and the ash melted down with fluxes. He does not consider it suitable for large tonnages of solution.

H. R. Edmands in a paper read before the I. M. M. (London) March 7th 1918, describes a process of charcoal precipitation which shows promise of great usefulness in certain circumstances.

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[1] The Cyanide Process, Page 206 (Fifth Edition).

For this process the charcoal is ground to a fine powder. The author says,

> "Fine grinding is very beneficial; it must be carried beyond anything measurable by screen tests, but there would appear to be a point beyond which it is unadvisable to proceed. This point can be best determined by precipitation tests, or perhaps by noticing the color of an emulsion of a known weight of charcoal in distilled water.
>
> "In practice it is arrived at by the time allowed for crushing with a given weight of balls. Charcoal quenched in water, when red hot and then wet ground, is superior to the air-cooled dry crushed product. In most cases 2 to 3 hours suffice for precipitation, when the charcoal is finely ground and agitation given, but the bulk of the precipitation takes place much sooner. At the Yuanmi plant precipitation is effected by filtering through thin layers of charcoal and the time of contact must be measured in seconds.
>
> "Charcoal is a much less effective precipitant of silver and copper than of gold."

The apparatus used, known as the "Moore-Edmands Extractor" is a rectangular box with V-shaped bottom, in which are suspended filter leaves after the manner of the Butters filter except that there are horizontal propellors in the bottom for producing agitation. The method of procedure is as follows:

The charcoal after removal from the kiln is washed to free it from ash, and is charged moist to the tube-mill with about ⅓ its weight of water, where it is ground with small iron balls for about 18 hours. Measurement of charcoal in pulp is effected by sp. gr. determination. Solution is run into the box until the leaves are well covered and then charcoal pulp is pumped in and the propellors started. After a few minutes solution is run in and a cake of charcoal built up on the leaves. It takes about 20 minutes for the solution in the box to become clear and then pregnant solution is added and drawn through the cake for 6 hours. At the end of that time the charcoal cake is dropped and re-emulsified and the cakes re-formed.

Three precipitation tanks are employed and are run on the counter-current principle. A given charge of charcoal remains in its unit for 9 days. A new charge forms the 3d unit in the series for the first 3 days, it then becomes the 2d unit, and for the final 3 days is the 1st unit, the object being to enrich the unit as much as possible with pregnant solution before cleaning up, and to have charcoal that is more or less barren at the end of the series to strip the solution before going to the barren sump. The charcoal cake, after being three days in the 1st unit is discharged and burnt and the unit charged with fresh charcoal. The clean up is made by emulsifying the cake and pumping to a plate and frame press, where it is washed, the cake discharged from this being ready for burning. This is done in braziers in the bottom of which a few hot coals are laid and the damp cake shovelled in on top. It needs about 48 hours to consume all the carbon. The ash is then melted down in the usual way, the flux recommended by Edmands being 40–50 parts of borax, 50 of sand and 10 of salt for every 100 pounds of ash.

CHAPTER XI

CLEANING UP AND MELTING

With zinc shaving precipitation the usual method of clean-up is to shut off the flow of solution and starting at the head compartment of a box to wash the shavings gently in the solution avoiding any action that would tend to break up the attenuated threads into short pieces. The zinc is then removed and after draining may be placed in a tub of water to protect it from oxidation by the atmosphere. A plug or cock is then opened at the bottom of the compartment and the solution, sludge, and short zinc allowed to run out into a launder which conveys it to the clean-up tank. The interior of the compartment is then carefully hosed out and cleansed with brushes or sponges, the outlet closed, and the screen-bottomed tray replaced. The long zinc is removed from the tub of water and carefully packed in, and the next compartment proceeded with in a similar manner, the shortage of zinc in the first being supplied by that removed from the second, and so on down the line. As soon as each compartment is cleaned and repacked the solution flow is opened and allowed to run long enough to cover the zinc and protect it from the air.

In some plants it is customary to replace the solution in the boxes with water before beginning the clean-up so as to avoid the troublesome effect of the cyanide upon the hands and arms of the workmen; when this is not done it is usual to supply rubber gloves for their use.

The clean-up tank may be six feet in diameter and four feet deep with arrangements for keeping the charge in agitation by means of stirring arms. At the top, on one side, is placed a large screen of from 20 to 60 mesh, which may be supported on springs to assist in working the sludge through it. The latter after leaving the precipitation box runs to this screen where the coarse zinc is removed while the precious metal mud together with some fine zinc falls through into the tank, the operation being assisted by a stream of water or solution under pressure from a hose. For a large plant the screening of the zinc shorts is more conveniently done by running the contents of the zinc boxes to a revolving trommel provided with the desired mesh of wire cloth and a spray of clean water to wash the short zinc free from precipitate. The suction pipe of a small plunger pump is connected with the bottom of the tank and the product delivered into a filter press of the ordinary metallurgical type.

The broken zinc which is too big to pass the screen is usually returned to the zinc room to be utilized in one or other of the ways already described, but some metallurgists prefer to get rid of it either by dissolving it in acid, leaving a rich residue to be washed and filtered or by melting down in crucibles with a special flux. In some plants it is the custom to treat the whole of the precipitate with sulphuric acid before sending it to the filter press and in that case some or all of the "zinc shorts" may be allowed to enter the tank and share the acid treatment.

If the clean-up tank is to be used for acid treatment, it should be fitted with an adjustable hood and chimney to carry off the gases which are often poisonous and also with a decanter for removing the supernatant liquor after settlement.

When the precipitate is to be acid-treated it is advisable to displace the cyanide solution in the zinc boxes with water before beginning the clean-up so as to avoid the formation of hydrocyanic acid on the addition of the sulphuric acid. Water should be present to the amount of about 10 parts by weight to one of acid necessary, and the latter is added to the charge little by little, care being taken to see that the action is not sufficiently violent to cause it to boil over; the agitator meanwhile keeping the contents of the tank well stirred.

The acid sulphate of soda, $NaHSO_4$, is sometimes used as a cheap substitute for sulphuric acid.

$$2NaHSO_4 + Zn = Na_2SO_4 + ZnSO_4 + H_2.$$

When all action has ceased more water may be run in to fill the tank, and the charge may be allowed to settle. The clear liquor is decanted and more water added, agitated, settled and decanted as before. The thickened sludge is then pumped into the press and when the latter is filled the cakes are further washed by forcing water through them from cloth to cloth by means of the special washing ports. The Rand practice is to use hot water for this purpose for the more thorough removal of acid and sulphates. It is important that these be washed out as far as possible so as to minimize the formation of matte in the subsequent melting operations.

Whether acid treatment be used or not it is advisable to dry the cake down to about 10 to 15 per cent. moisture before fluxing and this may be done by spreading it on trays which are placed in a drying oven heated by steam or hot air. A small amount of moisture left in the cake tends to prevent loss by dusting when disintegrating and mixing with the flux. Many important plants do not dry the precipitate before fluxing except in so far as to allow compressed air to pass through the cake in the filter press for an hour or two before opening and discharging.

The early practice was to calcine the precipitate in a reverberatory or muffle furnace before melting so as to oxidize the zinc and cause it to enter the slag, but this procedure has now been almost entirely superseded by the acid process. In the treatment of silver and silver-gold ores the precipitate is usually of so high a grade that neither calcining nor acid treatment is necessary and direct melting will produce bullion from 850 to 950 fine.

Melting is usually done in graphite crucibles, whose size varies in different mills, ranging from No. 50 up to No. 400. The firing is done with coke or fuel oil; charcoal is sometimes used but has no sustaining power and requires continual replenishing, resulting in violent fluctuations of temperature. This effect may be remedied to some extent by using a mixture of coke and charcoal, and expenses may often be reduced in this way. In South Africa a reverberatory furnace capable of containing a dozen crucibles is sometimes used. In recent times the oil fired tilting furnace to hold one large crucible of No. 300 or 400 size has found considerable favor.

At the Nipissing Mill, the bullion from all sources is melted down in an oil-fired reverberatory. After the slag has been removed the metal is refined by playing a blast of air over the surface of the molten bath of metal by which means silver of 998 to 999 fineness is obtained.

The Nipissing type of furnace has recently found favor in Mexico and in many large plants has entirely displaced crucible melting, with a considerable saving on account of crucibles, fuel and labor. The precipitate cakes from the presses without any preliminary drying are placed on trays and covered with about 5% of flux, usually composed of borax and bottle glass and charged into the furnace at intervals until the desired amount of silver has been accumulated. The Nipissing refining procedure is however, omitted, and as soon as the charge is fused, it is tapped out into shipping-bar moulds, the slag following at the end of the pour. About 70 shipping bars, each 35 kilos in weight, are produced at each fusion. When melting about 8000 kilos of fine bullion a month, the furnace bottom will last for 6 to 8 months before it has to be replaced.

Fluxing the Charge.—The flux ordinarily used will vary with the composition of the precipitate and in order to determine the best mixture a number of small fusions may be made with varying proportions and the results noted. Borax is the most generally useful reagent but if used in large quantity, it makes the fusion rather expensive. Its place may be taken to some extent by carbonate of soda and sand, utilizing the tendency of bases to form fusible double silicates. For this purpose only sufficient soda should be used to form the second base in the double silicate. When the solutions are not properly filtered before precipitation there may be a considerable amount of silica in the precipitate and it may not be necessary to add any sand, but such a state of things usually implies bad working conditions. A little fluor-spar is sometimes used to flux difficultly fusible substances. For soda most text books recommend the bi-carbonate though for what reason it is difficult to see, and the writer prefers the normal carbonate in the form of commercial soda ash. Sodium bi-carbonate evolves a large volume of carbonic acid gas at a temperature below that of sintering of the charge, and if the precipitate is too dry, a considerable loss by dusting may occur; moreover, soda ash is less bulky and also cheaper per unit of fluxing power.

Julian and Smart[1] give the following as a typical flux mixture.

Precipitate	100 parts
Bi-carbonate of soda	40 parts
Borax	40 parts
Sand	15 parts

Park[2] gives,

TABLE 13

	Clean precipitate, little sand	Much zinc and little sand	Very sandy
Precipitate	100	100	100
Bi-carbonate of soda	6	20	50
Borax	50	50	30
Sand	3	15	
Fluor-spar	...	..	2

If sodium carbonate had been used instead of the bi-carbonate only about ⅔ of the above amounts of soda would have been needed.

By adding an oxidizing agent to the flux it is often possible to raise the fineness of the bullion by oxidizing some of the base metal and throwing it into the slag. Nitre is sometimes used for this purpose but it has a low efficiency owing to the rapidity of its reaction and E. H. Johnson and W. A. Caldecott[3] have sub-

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[1] Cyaniding Gold and Silver Ores, page 168 (Second Edition).

[2] The Cyanide Process, page 188 (Fifth Edition).

[3] *Proceedings* Chem. Metall. and Min. Soc. of S. A., Vol. III, pages 46–63 (July, 1902).

stituted manganese dioxide, and suggest the following mixtures:

Precipitate	100 parts
Fused borax	20 to 35 parts
Manganese dioxide	20 to 40 parts
Sand	15 to 40 parts

They state that manganese dioxide has a tendency to carry silver into the slag, so if much silver is present, it should be used cautiously or not at all.

When an oxidizing flux is used the operation should be carried out in crucibles with removable or permanent fire-clay linings, as graphite would tend to counteract the effect of the oxidizing agent.

To one accustomed to the high-grade precipitates produced in the silver mills of Mexico the proportions of flux previously mentioned will seem excessive, and in such cases the quantity may be considerably reduced. A typical flux mixture for these rich silver precipitates is that used by Walter Neal[1] at the Dos Estrellas mine in Mexico.

