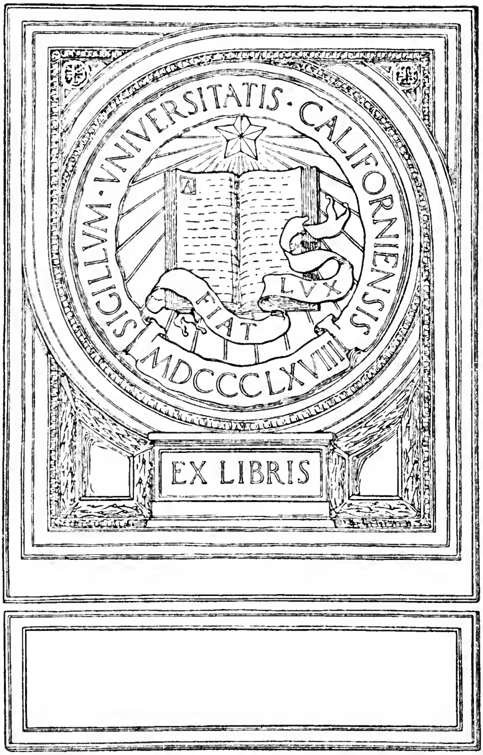
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TEXT BOOK OF CYANIDE PRACTICE

BY

H. W. MacFARREN

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"Practical Stamp Milling and Amalgamation."
"Mining Law for the Prospector, Miner and Engineer."

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PREFACE

THE cyanide process is a subject comprehending many divisions, and one that can best be treated along some phase or with a special point in view. The point in view or purpose of this work is to furnish students, cyanide workers, and those generally and technically interested in the subject, with a practical and technical exposition of the principles and basic practice applicable to cyanidation in general, and not of the particular practice at any plant or locality. An exposition not too technical and complicated or comprehensive for those who are acquiring or about to acquire their technical equipment, nor too superficial for the experienced operator. Intended primarily to guide the first footsteps and early progress of those who hope to eventually operate plants, the author has used simple explanations and many repetitions and references to other parts of the text in the effort to clarify a subject that is confusing, to say the least, to the beginner. Desiring to produce a work that will be of use in actual practice, rather than something to simply add to one's technical library, the author has not kept within academic limitations, but has resorted to homely methods of explanation where deemed advisable. Though going into considerable detail, no branch of the subject has been carried to that point where it should be taken up as a special subject. It aims to lift the careful reader and student to that point where he may intelligently seek further information in the literature on investigations into special and abstruse details of the subject, for which purpose and as an aid to the advanced worker an extended Classified Bibliography of the more accessible literature on the cyanide process is included.

Where it has been impossible to give definite figures without going into explanatory details, obviously beyond the scope of this work, figures representing the average or rational extremes in practice have been given. The reader may consider these as carefully selected to represent accurately approved modern practice.

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TEXT BOOK OF CYANIDE PRACTICE

CHAPTER I

HISTORY AND DEVELOPMENT

The cyanide process for the extraction of gold and silver from their ores is based on the facts that a very dilute cyanide solution will dissolve the precious metals from the ore, and that when this enriched solution is brought into contact with finely divided zinc, the gold and silver will be precipitated so that it may be collected and melted into a bar of bullion.

Discovery and Early Use of Cyanide. — Prussian blue, the first cyanide compound known, was discovered in 1704. In 1782 it was first dimly noted that a cyanide solution would dissolve gold and silver. During the succeeding years many different compounds of cyanide were determined and something of their properties learned. The first patent on the solubility of gold in cyanide solution was taken out in Great Britain in 1840, and led to the use of cyanide solution for dissolving gold and silver for electroplating purposes. In 1844 Elsner published valuable investigations regarding the solubility of gold and silver in cyanide solution.

The first patent purporting to use a cyanide solution for dissolving gold and silver from their ores was taken out in the United States by J. H. Rae in 1867. Later on several somewhat similar patents were issued, the most important one being to J. W. Simpson in 1885. In this last patent, cyanide of potassium was to be used in connection with other chemicals for extracting gold, silver, and copper from their ores; while the metals were to be precipitated from the solution on zinc plates, and obtained therefrom by scraping the plates or dissolving them in sulphuric or hydrochloric acid. Electroplaters were at this time making use of zinc to recover gold and silver from cyanide solution.