Precipitate	100 parts
Borax	15 parts
Bi-carbonate of soda	8 parts
Sand	4 parts
Scrap wrought iron in excess	

The iron is used to remove the silver from the matte which is so often produced in melting these precipitates even when no acid treatment is given. The same writer, in cleaning-up, makes two short-zinc products. He first screens his material with a 20-mesh sieve and afterward with a 60-mesh. The zinc remaining on 20 he returns to the boxes and that passing 20 but remaining on 60 he melts separately with a special flux:

Short zinc	100 parts
Borax	40 parts
Soda	20 parts
Sand	10 parts
Lime	5 parts

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[1] *Mining and Scientific Press*, February 29, 1909.

The resulting metal, which contains 20% of zinc, is added to the high-grade bullion when the latter is remelted for running into bars.

A flux the writer has often found useful for clean silver precipitates consists of

Precipitate	100 parts
Borax	15 parts
Sand	8 parts
Soda (Na_2CO_3)	8 parts

Preparation of The Charge.—Opinions vary as to the degree of mixture necessary. Some metallurgists break up the lumps of precipitate and pass the whole through a fine sieve, varying from ¼ in. to 20 mesh, before or during the addition of the fluxes, making a thorough mixture before charging into the crucibles, while others, avoiding breaking the lumps as far as possible, spread the precipitate in a layer on the fluxing table, sprinkle the fluxes evenly on top and shovel at once into the pot. The latter method has the advantage of reducing the dust loss to a minimum, but on the other hand is probably more suitable for a pure high-grade precipitate than for a base and dirty one, and the choice of procedure must be determined by the circumstances of each mill.

Briquetting has much to recommend it in the avoidance of accidental spilling of product outside the edges of the pot and minimizing of dust losses in charging, especially where it is the custom to add fresh batches of the material on top of a charge already partly fused, but it is expensive and also conducive to losses due to the additional handling necessary.

Charging.—The practice in some mills is to charge a crucible only at the beginning of each fusion and to pour it as soon as ebullition ceases. It is a wise precaution against dust loss but is less efficient in fuel, time and labor, and its adoption should be determined by the value of the precipitate per unit of weight. To avoid spilling when charging, the fluxed material is sometimes put into stout paper bags holding from one to two pounds, which can be placed in the crucible with a pair of tongs.

Pouring.—The most usual procedure is to pour the fused charge, slag and all, into a conical mould, and when cool to knock off the bottom and put aside for remelting into a shipping bar. At the Butters Divisadero Company's mill, however, G. H. Clevenger some years ago introduced the practice of carefully tilting the crucible to decant the bulk of the slag into a conical mould, stopping when about an inch only of slag remained on top of the metal: a little dry sand was then thrown in to cool and thicken the slag, which was raked off with a small scraper, the pot being kept tilted so as to bring the level of the molten metal almost to the lip. The clean bullion was poured direct into the shipping-bar mould and a remelting avoided, resulting in a marked saving in fuel and labor. The first batch of slag removed was low grade and free from metallics, but the thickened slag had to be remelted and the thickening agent, being clean quartz, instead of bone ash, was easily fluxed with a little soda. The resulting bullion averaged from 900 to 950 fine in gold and silver.

Sampling.—There are various methods in use for sampling the bullion bars for assay. Probably the most common is that of chipping off two diagonally opposite corners of the bar and averaging the resulting assays. Sometimes, however, the difference between them is so great as to throw doubts on the accuracy of both and in that case it will often be possible to obtain a better sample by stirring the molten metal and dipping out a little in a small well-heated ladle just before pouring. This sample may be granulated by pouring it while molten onto a sloping board immersed in a bucket of water, or better still, by pouring into a conical assay mould and when cold drilling a hole right through the centre of the button and assaying the borings.

The by-products of the melt consist of slag, sometimes of matte, and when coke or charcoal is used for firing, of ashes.

When the slag contains visible metallics it is usually ground up and concentrated either in a sluice box, a cradle, or on a Wilfley table and the tailings shipped to a smelter or thrown into the mill bins. The latter course has in some cases been known to complicate the cyanide treatment of the ore and should be adopted with caution.

The matte may be either shipped to a smelter or treated at the mill. There are various ways of accomplishing the latter. One is to melt the matte in a crucible with metallic iron, forming iron sulphide and throwing down the precious metal. Another is to blow air through the molten mass. A. E. Drucker[1] recommends the following: the matte is broken fine in a rock breaker and alternate layers of borax, matte, and broken cyanide are charged into a No. 60 crucible until it is nearly full, a layer of borax being added as cover. The charge is fused down in the furnace at a white heat until tranquil. The slag will be thick, and is removed with a skimmer, the resulting metal being then poured into a mould, and the slag remelted with a subsequent charge. R. C. Kline, in a private communication to the writer, states that in some cases if the matte be ground to pass 200 mesh and treated with strong cyanide solution and a lead compound, a recovery of 99 to 100% of the assay content may be made.

Furnace ashes nearly always contain some precious metal and should be roughly crushed and run over a sluice box or otherwise concentrated. They should never be put into the mill to be treated with the ore on account of the presence of unconsumed carbon, which would tend to absorb gold from the cyanide solution.

The Tavener Process.—On the Rand, P. S. Tavener[2] introduced the process of melting the precipitate in a reverberatory furnace with litharge, thus forming a lead bullion which was afterward cupelled in the usual way. He deals with the zinc shorts by the same method but recommends that the two products be partially dried and separately fluxed. For the precipitate he uses the following mixtures:

Product....................	100 parts by weight
PbO......................	40 to 60 parts by weight
Assay slag..................	10 to 15 parts by weight
Slag from previous melts......	10 to 15 parts by weight
Sawdust..................	½ to 1½ parts by weight

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[1] *Mining and Scientific Press*, May 18th, 1907.

[2] *Proceedings* Chem., Metall. and Min. Soc. of S. A., pages 112–159 (Oct., 1902).

These ingredients are mixed and rubbed through a sieve and then shovelled into the reverberatory before starting the fire. Regarding the amount of litharge used, Tavener recommends that it should be adjusted to produce a lead bullion containing not over 8 to 10 per cent. of precious metal. The short zinc is fluxed differently.

Zinc	100 parts
PbO	100 to 150 parts
Mixed slag	25 to 30 parts

No sawdust is added here. This mixture is spread over the charge already in the furnace, and on that a sprinkling of litharge and over all a thin layer of easily fusible slag. The fire is then lighted and the temperature gradually increased and maintained at a high point until fusion is complete. The charge is then stirred well and sawdust added to reduce any litharge that may still remain in the slag. The liquid slag is tapped or rabbled off to within an inch of the metal surface and the remainder thickened and carefully skimmed. The clean lead is tapped off into bar moulds of a suitable size for feeding into the cupel furnace, which is of the usual type.

In the case of silver it is usual, after cupellation is finished and the molten surface bright and clean, to ladle the bullion out into moulds but in dealing with gold where a depth of only ½ to one inch of metal remains in the cupel after the operation is complete, it may be cooled just to the point of solidification; then an iron bar gently placed under the edge of the sheet of metal and pried upward will cause it to break into two or more pieces which may be removed with tongs: if these pieces be at once tapped with a hammer before cooling further they will break up into small pieces of any desired size for charging the crucible in which the shipping bars are produced.

At the Homestake[1] the precipitate after acid treatment is mixed with litharge, borax, and assay slag, and briquetted. The

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[1] The Metallurgy of the Homestake Ore. Clark and Sharwood. *Proceedings* I. M. M. (London), 1913.

briquettes are dried in a furnace to a point where they can be charged on a bath of molten lead without spitting. In this case the melting and cupellation are done in the same furnace, which is an English-type cupel furnace with oval test, made of a mixture of cement and limestone in the proportion of 75% to 25%. After the fire is started lead is fed into the furnace and when a bath of well-heated metal has been obtained the briquettes are fed in on top, more being added as they melt down, until the test is full of fused material. The slag is tapped off by cutting down the front wall of the test, the aperture closed, and more briquettes charged in, the operation being repeated until all the precipitate has been disposed of. Finally more lead is added to raise the level of the bath and the slag and matte remaining finally cleaned off. Cupellation is then proceeded with until all the lead has been eliminated and only fine bullion remains.

CHAPTER XII

LABORATORY EXPERIMENTS

Small scale experiments are very valuable if carefully and skilfully performed, but may be misleading and inconclusive if the proper precautions are not observed.

In all such experiments the operator should ask himself whether the conditions of his test differ in any vital particular from those that would obtain on a commercial scale; if so, there is no guarantee that his results would be borne out in a working plant.

Examination for Metallic Gold and Silver.—It is advisable first of all to examine the ore for visible gold and silver, the presence of which might influence the choice of a method of treatment. If no gold or native silver can be seen in the rock, take a representative sample of about 300 grams and grind it up to pass a No. 30 sieve. Weigh out 250 grams and carefully pan down until most of the gangue is removed. Examine the heavy residuum for metallics. Should any be found it will be well to see how much can be amalgamated.

Amalgamation Test.—This may be made by putting about an ounce of gold-and-silver-free mercury in a small stoppered bottle and adding the heavy material from the pan with a little water, and agitating violently for 5 or 10 minutes. The mercury is then panned free from mineral, and the latter examined again with a pocket lens. The precious metal content of the mercury may be determined either by volatilization in the muffle or by dissolution in nitric acid.

If it is desired to estimate the whole amount of gold and metallic silver amalgamable in the ore, a similar experiment may be made by weighing up 250 grams of the pulp, diluting with water at 1:1, and agitating violently with mercury for about 30 minutes. The mercury is then panned from the pulp, and the latter assayed. The mercury may also be retorted, and the contents weighed, or, in the case of gold alone, the mercury may be dissolved in nitric acid. This test is likely to flour a large part of the mercury, but after being panned clean it may be collected by the addition of a little sodium amalgam.

For these amalgamation tests it is usually better to render the water in the pulp strongly alkaline with caustic soda, and to see that it remains alkaline throughout the experiment, but a test should also be made with plain water. It is also a useful precaution to weigh the mercury before and after the test.

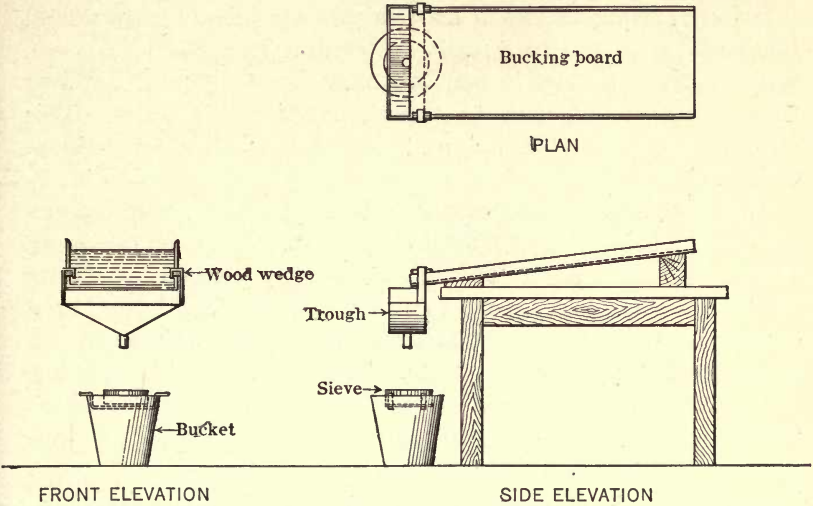
Visible gold and silver are likely to give trouble in cyaniding unless ground exceedingly fine so as to accelerate the dissolving action, and when either of them is found it will be a question to be determined whether it would be better to amalgamate before cyaniding or grind the whole ore so fine that the metallics will be dissolved in the cyanide solution in a reasonable time.

If the ground ore be found to contain metallics an amalgamation test should be made on the ore prior to cyanidation for which purpose a kilo of the ground ore may be made into a pulp with water and passed over a small amalgamated copper plate several times.

The ore should also be tried by cyanidation under various conditions without previous amalgamation.

Wet Grinding of Samples.—In all experimental work a rule should be made never to dry prior to cyanidation a pulp that has once had water added to it. If, after amalgamating or concentrating a sample, it is desired to grind it finer, this should be done wet, with a pestle and mortar or some similar contrivance, and the screening should be done in water or solution.

The regrinding may be done satisfactorily in a small tube mill if certain precautions are observed. The usual tendency in this method is to overgrind the sample; that is, if it is desired to grind to -100 mesh and the sample is kept in the tube mill until all will pass the said screen the resulting product will be for practical purposes a -200 pulp. If a 200-mesh grinding is aimed at the resulting pulp will be far finer than would be produced on the milling scale with the same end in view, and the result is that the extraction results may be quite misleading. To avoid this the product to be re-ground should be first wet screened on the desired sieve and the oversize only placed in the mill. This oversize is then allowed to grind for 5 to 15 minutes and is then removed and screened, the oversize being replaced in the mill. This process is repeated until all will pass the screen. A good scheme for regrinding wet sand is to take an ordinary bucking table and raise the back end a few inches with some wooden blocks and attach to the front a narrow wooden trough of hopper shape with a ½-inch nipple at the apex. This may be clamped on or wedged onto the edge of the bucking table so as to be easily and quickly removed. The sample is then bucked down in the usual way using just enough water to form a paste that will not run off by gravity. Just as in dry bucking the pulp is periodically screened by washing it down with a wash bottle and a piece of rubber belting onto a screen of the desired size suspended over a bucket of water.



FRONT ELEVATION SIDE ELEVATION

FIG. 29.—Showing Method of Bucking and Screening Wet Sand.

For purposes of weighing a given quantity for a test the specific gravity of the pulp may be determined or a portion of it dried to ascertain the percentage of moisture, and then sufficient of the wet pulp taken to give the required weight of dry ore, due allowance for contained moisture being made when adding water or solution to produce the dilution needed for treatment.

The reason for avoiding the drying of a sample is that this procedure, even when artificial heat is not applied, tends to oxidize its less stable constituents and produce changes which may materially affect the results of subsequent cyanidation.

Before starting a series of experiments considerable care should be expended in getting a really representative sample of the ore, mixing, grinding coarsely (say to No. 20 sieve), and cutting down to an amount sufficient to serve for the whole series. This should be again well mixed, and stowed away in a covered can or box.

Agitation Tests.—Laboratory tests are usually made by mechanically shaking up in a bottle for a given time a charge of ore and cyanide solution. The most generally convenient device for this purpose is a wheel, to which are attached boxes, each capable of containing a standard acid bottle, and with means for securing the bottle firmly in place. The wheel is rotated at about 30 r.p.m. by a belt driven off a line of shafting. Objections have been raised to this device on the ground that agitation in a stoppered bottle does not allow of a proper aeration of the charge. Such objections, while plausible, are not borne out by facts, actual experience showing that the aeration obtained in this form of agitation is amply sufficient for almost any ore. It is of course necessary to leave an adequate air volume in the bottle, and for this reason the charge should not more than ⅓ fill it.

A useful form of agitating wheel is shown in Fig. 30.

A form of bottle agitator devised by G. H. Clevenger is useful in some cases. It was designed to meet conditions where the material to be cyanided displays exceptional reducing tendencies and absorbs an abnormal quantity of oxygen, though the writer has found that in some cases it seems to give a less efficient aeration of the pulp than the usual agitation as already described.

It consists of two or more horizontal rollers 6 or 8 inches in diameter and of any desired length, placed parallel to one another with 2 or 3 inches clearance between their peripheries and belted to a line of shafting. The charged bottles are laid, unstoppered, horizontally on the space between two rollers, and thus rotated for any desired period.

When this device is used in warm dry climates the evaporation is so considerable during a two or three day agitation that calculations as to cyanide strength and consumption are rendered quite unreliable, and to correct this it is necessary to weigh the bottle at intervals (and especially at the end of treatment), and add sufficient water each time to compensate for the evaporation loss, the original weight of bottle and charge being of course known.

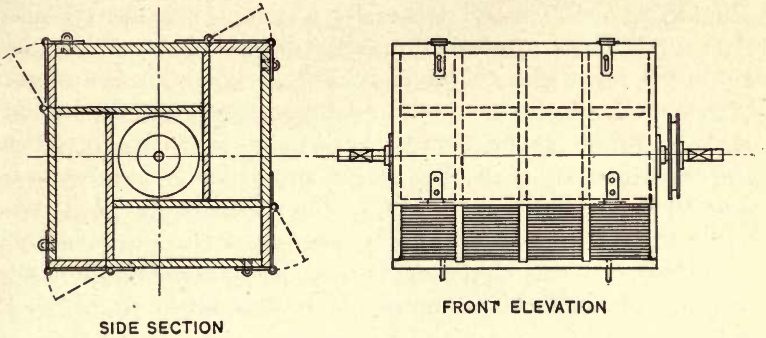


Fig.. 30.—Bottle Agitating Device.

Laboratory Pachucas.—As a rule these miniature Pachucas are not satisfactory as it is difficult to proportion the air to the volume of pulp so as to correspond with milling conditions. In addition to this a small amount of material especially the heavier and more mineralized particles is liable to accumulate in the dead zone at the apex of the cone and escape proper treatment, and when the contents of the tank are dumped for sampling this partially treated material may seriously affect the assay.

Alkali Consumption.—Before making up a bottle charge on an unknown ore it is necessary to ascertain the probable alkali consumption. For this purpose weigh up 4 lots of pulp of about 200 grams each, and add water at a ratio of 3:1, numbering the bottles 1, 2, 3, and 4. Then add slaked lime powder at the rate of 5 lb. per ton of ore to No. 1, 10 lb. to No. 2, 15 lb. to No 3, and 20 lb. to No 4, and agitate for from 6 to 12 hours. At the end of the period filter off some of the liquor from each and test for alkalinity. The quantity found in solution will furnish an estimate of how much lime will be needed for making up the cyanide extraction test. Generally, a residual strength of 0.05% CaO will indicate a suitable quantity of lime for an all-gold ore, and 0.1% for a silver or silver-gold ore. To obtain the amount of lime needed in terms of pure calcium oxide (CaO) the strength of the lime used for the test may be ascertained closely by putting one gram in a litre of distilled water, and agitating in a bottle for 8 or 10 hours. The water is then filtered and titrated for alkalinity and the amount of CaO dissolved from 1 gram of lime calculated in terms of percentage. A more accurate method of determining available alkalinity of lime is given in Chap. II, p. 29.

Test for Necessary Fineness of Grinding.—This is most conveniently made by the agitation method, and it will be necessary to make up 3 or 4 bottle charges in order to have enough ore for the subsequent screen analysis.

Take an average sample of the ore and grind on the bucking table to pass No. 30 sieve. Weigh up 4 lots of 300 grams each and place in 4 standard acid bottles, add to each the quantity of lime estimated from the alkali consumption test, and water in the ratio of 3:1. For addition of cyanide it is best to make up a concentrated cyanide solution and ascertain the strength by titration. The number of cc equivalent to a given weight of cyanide may then be run in to the charge of ore and water in the bottle from a burrette. The volume of liquor to be so added may be allowed for when diluting the pulp, so as to give the exact ratio of solution to ore originally decided on. It is usually advisable to shake up the bottle containing the ore water and lime before adding the cyanide.

Add sufficient cyanide to make a solution strength of 0.3% KCN, stopper the bottles securely, and place in the agitating wheel. For the purpose of this test it is well to give an agitation period of 6 or 7 days. Treatment of sand by agitation will usually give the same result as by leaching, only in less than half the time. For the first 3 days of agitation (and longer if the cyanide consumption continues to any considerable extent), open the bottle each day and test 10 cc of the solution for cyanide and alkali. If necessary add lime and cyanide sufficient to maintain alkalinity between 0.05 and 0.1, and cyanide at 0.3% KCN.

Some metallurgists prefer to add a given amount of cyanide and let the experiment run the full time of treatment without testing and maintaining the cyanide strength. This does not seem to have anything to recommend it, and is at variance with normal working conditions, because it is usual in practice to maintain the cyanide strength at a definite figure for all or the greater part of the time of treatment. Moreover, it may often happen in making a bottle test on an unknown ore that the whole of the cyanide added at the start may be consumed long before the time of treatment is up, and the fact is not known till the end, resulting in a useless experiment and wasted time.

The object of this test is to obtain the maximum dissolution of precious metals from each grade of material, leaving out of sight for the moment commercially economical conditions.

At the end of the period the solution is titrated and consumption of cyanide calculated, but the result thus shown should not be used for basing working figures upon. The charge is then washed, either on a laboratory filter or by decantation. In the latter case the four charges may be combined in an enamelled bucket, and settled, decanted, and washed, as one sample. The whole is afterward dried, and 1000 grams weighed up for screen analysis, while a sample of the remainder is assayed as a check.

A convenient screen series for the sizing will be Nos. 40, 60, 80, 100, 150, and 200, or if the Tyler Standard screens be used, 35, 48, 65, 80, 100, 150, and 200. If desired the −200 product may be further separated into granular and colloidal matter by panning, though such a division made by this means is arbitrary, and results obtained by different operators are not comparable.

For the screen analysis it is often more satisfactory first to separate out the colloidal matter by panning and then dry the granular part before screen sizing. Another plan is to remove the colloids and the finest of the sand by washing them into a bucket through a No. 200 sieve under the water tap. The two products are then dried separately, and the oversize graded in the usual way. In this case there will probably be some −200 sand left in the original oversize which may be screened out dry and added to that which was washed through the sieve with water.

Some metallurgists prefer to do the whole of the screen-sizing in water, and this has the advantage of avoiding any possible error due to decrepitation of the larger particles in drying, but such a risk is small when the drying is conducted at a low temperature; moreover, water screening takes longer to attain the ultimate limit of separation between each grade, and there is a danger of stopping the operation before it is complete.

In dry screening the common practice of putting a washer on the screen or rubbing the pulp on it with a block of wood is not to be recommended, because it forces through the meshes particles which would not otherwise pass, giving a higher reading for the minus product than the correct one, and incidentally it displaces the wires making the mesh irregular and soon impairing the standard quality of the screens besides materially shortening their life. The main incentive to this practice is the difficulty often experienced through blinding of the meshes by the clayey matter in the ore. For this reason it is recommended that the colloids be first removed by water washing, but if it is found necessary or advisable to dry-screen a sample entire, the meshes may be kept free from the fine slime by continually tapping sharply on the top edge of the sieve with the handle of a spatula or other similar instrument, and in this way good time may be made even on very colloidal material.

In the case of ground pulp that has been wet and subsequently dried, however, this method is not satisfactory as the colloidal part in drying forms hard pellets or granules which remain on the screens, and are recorded as sand. In such cases in order to obtain reliable results the dry pulp should be thoroughly disintegrated in water and the colloids floated off before the screen test is made. If a known quantity of the dry pulp be first weighed up the sample after disintegration in water may be washed on the 200-mesh screen under the water tap, the part that passes being allowed to run to waste, and the amount calculated by difference. In this case, as already stated, the portion remaining on 200 should after drying be re-screened on a 200 sieve as it will be found possible to make a further elimination of −200-mesh material in this way.

A list of screens and their apertures comparing the Tyler Standard, and the Institution of Mining and Metallurgy (London) standard will be found among the tables at the end of this book.

The assays of the various sizings will give a good indication of the degree to which comminution should be carried in order to liberate the precious metals, and expose them to the action of the cyanide. It is well to check the result by multiplying each assay by its proportional percentage, and dividing the sum of all the products by 100. The result should check closely the assay of the sample taken out before screen sizing.

The test may afterward be repeated using 0.1% cyanide strength, and only 2 or 3 days agitation, in order to get an idea as to the maximum size of particle that will give a good extraction under the usual conditions of slime treatment.

It is often instructive to make a screen analysis of the pulp before cyaniding, so as to see how the assay values are distributed, and what percentage is extracted from each grade.

"All-sliming" Tests.—Having obtained an idea of the effect of cyanide treatment on each grade of material, an "all-sliming" test may be made, with the object of finding out the probable maximum extraction by simple cyanidation.

The ore is ground so that the whole will pass a No. 200 sieve, and 250 grams weighed up for a charge. Lime is added in accordance with the experience gained in the previous test, and the dilution made up to 3:1 with water to which the necessary amount of cyanide has been added to give a strength of, say, 0.3% KCN. The bottle is placed in the agitating wheel and treated for 3 or 4 days, the solution being titrated each day for cyanide and alkali, and the strength maintained at the original figure.

Grinding Samples to a Given Screen Size.—It is impossible in the laboratory to imitate exactly the degree of grinding in a working scale mill by using a certain screen size as a standard. For instance, if it is decided that a −100-mesh product is desired an operator may produce several entirely different degrees of grinding though using the same mesh screen, according to the frequency of the screening. If screening is done after long intervals of grinding, on a bucking table or with closely set discs if using a disc grinder, the resulting product though nominally −100 mesh will have a far finer character as a whole than if the screening be frequent during bucking or if the discs of the grinder be set apart and gradually closed up after each screening. And probably none of the products obtained by these different procedures will correspond to the pulp of a mill where −100-mesh grinding is aimed at. In one case known to the writer the product obtained from a bucking table, putting everything through a 100-mesh sieve, only contained 50% of −200 material whereas the finished mill pulp on the same ore contained 5% of +100-mesh material and 75% of −200. It is thus apparent that deductions drawn from laboratory extraction results on pulps all of which pass a given screen size must be received with discrimination.

Computing Cyanide Consumption.—This may be done as in the following example: Ore taken, 250 grams. Ratio of solution to ore, 3:1 = 750 cc: 250 grams. Cyanide strength, 0.3% KCN.

Cyanide (KCN) added		Cyanide (KCN) recovered			
Making up....	2.25 grams	Titrated,	25 cc	at 0.10%..	0.025 gram
1st day.......	1.45 gram	Titrated,	25 cc	at 0.23%..	0.057 gram
2d day.......	0.49 gram	Titrated,	25 cc	at 0.28%..	0.070 gram
	4.19 grams	Remaining at finish	675 cc	at 0.27%..	1.822 grams
			750 cc		1.974 grams

4.19 grams
1.974 grams

Loss, 2.216 gram

$$\text{KCN consumption per ton} = \frac{2000 \times 2.216}{250} = 17.7 \text{ lb.}$$

Or when working on the metric system,

Ore taken 200 grams,
ratio of solution to ore 3 : 1 = 600 cc : 200 grams,
cyanide strength 0.3% KCN

Cyanide (KCN) added		Cyanide (KCN) recovered			
Making up....	1.8 grams	Titrated,	25 cc	at 0.10%..	0.025 gram
1st day.......	1. 15 gram	Titrated,	25 cc	at 0.23%..	0.057 gram
2d day.......	0.385 gram	Titrated,	25 cc	at 0.28%..	0.070 gram
	3.335 grams	Remaining at finish	525 cc	at 0.27%..	1.417 grams
			600 cc		1.569 grams

3.335 grams
1.569 grams

Loss, 1.766 gram

$$\text{KCN consumption per ton} = \frac{1000 \times 1.766}{200} = 8.83 \text{ kilos.}$$

It should be noted that cyanide consumption as figured above represents chemical consumption only. In practice there will also be a mechanical loss of cyanide in residues, and also a loss to some extent in precipitation, if the zinc process be used. The mechanical loss in residues will depend on the strength of solution used, and also on whether the slime residue is dewatered by a filter or by decantation.

In the case of silver ores the chemical consumption indicated in a bottle test usually corresponds very closely with the actual total consumption to be expected in practice, and for this reason; an important factor in the cyanide consumption is the amount that combines with the silver: on precipitation this silver is replaced by zinc, and this zinc solution on coming in contact with the new lime added in the mill is partly dissociated and some free or available cyanide liberated. The amount of such cyanide thus recovered (a phenomenon which is not apparent in the ordinary laboratory test) comes near to offsetting the mechanical loss of cyanide in the filter residues.

Tests made as described will be found in most cases to approach very closely to working conditions, and to give results concordant with large-scale figures. A variety of combinations will suggest themselves to the operator. Assuming the ore to be "all-slimed" he will need to ascertain, among other things, the best ratio of solution to ore, the best and most economical cyanide strength, and the time of treatment necessary; and a series of bottle charges may be made up and carried through to elucidate each point separately.

The figure for the extraction obtained on such tests is usually based on the difference between the assays of the head and the residue, and this seems to be a perfectly proper procedure, since both head and residue sampling is completely under control. Some experimenters make a point of checking the extraction, calculated as above, by an assay of the solution, but on these small-scale bottle tests the fact of a portion of the solution being withdrawn each day for titration complicates the operation and renders the result less reliable than the figures based on the ore and residue assays.

Leaching Tests.—If the preliminary test seemed to indicate that excessively fine grinding was not necessary, it is probable that it will be more profitable to separate the pulp into sand and slime, percolating the former and agitating the latter.

As already stated, a good idea of the extraction to be expected by leaching may be obtained by agitating the sample in a bottle.

Actual leaching experiments on a laboratory scale, with charges of one or two kilos, may be made to confirm this fairly satisfactorily if proper precautions are taken. The usual method is to use a glass percolator or an inverted acid bottle whose bottom has been cut off, placing a filtering medium in the small end, and fitting a cork in the neck, through which passes a piece of glass tubing tipped with rubber tube and a screw clip. About 3 kilos of the ore, ground to No. 30 sieve are taken and the slime and −200 sand washed out in water through a 200 sieve, or the slime alone may be removed by panning. (This water may be eventually used for making up the cyanide solution.) The −200 material or slime is settled with lime water and decanted, and the +200 material or sand is drained as free from water as possible by continued tapping in a gold pan and pouring off the supernatant liquor. In this way sufficient moisture may be removed from the sand to permit of its being well mixed by rubbing with the hands. A sample for assay is taken of both sand and slime. Of the thickened slime one or more bottle charges are made up for an agitation test, and the sand, or an approximately known weight of it, is mixed with a suitable amount of fine lime, and charged into the percolator, each layer being firmly pressed down with the foot of a measuring cylinder or other tamping device.

A certain amount of cyanide solution is poured on each day and allowed to percolate slowly through, the daily proportion being regulated by the number of days treatment and ratio of solution decided on. Such ratio may be from ¾ to 1½ times the weight of ore taken for percolation.

If there is not too much fine sand in the pulp to interfere with percolation the separation into sand and slime may be made by merely washing in the gold-pan, though in this method most of the fine sand will remain in the portion to be percolated while in practice a large part of it would pass over into the agitation product.

In the case of readily oxidized ores it is sometimes more convenient to avoid handling wet products before treatment, and separation may be made by dry-screening the sample with a 200 or 150 sieve.

One of the principal difficulties in such percolation tests lies in the fact that the charge is so much shallower than a working charge that there is not sufficient head to overcome the capillarity of the interstices, so that, even with sand coarse enough to percolate very rapidly, the level of solution will not fall much below the level of the sand. The result of this is that the charge is not aerated (as it is in practice), by the air following the solution down into the interstices, between each wash. This may be overcome by applying a vacuum under the filtering medium after the solution has ceased to percolate by gravity. In this way the residual solution is drawn off and the air follows it down. The charge should then stand for several hours before the next wash is applied. This procedure is more important than it may seem, since a difference in extraction of 20 per cent. to 30 per cent. has been in some instances observed according to whether the vacuum was applied or not.

Another difficulty with small-scale leaching tests lies in the evaporation of solution, which is excessive, and out of all proportion to the size of the charge, rendering determination of the cyanide consumption quite unreliable unless careful measurement be made of the solution coming off, and the deficit made good by addition of water. Even so, the figure obtained is to be received with caution, as it is usually considerably in excess of working-scale results.

The most reliable way to conduct a leaching experiment is to use a vertical piece of iron pipe the depth of the tank proposed to be used in the plant, say, 6 ft. high, fitted at the bottom with a sleeve and plug, on top of which is laid a filtering medium. The plug is bored in the centre to admit a small iron cock. The diameter of the pipe should be at least 6 inches, because if too small the sand will not pack to a normal density, and will also be difficult to discharge when finished. Even with a 6-in. pipe it is often advisable, if charging with damp sand, to pack each layer gently down with a broom handle so as to approximate to the density found in a large working tank. Such a leaching test will need from 50 to 100 lb. of sand for each charge, and of course will necessitate provision for grinding and classifying several hundred pounds of ore at a time.

Concentration Tests.—If the agitation tests on the sample of ore ground to pass 200 sieve are not satisfactory, it will be necessary to try the effect of concentration. For this purpose take about 500 grams of ore, crushed to No. 30 sieve, and carefully pan twice over. The tailing may then be ground in water or cyanide solution as already described and screened wet through a No. 80 or 100 sieve, and again panned. To assist the operation of panning the −100 product, a large proportion of which will be slime, the fine part may be floated off and run over a little canvas table, leaving the granular part only to be panned. In this case, however, the product of the canvas table will need to be cleaned in the pan, but this is simpler and less tedious than panning the whole. The tailing, after being sampled for assay, may be cyanided as it is, or first ground to pass a No. 200 sieve.

Lead Salts.—When experimenting with silver ores some charges should always be made up with the addition of varying amounts of lead acetate or litharge as the presence of lead will often be found a considerable aid to extraction of the silver, particularly when there is no zinc present in the solution.

If litharge be used for this purpose it should be ground fine, say to pass 200 mesh.

In the case of many ores a critical point in the addition of lead compounds is reached which yields the maximum increase in extraction and beyond which the improvement due to its use declines until a point is reached when the extraction is no better and sometimes even worse than without the addition of lead so the determination of the right amount of lead compound to be used is a matter of great importance.

Test for Repeated Use of the Same Solution.—In dealing with any unknown ore there is an important experiment which should never be neglected. It sometimes happens, especially with ores containing arsenic and antimony, that a test made with new solution will give a high extraction, while a similar test made up with solution which has been used over several times after precipitation with zinc will give a much lower one. In the case of such ores the presence of lead is often as inimical to a good extraction as the presence of zinc.

To make this test it is necessary to start the series with several bottle charges so as to furnish sufficient solution with which to carry through the experiment. Each time the solution is used there is a loss due to withdrawals for titration and other causes. The first of the series may be made with 4 bottles, charged in the usual way, the solution after treatment and precipitation with zinc dust being made up to cyanide strength and used to treat the next series of perhaps 3 bottle charges of the original size or 4 smaller charges, and so on until the same solution has been used over 5 or 6 times, by which time the original solution will probably have shrunk to only sufficient for one bottle charge. If by the 6th round no deterioration in extraction has been noted it is safe to assume that the activity of the solution will not be impaired by repeated use on a working scale, under the conditions of zinc precipitation. For this experiment the use of a laboratory vacuum filter is almost a necessity.

The whole amount of wash, however, necessary for displacing the dissolved metals preparatory to the assay of the ore residue should not be added to the stock solution as that would dilute it out of all proportion to milling conditions. Assuming that the filter cake when firmly set contains about 25% moisture, and working with a 250-gm. charge of ore 83 cc of water would be first put on and allowed to leach through, to be at once removed from the receiver and added to the stock solution. After this ample water washes are given to ensure the complete removal of soluble gold and silver from the cake.

Use of Working Plant Solution.—When making experiments on an unknown ore using barren solution from a working plant where zinc precipitation is used a serious error in the calculation of cyanide consumption is likely to occur if the figures are based on the titrations for free cyanide, because when such solution comes in contact with the lime added in making up a bottle charge some of the zinc-potassium cyanide will be regenerated and recorded as free cyanide, and thus the consumption will appear much less than the actual figure should be. This may be avoided by making a complete set of titrations for total cyanide when making the usual ones for free cyanide, and then basing the consumption calculation on the readings for total cyanide.

When the ore, however, itself contains zinc soluble in cyanide solution it is practically impossible to estimate the cyanide consumption correctly when using barren plant solution.

Larger Agitation Charges.—Where there are conveniences for grinding and classifying several hundred pounds of ore at a time it is often of advantage to check the results of bottle tests by charges of 50 to 100 lb. For this purpose it is easy to fit up some small tubs of wood or sheet iron and fix mechanical stirring devices in them. Such mechanical agitation is sufficient to give the maximum extraction in many cases, but with ores that have a reducing tendency it is often necessary to supplement it with an air lift, to facilitate aeration of the charge. If there is a workshop available it is not difficult to have some small Pachuca tanks made out of sheet iron, and these are often very useful for experimental work. When using such tanks in dry climates it is needful to be on the watch for considerable evaporation, resulting in a decreased ratio of dilution, and consequent concentration of precious metal and cyanide content in the solution.

It is also necessary, as already pointed out, to see that the chances for an accumulation of dead pulp in the cone are reduced to a minimum as this is a fertile cause of salting the residue sample.

High Cyanide Consumption.—If experiments on any ore show an abnormally heavy consumption of cyanide (and assuming, of course, that protective alkali has been present in excess the whole time), it is advisable to try to ascertain the cause of the loss. By testing the solution after contact with the ore, for zinc, copper, silver, sulphocyanate, and ferrocyanide, it is often possible to account for nearly the whole amount consumed. Should there remain a considerable proportion unaccounted for it may be due to the presence and dissolution of some metal or metals other than those mentioned, or it may be due to oxidation by the use of excessive aeration, or occasionally to the presence in the pulp of organic substances.

It is only possible here to give an outline of the more common methods of preliminary experiment, and to suggest lines of investigation. Special conditions will call for variation and increase of detail, which each operator will be able to develop according to his needs.

CHAPTER XIII

COST SHEETS AND PLANT DATA

TABLE 14.—SAND COLLECTING TANKS

Name of mill	Number of tanks	Size, ft., diam., depth	Total capacity, cu. ft.	Capacity per 24 hours, cu. ft. per ton
Waihi Grand Junction....	4*	22.5 × 10	15,904	115 (estimated)
Tonopah Belmont........	2	28.0 × 7	8,616	
Homestake No. 1 Plant...	20*	44.0 × 9	273,695	

TABLE 15.—SAND LEACHING TANKS

Name of mill	Number of tanks	Size, ft., diam., depth	Total capacity, cu. ft.	Capacity per 24 hours, cu. ft. per ton	Time of treatment
Waihi Grand Junction.....	Direct treatment in collecting tanks			100 (estimated)	48 hr.
Tonopah Belmont........	12	28 × 7.0	51,696	689 (estimated)	18 days
Homestake No. 1 Plant....	Direct treatment in collecting tanks			171	145 hr.
El Oro..................	12	40 × 6.5	98,018	594	360 hr.

TABLE 16.—SLIME COLLECTING TANKS

Name of mill	Number of tanks	Size, diam., depth	Total capacity, cu. ft.	Capacity per 24 hours, cu. ft. per ton
Nipissing................	3	34 ft. 13 ft.	35,406	145
Waihi Grand Junction.....	6			
Tonopah Belmont.........	...	28 × 20		
Homestake..............				
El Oro..................	4	36 × 20	81,430	215

* Direct treatment.

TABLE 17.—SLIME TREATMENT TANKS

Name of mill	Style of tank	Size, ft., diam., depth	Number of tanks	Total capacity, cu. ft.	Capacity per ton per day, cu. ft.	Time of treatment including settling (if any)
Nipissing	Stirring arms and air lift	34×13	7	82,619	338	72 hr.
Waihi Grand Junction	Pneumatic	13.25×55	10	60,300	240 (estimated)	24–48 hr.
Tonopah Belmont	Stirring arms and air lift	28×20	5	61,544	424 (estimated)	50 hr.
Homestake		Treatment in Merrill presses.				
El Oro	Stirring arms	34×12	15	163,440	270	56 hr.

TABLE 18.—SLIME FILTERS

Name of mill	Average daily tonnage	Type of filter	Number of leaves	Size of leaf, ft.	Total area, sq. ft.	Area per ton per day, sq. ft.	Time of cycle	
							Hr.	Min.
Nipissing Desulphurizing	244	Butters	60	5×10	5,100	21.0	2	30
Cyaniding	244	Butters	80	5×10	6,800	28.0	3	00
Waihi Grand Junction	250 (estimated)	Moore	96	4.25×16	10,752	43.0	4	30
Tonopah Belmont	145 (estimated)	Butters	120	5×10	10,200	70.0	4	50
Homestake (complete cyanide treatment in press)	1,700	Merrill	2,576	4×6 (outside)	61,824	36.0	7 to 9	0 0
North Star	50	Oliver	Revolving drum. (10×7)		220	4.4		

TABLE 19.—GENERAL REDUCTION COSTS

Name of company	Total tons milled per day	Average assay value		Costs for milling	Costs for cyaniding	Total reduction costs	
		Au	Ag, oz.				
Nipissing low-grade ore.....	250	...	25.0	$0.873	$2.079[1]	$2.952	James Johnston *A. I. M. E.*, Feb., 1914.
Homestake...............	4160	4.13	...	.....	......	......	Clark & Sharwood *I. M. M.* Lond., Nov., 1912.
Liberty Bell..............	485	5.58	2.5	.....	......	1.805	Megraw "Details of Cyanide Practice."
Stratton's Independence.....	333	3.50	...	0.685	0.445	1.239	Megraw "Details of Cyanide Practice."
Goldfield Consolidated.....	910	12.57	...	.....	0.618	1.606	*E. & M. J.*, June 12, 1915.
Hollinger.................	583	13.60	...	.....	......	1.237	*E. & M. J.*, May 22, 1915.
Tonopah Belmont..........	500	4.82	22.3	.....	......	2.564	*E. & M. J.*, June 19, 1915.

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[1] Including preliminary desulphurizing treatment.

TABLE 20.—MILLING AND CYANIDING COSTS, TONOPAH BELMONT[1]

	Labor	Supplies	Power	Total
Crushing and conveying	$0.051	$0.038	$0.028	$0.117
Stamping and regrinding	0.176	0.218	0.348	0.742
Concentration	0.079	0.027	0.038	0.144
Elevating	0.023	0.010	0.037	0.070
Conveying and discharging sands	0.063	0.031	0.015	0.109
Leaching sands	0.050	0.566	0.018	0.634
Agitating slimes	0.060	0.570	0.104	0.734
Filtering and discharging slimes	0.108	0.074	0.023	0.205
Precipitation	0.070	0.194	0.011	0.275
Refining	0.065	0.144	....	0.209
Assaying	0.053	0.025	....	0.078
Motors and lights	0.018	0.015	....	0.033
General expenses	0.181	0.165	0.011	0.357
Total	$0.997	$2.077	$0.633	$3.707

TABLE 21.—MILLING AND CYANIDING COSTS, WAIHI GRAND JUNCTION[2]

Rock breaking	$0.132
Stamping	0.214
Tube milling	0.324
Concentration	0.064
Treatment	0.574
Bullion clean-up	0.064
Assaying	0.044
General	0.100
	$1.516

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[1] "Tonopah Belmont Cyanide Plant," by CLAUDE RICE, *E. & M. J.*, July 15, 1911.

[2] "Cyaniding of Gold-silver Ores at Waihi Grand Junction," by NOEL CARLESS, *I. M. M.*, Lond., April, 1915.

TABLE 22.—CYANIDING COSTS, NIPISSING[1]

	Labor	Supplies	Power	Work shop	Totals
Slime collecting, desulphurizing and transferring	$0.042	$0.294	$0.022	$0.008	$0.366
Alkaline filtering and transferring	0.070	0.017	0.023	0.004	0.114
Cyanide treatment and transferring	0.120	0.793	0.071	0.020	1.004
Cyanide filtering and discharging	0.072	0.041	0.034	0.015	0.162
Clarifying and precipitation	0.050	0.221	0.008	0.005	0.284
Drying, melting and refining	0.074	0.050	.....	.....	0.124
Water supply	0.001	0.001	0.021	0.002	0.025
Totals	$0.429	$1.417	$0.179	$0.054	$2.079

TABLE 23.—ALVARADO MINING AND MILLING COMPANY, PARRAL, CHIHUAHUA, MEXICO[2]

It should be noted that this table of costs represents the work in 1918 under the adverse conditions caused by the European war in addition to those arising from the unsettled condition of Mexico itself.

Account *Milling Department* *For Year* 1918

Tonnage for year: 157,339 U. S. Currency

	Average cost per metric ton
Crushers & Conveyors	$0.0553
Stamp Batteries	0.1645
Sand Tables	0.0117
Elevators & Classifiers	0.0440
Tube Mills	0.2264
Slime Tables	0.0086
Mill Storage Tanks & Pipe Lines	0.0043
Electric Power	0.4827
Salaries	0.0117
Ball Mills	0.1534
Water Supply	0.0114
	$1.1740

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[1] "Mill and Practice of the Nipissing Mining Co." by JAMES JOHNSTON, *A. I. M. E.*, February, 1914.

[2] Kindly furnished by T. G. Hawkins, Jr., Resident Manager.

TABLE 23.—ALVARADO MINING AND MILLING COMPANY, PARRAL, CHIHUAHUA, MEXICO—(*Continued*)

Account *Cyaniding Department* *For Year* 1918

Tonnage for year: 156,968 U. S. Currency

	Average cost per metric ton
Dorr Thickeners	$0.0102
Pachuca Tanks & Elevators	0.0277
Butters Filters	0.0860
Oliver Filters	0.0266
Storage Tanks	0.0003
Clarifiers	0.0056
Pumps	0.0173
Zinc House	0.0499
Tailing Pond	0.0027
Chemicals—Cyaniding	0.8053
Chemicals—Zinc House	0.2927
Electric Power	0.0738
Salaries	0.0152
Water Supply	0.0115
Compressed Air	0.0384
	$1.4632

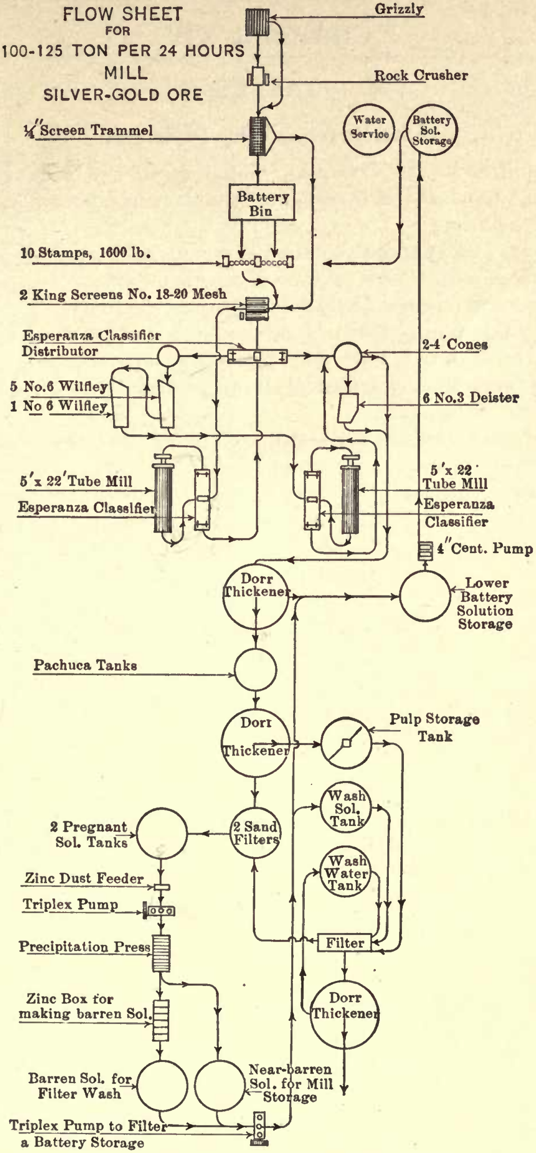

Fig. 31.—Flow Sheet of a Mexican Mill.

CHAPTER XIV

TABLES

Antidote for Cyanide Poisoning

The Committee of the Chemical, Metallurgical and Mining Society of South Africa,[1] appointed to investigate cyanide poisoning, recommend as an antidote the following:

1. Thirty cc of a 23 per cent. solution of ferrous sulphate.
2. Thirty cc of a 5 per cent. solution of caustic potash.
3. Two grams of powdered oxide of magnesium (light).

In every cyanide-room there should be kept three boxes, containing—

1. A metal receptacle to hold about a pint, and a spoon.
2. A blue hermetically sealed phial, containing 30 cc of a 33 per cent. solution of ferrous sulphate.
3. A white phial, hermetically sealed, containing 30 cc of caustic potash.
4. A packet of oxide of magnesium (light).

Preparation of Antidote.—Quickly empty the contents of the blue phial, of the white phial, and of the magnesia package into the metal receptacle, and stir well with the spoon. This should be done as rapidly as possible, as the patient's chance of life depends on promptness.

Administration of the Antidote.—If the patient is conscious, make him swallow the mixture at once and lie down for a few minutes. If the patient is not conscious, place him on his back and pour the mixture down his throat in small quantities, if necessary pinching his nose in order to make him swallow.

Incite Vomiting.—After the antidote has been given, try to make the patient vomit by tickling the back of his throat with a feather or with the finger, or give a tumblerful of warm water and mustard.

NOTE.—It must always be kept in mind that promptness of action is the first essential, as the poison acts quickly.

The box of remedies should never be removed from its place, but always kept complete and ready for an emergency. It is advisable to keep the remedies in duplicate in case of the accidental breaking of a sealed bottle.

In all cases of cyanide poisoning medical aid should be summoned without delay.

Tyler Standard Sieves

This screen scale has as its base an opening of 0.0029 in. which is the opening in 200 mesh 0.0021 in. wire, the standard sieve as adopted by the Bureau of Standards of the U. S. Govt., the openings increasing in the ratio of the sq. root of 2 or 1.414 except in the sizes Nos. 65 to 200 which increase in the ratio of the 4th root of 2 or 1.189.

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[1] *Journal Chem., Metall. and Min. Soc. of S. A.*, May, 1904.

TABLE 24

Opening in inches	Opening in millimeters	Mesh	Diameter of wire in decimals of 1 in.
1.0500	26.670	...	0.1490
0.7420	18.850	...	0.1350
0.5250	13.330	...	0.1050
0.3710	9.423	...	0.0920
0.2630	6.680	3	0.0700
0.1850	4.699	4	0.0650
0.1310	3.327	6	0.0360
0.0930	2.362	8	0.0320
0.0650	1.651	10	0.0350
0.0460	1.168	14	0.0250
0.0328	0.833	20	0.0172
0.0232	0.589	28	0.0125
0.0164	0.417	35	0.0122
0.0116	0.295	48	0.0092
0.0082	0.208	65	0.0072
0.0069	0.175	80	0.0056
0.0058	0.147	100	0.0042
0.0049	0.124	115	0.0038
0.0041	0.104	150	0.0026
0.0035	0.088	170	0.0024
0.0029	0.074	200	0.0021

Table 25

Standard Sieves

Adopted by the I. M. M., London

Opening in inches	Opening in millimeters	Mesh	Diameter of wire in decimals of 1 in.
0.1000	2.540	5	0.1000
0.0620	1.575	8	0.0630
0.0500	1.270	10	0.0500
0.0416	1.057	12	0.0417
0.0312	0.792	16	0.0313
0.0250	0.635	20	0.0250
0.0200	0.508	25	0.0200
0.0166	0.422	30	0.0167
0.0142	0.361	35	0.0143
0.0125	0.318	40	0.0125
0.0100	0.254	50	0.0100
0.0083	0.211	60	0.0083
0.0071	0.180	70	0.0071
0.0062	0.157	80	0.0063
0.0050	0.127	100	0.0050
0.0033	0.084	150	0.0033
0.0025	0.064	200	0.0025

Air-lift Data Tabulated

By A. W. Allen

(Engineering and Mining Journal)

The accompanying data on air lifts have been put into tabular form in order that they may be more conveniently referred to when necessary. The figures of principal importance are given.

TABLE 26.—AIR PRESSURE TO OPERATE LIFTS

Lift, ft.	Submergence, ft.	Theoretical gage pressure in pounds per square inch to elevate fluid of specific gravity					
		1.0	1.1	1.2	1.3	1.4	1.5
5	7.5	3.2	3.6	3.9	4.2	4.5	4.9
10	15.0	6.5	7.1	7.8	8.4	9.1	9.7
15	22.5	9.7	10.7	11.6	12.6	13.6	14.5
20	30.0	13.0	14.3	15.6	16.9	18.2	19.5
25	37.5	16.2	17.8	19.4	21.0	22.7	24.3
30	45.0	19.5	21.4	23.4	25.3	27.3	29.2
35	52.5	22.7	25.0	27.2	29.5	31.8	34.0
40	60.0	26.0	28.6	31.2	33.8	36.4	39.0
45	67.5	29.2	32.1	35.0	38.0	40.9	43.8
50	75.0	32.5	35.7	39.0	42.2	45.5	48.7
5	10.0	4.3	4.7	5.2	5.6	6.0	6.5
10	20.0	8.7	9.6	10.4	11.3	12.2	13.0
15	30.0	13.0	14.3	15.6	16.9	18.2	19.5
20	40.0	17.3	19.0	20.8	22.5	24.2	25.9
25	50.0	21.6	23.8	25.9	28.1	30.2	32.4
30	60.0	26.0	28.6	31.2	33.8	36.4	39.0
35	70.0	30.3	33.3	36.4	39.4	42.4	45.4
40	80.0	34.6	38.0	41.5	45.0	48.4	51.9
45	90.0	39.0	42.9	46.8	50.7	54.6	58.5
50	100.0	43.3	47.6	52.0	56.3	60.6	64.9

An increase in pressure of from 10 to 20% over the figures given will be necessary in practice.

TABLE 27.—AVERAGE CAPACITY OF AIR-LIFTS

Diameter of lift pipe	Cubic feet per minute	Tons per 24 hours with fluid of specific gravity					
		1.0	1.1	1.2	1.3	1.4	1.5
2 in.	6	270	297	324	351	378	405
2½ in.	10	450	495	540	585	630	675
3 in.	14	630	693	756	819	882	945
3½ in.	20	900	990	1080	1170	1260	1350
4 in.	25	1125	1237	1350	1462	1575	1687
5 in.	40	1800	2980	2160	2340	2520	2700
6 in.	56	2520	2772	3024	3276	3528	3780

TABLE 28.—CONVERSION TABLE, FAHRENHEIT TO CENTIGRADE SCALE

Deg. C.	Deg. F.	Deg. C.	Deg. F.	Deg. C.	Deg. F.	Deg. C.	Deg. F.
100	212.0	70	158.0	40	104.0	10	50.0
99	210.2	69	156.2	39	102.2	9	48.2
98	208.4	68	154.4	38	100.4	8	46.4
97	206.6	67	152.6	37	98.6	7	44.6
96	204.8	66	150.8	36	96.8	6	42.8
95	203.0	65	149.0	35	95.0	5	41.0
94	201.2	64	147.2	34	93.2	4	39.2
93	199.4	63	145.4	33	91.4	3	37.4
92	197.6	62	143.6	32	89.6	2	35.6
91	195.8	61	141.8	31	87.8	1	33.8
90	194.0	60	140.0	30	86.0	0	32.0
89	192.2	59	138.2	29	84.2		
88	190.4	58	136.4	28	82.4		
87	188.6	57	134.6	27	80.6		
86	186.8	56	132.8	26	78.8		
85	185.0	55	131.0	25	77.0		
84	183.2	54	129.2	24	75.2		
83	181.4	53	127.4	23	73.4		
82	179.6	52	125.6	22	71.6		
81	177.8	51	123.8	21	69.8		
80	176.0	50	122.0	20	68.0		
79	174.2	49	120.2	19	66.2		
78	172.4	48	118.4	18	64.4		
77	170.6	47	116.6	17	62.6		
76	168.8	46	114.8	16	60.8		
75	167.0	45	113.0	15	59.0		
74	165.2	44	111.2	14	57.2		
73	163.4	43	109.4	13	55.4		
72	161.6	42	107.6	12	53.6		
71	159.8	41	105.8	11	51.8		

Conversion Formula

$$c \times \frac{9}{5} + 32 = F$$

$$F - 32 \times \frac{5}{9} = C$$

TABLE 29.—INTERNATIONAL ATOMIC WEIGHTS, 1917

Element	Symbol	Atomic weight	Element	Symbol	Atomic weight
Aluminium	Al	27.100	Molybdenum	Mo	96.000
Antimony	Sb	120.200	Neodymium	Nd	144.30
Argon	A	39.880	Neon	Ne	20.20
Arsenic	As	74.960	Nickel	Ni	58.68
Barium	Ba	137.370	Niton (radium emanation)	Nt	222.40
Bismuth	Bi	208.000	Nitrogen	N	14.01
Boron	B	11.000	Osmium	Os	190.90
Bromine	Br	79.920	Oxygen	O	16.00
Cadmium	Cd	112.400	Palladium	Pd	106.70
Cæsium	Cs	132.810	Phosphorus	P	31.04
Calcium	Ca	40.070	Platinum	Pt	195.20
Carbon	C	12.005	Potassium	K	39.10
Cerium	Ce	140.250	Praseodymium	Pr	140.90
Chlorine	Cl	35.460	Radium	Ra	226.00
Chromium	Cr	52.000	Rhodium	Rh	102.90
Cobalt	Co	58.970	Rubidium	Rb	85.45
Columbium	Cb	93.100	Ruthenium	Ru	101.70
Copper	Cu	63.570	Samarium	Sa	150.40
Dysprosium	Dy	162.500	Scandium	Sc	44.10
Erbium	Er	167.700	Selenium	Se	79.20
Europium	Eu	152.000	Silicon	Si	28.30
Fluorine	F	19.000	Silver	Ag	107.88
Gadolinium	Gd	157.300	Sodium	Na	23.00
Gallium	Ga	69.900	Strontium	Sr	87.63
Germanium	Ge	72.500	Sulphur	S	32.06
Glucinum	Gl	9.100	Tantalum	Ta	181.50
Gold	Au	197.200	Tellurium	Te	127.50
Helium	He	4.000	Terbium	Tb	159.20
Holmium	Ho	163.500	Thallium	Tl	204.00
Hydrogen	H	1.008	Thorium	Th	232.40
Indium	In	114.800	Thulium	Tm	168.50
Iodine	I	126.920	Tin	Sn	118.70
Iridium	Ir	193.100	Titanium	Ti	48.10
Iron	Fe	55.840	Tungsten	W	184.00
Krypton	Kr	82.920	Uranium	U	238.20
Lanthanum	La	139.000	Vanadium	V	51.00
Lead	Pb	207.200	Xenon	Xe	130.20
Lithium	Li	6.940	Ytterbium (Neoytterbium)	Yb	173.50
Lutecium	Lu	175.000	Yttrium	Yt	88.70
Magnesium	Mg	24.320	Zinc	Zn	65.37
Manganese	Mn	54.930	Zirconium	Zr	90.60
Mercury	Hg	200.600			

WEIGHTS AND MEASURES, ENGLISH AND AMERICAN SYSTEM

Linear Measure

12 inches	1 foot
3 feet	1 yard
5.5 yards	1 rod
40 rods	1 furlong
8 furlongs	1 mile

Inches	Feet	Yards	Rods	Furlongs	Mile
36	3.0	1.0			
198	16.5	5.5	1		
7920	660.0	220.0	40	1	
63,360	5280.0	1760.0	320	8	1

Square Measure

144 square inches	1 square foot
9 square feet	1 square yard
30¼ square yards	1 square rod
160 square rods	1 acre
640 acres	1 square mile

Cubic Measure

1728 cubic inches	1 cubic foot
27 cu. feet	1 cubic yard

Avoirdupois Weight

English

437.5 grains	1 ounce (oz.)
16 ounces	1 pound (lb.)
14 pounds	1 stone (st.)
28 pounds	1 quarter (qr.)
112 pounds	1 hundredweight (cwt.)
20 cwt. (2240 lb.)	1 ton (T.)

American

437.5 grains	1 ounce (oz.)
16 ounces	1 pound (lb.)
100 pounds	1 hundredweight (cwt.)
20 cwt. (2000 lb.)	1 ton (T.)

Troy Weight

24 grains	1 pennyweight (dwt.)
20 pennyweights	1 ounce (oz).
12 ounces	1 pound (lb.)
1 lb. Troy	5760 grains
14.5833 oz. Troy	1 lb. avoirdupois
29,166.6 oz. Troy	2000 lb. avoirdupois

Liquid Measure

4 gills	1 pint
2 pints	1 quart
4 quarts	1 gallon

1 U. S. gallon—231 cu. in.—0.134 cu. ft.
1 cu. ft.—7.481 U. S. gallons
1 British imperial gallon—277.274 cu. in.—0.16046 cu. ft.
1 cu. ft.—6.234 imperial gallons

1 cu. ft.	62.42 lb. of water
1 ton (2000 lb.)	32.04 cu. ft. of water
1 ton (2000 lb.)	239.69 U. S. gallons

THE METRIC SYSTEM

Linear Measure

10 millimeters (mm.)	1 centimeter (cm.)
10 centimeters	1 decimeter (dm.)
10 decimeters	1 meter (m.)
10 meters	1 decameter (Dm.)
10 decameters	1 hectometer (Hm.)
10 hectometers	1 kilometer (Km.)

Square Measure

100 square millimeters (mm.^{2})	1 square centimeter (cm.^{2})
100 square centimeters	1 square decimeter (dm.^{2})
100 square decimeters	1 square meter (m.^{2})

Cubic Measure

1000 cubic millimeters (mm.^{3})	1 cubic centimeter (cm.^{3} or cc)
1000 cubic centimeters	1 cubic decimeter (dm.^{3})
1000 cubic decimeters	1 cubic meter (m.^{3})

Liquid Measure

10 millilitres (ml.)	1 centilitre (cl.)
10 centilitres	1 decilitre (dl.)
10 decilitres	1 litre (l.)
10 litres	1 decalitre (Dl.)
10 decalitres	1 hectolitre (Hl.)
10 hectolitres	1 kilolitre (Kl.)
1 litre	1000 cubic centimetres
1 cu. ft. of water	28.317 litres
1 litre of water	61.023 cu. in. = 0.03531 cu. ft.
1 metric ton of water	35.319 cu. ft.
1 metric ton of water	264.2 U. S. gallons

Measure of Weight

10 milligrams (mg.)	1 centigram (cg.)
10 centigrams	1 decigram (dg.)
10 decigrams	1 gram (g.)
10 grams	1 decagram (Dg.)
10 decagrams	1 hectogram (Hg.)
10 hectograms	1 kilogram (Kg.)
1000 kilograms	1 ton (T.)

1 gram = the weight of 1 cubic centimeter of pure distilled water at a temperature of 4° cent.

1 kilogram = the weight of 1 litre of water

1 ton = the weight of 1 cubic meter of water

CONVERSION TABLES

Linear Measure

1 in.	2.53998 cm.	1 cm.	0.3937043 in.
1 ft.	0.30479 m.	1 m.	39.37043 in.
1 yd.	0.914399 m.	1 m.	3.28083 ft.
1 mi.	1.60934 km.	1 m.	1.09362 yd.
		1 km.	0.62137 mi.

Square Measure

1 centar (1 sq. m.)	1550 sq. in.—10.764 sq. ft.
1 ar	119.6 sq. yd.
1 hectar	2.47104 acres
1 sq. cm.	0.155 sq. in.
1 sq. meter	10.76390 sq. ft.
1 sq. km.	0.3861 sq. mi.
1 acre	0.40469 hectar
1 sq. in.	6.4516 sq. cm.
1 sq. ft.	0.092903 sq. m.

Cubic Measure

1 cu. in.	16.38720 cc
1 cc.	0.06102376 cu. in.
	0.0000353 cu. ft.
1 cu. ft.	0.028317 cu. m.
1 cu. m.	35.31445 cu. ft.
	1.30794 cu. yd.
1 cu. yd.	0.764553 cu. m.

Liquid Equivalents

1 fluid ounce	29.5737 millilitres
1 millilitre	0.03381 fluid ounce—0.061027 cubic inch
1 quart	0.94636 litres
1 litre	1.0567 quarts
1 U. S. gallon	3.78543 litres
1 litre	0.26417 gallons

Dry Equivalents

1 hectolitre	2.8377 bushels
1 U. S. bushel	0.35239 hectolitres

Weight

1 oz. avoirdupois	28.34954 grams
1 lb. avoirdupois	453.59 grams
1 (short) ton	907.18 kg.
1 gram	0.035274 oz. avoir.—0.00220 lb.
1 kg	35.27392 oz. avoir.—2.2046223 lb.
1 metric ton	1.102311 short ton—0.9842 long T.
1 grain	64.799 mg.
1 dwt	1.55518 grams
1 oz. Troy	31.1035 grams
1 lb. Troy	0.37324 kg.
1 gram	15.4324 grains—0.64301 dwt.
1 gram	0.03215 oz. Troy
1 kg	32.15076 oz. Troy—2.67923 lb. Troy

1 oz. Troy per ton of 2000 lb.—34.285 grams per metric ton

TABLE 30.—ASSAY CONVERSION TABLE

	Per cent.	Grams per metric ton (1000 kilos)	Oz. Troy per long ton (2240 lb.)	Oz. Troy per metric ton (2204.6 lb.)	Oz. Troy per short ton (2000 lb.)
1 per cent....................	1.00000	10,000.00	326.66600	321.50700	291.66600
1 gram per metric ton.........	0.00010	1.00000	0.03266	0.03215	0.02916
1 oz. Troy per long ton.......	0.00300	30.61200	1.00000	0.98420	0.89280
1 oz. Troy per metric ton.....	0.00310	31.10348	1.01600	1.00000	0.90710
1 oz. Troy per short ton.....	0.00340	34.28500	1.12000	1.10200	1.00000
1 dollar per short ton (gold at $20.67 per oz.).............	0.00016	1.66000	0.05400	0.05300	0.04800

VARIOUS FORMULAS

Area of a Circle

$$\text{Radius}^2 \times 3.1416 \quad \text{or} \quad \text{Diameter}^2 \times 0.7854$$

Circumference of a Circle

$$\text{Diameter} \times 3.1416$$

Cubic Contents of Cylindrical Tank

$$\text{Area} \times \text{depth}$$

Cubic Contents of Cone

$$\frac{\text{Area of base}}{3} \times \text{depth}$$

CALCULATION OF RECOVERY IN CONCENTRATE

Let C = assay value of concentrate
Let H = assay value of head
Let T = assay value of tailing
Let R = Ratio of concentration
Let W = weight of concentrate (per cent.)
Let P = percentage recovery
Let L = percentage loss in tailing

$$R = \frac{C - T}{H - T}$$

$$W = \frac{H - T}{C - T} \times 100$$

$$P = \frac{100 \times C(H - T)}{H\,(C - T)}$$

$$P = \frac{C \times 100}{H \times R}$$

$$L = \frac{100 \times T(C - H)}{H\,(C - T)}$$

Elimination of Middling by Calculation.—This is based on the assumption that the return of the middling to the head of the concentration system does not change the subsequent assay value of the concentrate and tailing. The weights of the various products and the assay of the middling need not be known.

Let X and Y = percentage weights of concentrate and tailing respectively when middling is eliminated.

Let h, c, t, = assay values of head concentrate and tailing respectively.

Let x and y = percentage of total values in concentrate and tailing when middling is eliminated.

$$X = \frac{100\,(h - t)}{c - t} \qquad Y = \frac{100(c - h)}{c - t}$$

$$x = \frac{100\,(h - t)c}{h(c - t)} \qquad y = \frac{100(c - h)t}{h(c - t)}$$

Power. I.H.P.—33,000 foot-pounds per minute = 4562.33 kilogram meters per minute = 0.745941 kilowatts = 1.01385 metric h.p.

1 metric h.p. = 32,549 foot-pounds per minute = 4.500 kilogram meters per minute = 0.735750 kilowatt = 0.98634 h.p.

Pressure.—1 atmosphere = 760 millimeters, or 29.922 inches of mercury column = 10.333 meters, or 33.901 feet of water column = 14.696 pounds per square inch = 1.0333 kilos per square centimeter.

1 pound per square inch = 0.070310 kilo per square centimeter.

1 kilo per square centimeter = 14.223 pounds per square inch.

TABLE 31.—MISCELLANEOUS FIGURES ON THE WEIGHT OF BROKEN ORE (Richards "Ore Dressing",

Weight per cubic foot, lb.	Cubic feet per ton	Material
100	20.0	Mine ore coming to Mill, North Star Mining Co.
100	20.0	Tailings of Mill, Calumet and Hecla Mining Co., Michigan.
100	20.0	"Blue ground" (altered peridotite) as it comes from the Kimberly mines, South Africa.
96	20.8	Broken quartz as it comes from the Jumpers' Deep mine, South Africa.
95	21.1	Broken quartz conglomerate as it comes from the Ferreira mine, South Africa.
87	23.0	Material from gravity stamps.
77	26.0	Material from gravity stamps, with the slimes removed.
75	26.6	Quartz stamped dry through 40 mesh and charged dry into vats at the Victoria mill, Hauraki peninsula, New Zealand.

TABLE 32.—METALS AND THEIR MINERALS BASED ON INFORMATION FROM DANA'S TEXTBOOK OF MINERALOGY

Metal	Composition	Mineral	Formula	Specific gravity	Hardness	Color and characteristics
Antimony.......	Sulphide	Stibnite (Antimony) glance)	Sb_2S_3	4.5–4.6	2.0	Lead-grey, metallic lustre, slightly sectile.
Cobalt..........	Arsenide	Smaltite	$CoAs_2$	6.4–6.6	5.5–6	Tin-white, metallic, opaque.
	Sulpharsenide	Cobaltite	$CoAsS$	6.0–6.3	5.5	Silver-white to redish, cubic, metallic.
Copper.........	Metallic	Native	Cu	8.8–8.9	2.5–3	Copper-red, malleable and ductile.
	Oxide	Melaconite (Black oxide)	CuO	5.8–6.2	3	Black.
	Oxide	Cuprite (Red oxide)	Cu_2O	5.8–6.1	3.5–4	Red, brittle.
	Carbonate	Malachite	$CuCO_3Cu(OH)_2$	3.9–4.0	3.5–4	Green, brittle.
	Carbonate	Azurite	$2\ CuCO_3.Cu(OH)_2$	3.7–3.8	3.5–4	Azure-blue.
	Sulphide	Chalcopyrite	$CuFeS_2$	4.1–4.3	3.5–4	Brass-yellow, brittle.
	Sulphide	Bornite	Cu_3FeS_3	4.9–5.4	3.0	Copper-red to brown, brittle.
	Sulphide	Covellite	CuS	4.59		Indigo-massive.
Copper.........	Sulphide	Chalcocite (Copper glance)	Cu_2S	5.5–5.8	2.5–3	Blackish lead-grey, often tarnished, sectile.
	Antimonide	Tetrahedrite (grey-copper)	$Cu_8Sb_2S_7$	4.4–5.1	3–4	Flint-grey to iron-black.
	Sulpharsenate	Enargite	Cu_3AsS_4	4.4	3.0	Greyish-black, brittle.
	Silicate	Chrysocolla	$CuSiO_3+2H_2O$	2–2.2	2–4	Turquoise-blue, translucent, vitreous, rather sectile.
	Oxychloride	Atacamite	$Cu_2ClH_3O_3$	3.75	3–3.5	Bright bottle green, brittle.
Gold...........	Metallic	Native	Au	15.6–19.3	2.5–3	Gold-yellow, malleable and ductile.
	Telluride	Sylvanite	$(Au.Ag)Te_2[Au:Ag=1:1]$	7.9–8.3	1.5–2	Steel-grey to silver-white to yellow, brittle,
	Telluride	Calaverite	$(Au.Ag)Te_2[Au:Ag=6:1]$	9.0	2.5	Pale bronze-yellow, massive.
	Telluride	Petzite	$(Ag.Au)_2Te[Au:Ag=1:3]$	8.7–9.0	2.5–3	Steel-grey to iron-black, brittle.

TABLE 32.—METALS AND THEIR MINERALS (*Continued*)

Metal	Composition	Mineral	Formula	Specific gravity	Hard-ness	Color and characteristics
Graphite.........	Carbon	Plumbago	C	2.0–2.23	1–2	Iron-black, dark-steel grey, greasy, flexible.
Iron.............	Metallic	Native	Fe	7.3–7.8	4–5	Steel-grey to iron black malleable.
	Oxide	Hematite	Fe_2O_3	4.9–5.3	5.5–6.5	Steel-grey, red.
	Oxide	Limonite	$2Fe_2O_3, 3H_2O$	3.6–4.0	5–5.5	Brown, opaque.
	Oxide	Magnetite	FeO, Fe_2O_3	5.16	5.5–6.5	Iron-black, magnetic.
	Carbonate	Siderite	$FeCO_3$	3.8	3.5–4	Grey-brown, reddish, brittle.
	Sulphide	Pyrite	FeS_2	4.9–5.1	6–6.5	Brass-yellow, brittle.
	Sulphide	Marcasite	FeS_2	4.85–4.9	6–6.5	Pale bronze-yellow, brittle.
	Sulphide	Pyrrhotite	Fe_5S_6 to $Fe_{16}S_{17}$	4.58–4.64	3.5–4.5	Bronze-yellow to copper-red, easily tarnished.
	Sulpharsenide	Mispickel	$FeAsS$	5.9–6.2	5.5–6	Silver-white, brittle.
	Titanate	Ilmenite	$FeTiO_3$	4.5–5	5–6	Iron-black, slightly magnetic.
	Tungstate	Wolframite	$(Fe, Mn)WO_4$	7.2–7.5	5–5.5	Brownish-black, brittle.
	Chromate	Chromite	$FeCr_2O_4$	4.3–4.57	5.5	Iron and brown black, brittle.
	Manganate	Franklinite	$(FeZnMn)O(FeMn)_2O_3$	5.1–5.22	5.5–6.5	Iron-black.
Lead............	Metallic	Native	Pb	11.37	1.5	Lead-grey.
	Carbonate	Cerussite	$PbCO_3$	6.46–6.57	3–3.5	White-grey, very brittle.
	Sulphate	Anglesite	$PbSO_4$	6.3	2.75–3	White, grey, green, very brittle.
	Sulphide	Galenite	PbS	7.4–7.6	2.5–2.75	Lead-grey.
Mercury.........	Metallic	Native	Hg	13.59	....	Tin-white, metallic, brilliant.
	Sulphide	Cinnabar	HgS	8.0–8.2	2–2.5	Cochineal-red, sectile.
Molybdenum.....	Sulphide	Molybdenite	MoS_2	4.7–4.8	1–1.5	Lead-grey, foliated, flexible, sectile.
	Oxide	Molybdite	MoO_3	4.5	1–2	Straw-yellow, capillary, earthy.

TABLE 32.—METALS AND THEIR MINERALS (*Continued.*)

Metal	Composition	Mineral	Formula	Specific gravity	Hardness	Color and characteristics
Nickel	Arsenide	Niccolite Kupfernickel	$NiAs$	7.3–7.67	5–5.5	Pale copper-red, metallic, opaque.
	Sulphide	Millerite	NiS	5.3–5.65	3–3.5	Brass to bronze-yellow, capillary crystals.
	Silicate	Garnierite	$H_2(NiMg)SiO_4$	2.3–2.8	....	Apple-green, amorphous, soft, friable.
Silver	Metallic	Native	Ag	10.1–11.1	2.5–3	Massive, filiform, malleable.
	Antimonide	Dyscrasite	Ag_3Sb to Ag_6Sb	9.4–9.8	3.5–4	Silver-white, sectile.
	Chloride	Cerargyrite (Horn-silver)	$AgCl$ [Cl 24.7, Ag 75.3]	5.5	1-1.5	Pearl-grey, resembles wax, highly sectile.
	Bromide	Bromyrite	$(AgBr)$	5.8–6	......	Bright yellow.
	Sulphide	Argentite (Silver-glance)	Ag_2S[S 12.9, Ag 87.1]	7.2–7.36	2–2.5	Lead-grey, sectile.
	Sulphantimonide	Pyrargyrite (Dark ruby silver)	Ag_3SbS_3	5.7–5.8	2.5	Black, grey-black, brittle.
	Sulpharsenide	Proustite (Light ruby silver)	Ag_3AsS_3	5.5–5.6	2–2.5	Scarlet-vermillion, brittle.
	Sulphantimonide	Stephanite (Brittle silver ore)	Ag_5SbS_4	6.2–6.3	2–2.5	Iron-black, brittle.
	Sulphantimonide	Polybasite	Ag_9SbS_6	6.0–6.2	2–3	Iron-black.
Tin	Oxide	Cassiterite	SnO_2	6.8–7.1	6–7	Brown, black, infusible.
	Sulphostannate	Stannite Tin Pyrites	Cu_2FeSnS_4	4.3–4.5	4.0	Steel-grey to iron-black.
Tungsten	Ferrous	Wolframite	$(FeMn)WO_4$	7.2–7.5	5–5.5	Dark grey to brown-black, opaque, weakly magnetic.
	Manganous	Hübnerite	$MnWO_4$	....	......	Red to hair-brown, nearly black.
	Calcareous	Scheelite	$CaWO_4$	5.9–6.1	4.5–5	White, yellow, brownish, translucent.
	Oxide	Tungstite	WO_3	....	......	Yellow, greenish, earthy.
Zinc	Oxide	Zincite	ZnO	5.4–5.7	4–4.5	Deep red, orange-yellow, perfect cleavage.
	Carbonate	Smithsonite	$ZnCO_3$	4.3–4.45	5.0	White, greyish, greenish, brittle
	Sulphide Zinc blende	Sphalerite	ZnS	3.9–4.1	3.5–4	Yellow, brown, black, perfect clevage.
	Silicate	Calamine	H_2ZnSiO_5	3.4–3.5	4.5–5.0	White, yellowish to brown, brittle.
	Silicate	Willemite	$ZnSiO_4$	3.89–4.18	5.5	White, apple-green, brown.

TABLE 33.—GANGUE MINERALS

Mineral	Composition	Formula	Specific gravity	Hardness	Color and characteristics
Barite (heavy spar)....	Barium sulphate	$BaSO_4$	4.3–4.6	2.5–3.5	White, perfect cleavage, vitreous.
Bauxite..............	Alumina oxide	$Al_2O_3, 2H_2O$	2.55		Whitish, yellow, red, concretionary, earthy.
Biotite (Black mica)...	Aluminium-potassium silicate	$(HK)_2(MgFe)_2Al_2(SiO_4)_3$	2.7–3.1	2.5–3	Black, lustre splendent, flexible laminæ.
Calcite...............	Calcium carbonate	$CaCO_3$	2.71	3.0	White, to colorless, transparent, perfect cleavage.
Corundum............	Alumina oxide	Al_2O_3	3.95–4.1	9.0	Blue, brown to black, brittle to tough.
Dolomite.............	Calcium-magnesium Carbonate	$(Ca, Mg)CO_3$	2.8–2.9	3.5–4	White, reddish, brown, vitreous, brittle.
Fluorite (Fluor-spar) ..	Calcium fluoride	CaF_2	3.0–3.25	4.0	White, green, rose, violet, phosphoresces when heated.
Gypsum..............	Calcium sulphate	$CaSO_4+2H_2O$	2.3	1.5–2.0	White, grey, yellow, blue, foliated, pearly lustre.
Hornblende...........	Calcium-magnesium silicate	$RSiO_3$	2.9–3.4	5–6	Black, green, vitreous, perfect cleavage.
Muscovite (White mica).........	Aluminium-potassium silicate	$(H, K)AlSiO_4$	2.76–3	2–2.5	Colorless, grey, transparent, flexible laminæ.
Quartz...............	Silicon oxide	SiO_2	2.65	7.0	Colorless, white, transparent to opaque.
Rhodochrosite	Manganese carbonate	$MnCO_3$	3.45–3.6	3.5–4.5	Rose-red, dark red, brown, vitreous to pearly.
Rhodonite............	Manganese silicate	$MnSiO_3$	3.4–3.68	5.5–6.5	Rose pink, brownish red, vitreous.
Rutile................	Titanium oxide	TiO_2	4.18–4.25	6–6.5	Brownish red, transparent to opaque.
Serpentine...........	Magnesium silicate	$3MgO\cdot 2SiO_2\cdot 2H_2O$	2.5–2.65	2.5–4	Green, yellow, brown, fibrous, silky lustre.
Spinel...............	Magnesium-aluminium oxide	$MgAl_2O_4$	3.5–4.1	8.0	Red, brown, black, vitreous, splendent
Talc.................	Magnesium silicate	$H_2Mg_3(SiO_3)_4$	2.7–2.8	1–1.5	Apple-green, white, greasy, flexible, sectile.

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