Cyanide of potassium was also being used to a very limited extent at this time in the amalgamation of gold ores, being introduced into the stamp-mill mortar, grinding pan, or other crushing device with a rather vague idea that it would increase the amount of gold amalgamated. This it undoubtedly did by removing any film of grease or oxide surrounding the grains of gold and brightening them for easier amalgamation, but as it also may have caused a little gold to be dissolved and carried away in solution, it was poor practice. However, since it sometimes caused a lower tailing and any shortage of gold was not noticed, the use of cyanide of potassium in this way had some favor.

MacArthur-Forrest Process. — Though the attention of scientists and metallurgists had been drawn to the solvent action of cyanide compounds on gold and silver, resulting in considerable experimentation, the various experiments, studies, and patents can hardly be considered as a prologue to the discovery of the present cyanide process, or as having a direct bearing upon it.

In 1886 extended experiments were being carried out in Glasgow, Scotland, by J. S. MacArthur, R. W. Forrest, and W. Forrest, for the purpose of developing an incipient gold-extracting process. In the course of their experiments, tests were made with all the known solvents of gold, and it was found that a solution of cyanide of potassium gave a high extraction with a small consumption of the chemical. Their first application for patent covering the dissolving power of cyanide was made in Great Britain in 1887. The principal detail and then novel feature was the low strength of solution to be used. The discoverers next turned their attention to winning the gold from the solution, resulting in a patent being taken out on the use of zinc in a state of fine division, such as in the form of shavings or threads, for the precipitation of the gold from the solution. Their patents also included the use of caustic alkalis to neutralize the cyanide-destroying acidity of the ore, which involved no new idea or detail.

Following these experiments of MacArthur-Forrest, practical and successful applications of the process were made with surprising rapidity in all parts of the world. This was mainly due to the inherent virtue and applicability of the process and to the fact that the syndicate under whose direction MacArthur-Forrest developed the process, trained a force of chemists and sent them into the principal gold-bearing regions of the world. The first plant on a commercial scale was established at Karangahake, New Zealand, in 1889. In South Africa the first plant was installed near Johannesburg, Transvaal, in 1890. Both of these plants were established under the direction of the owners of the patents. The first application of the process in America was made at Mercur, Utah, as a result of experiments instigated by the reports of the success being attained in Africa and Australia.

Development of the Cyanide Process. — The first material treated by the cyanide process was mill tailing taken from the ponds or banks in which they had accumulated, and treated by the leaching process. Then quickly followed the direct-filling method of conducting the tailing flow from the mill to revolving distributors operating similar to a revolving garden sprinkler, and known from its inventors as the Butters and Mein distributor. This distributor was placed over a leaching vat and operated to fill the vat with sand containing some slime, the major portion of the slime overflowing the rim of the vat. It was found that the sand charge was not easily leached owing to the amount of slime, and this led to the double-treatment system in which the sand is transferred in a drained condition from the collecting vat to a leaching and final-treatment vat. This method was perfected on the South African Rand, and was followed by the development there between 1894 and 1896 of the decantation process of slime treatment by J. R. Williams. In this process the settled and dewatered slime is diluted and agitated with several times its weight of cyanide solution until the gold and silver are dissolved, when the slime is allowed to settle and the supernatant clear, rich solution siphoned off; after which the slime is washed free of the dissolved metals by being again diluted, agitated, settled, and the clear solution drawn off, these washings being continued as long as profitable.

About 1898 the filter-press method of slime treatment was introduced in Australia by Sutherland, where it has been extensively and very successfully used. In America the filter press was used to only a limited extent up to the introduction of the leaf or vacuum filter, previous to which slime treatment was mainly by the decantation process. Filter presses, with the exception of the Merrill type, have fallen into disuse in America since the introduction of the vacuum filter. In the standard filter-press method the slime, after being agitated in cyanide solution until the precious metals are dissolved, is forced into a plate-and-frame filter press. After the press is full of slime, the dissolved metals are washed out of the slime by water or solution under pressure, when the press is opened and the cakes or plates of slime are dropped into a car or sluice for the waste dump.

The first practical vacuum or suction filter, often called the leaf filter, was devised in the United States by Moore in 1903. In this method a leaf consisting of a flat canvas slip or pocket stretched over a suitable frame is immersed in the slime pulp and a suction applied to the interior of the leaf, causing the slime to be drawn against it, and the solution within. This action induces a leathery coating or cake, one-half to three inches thick, to form, when the leaf with its cake is separated from the excess pulp and brought into contact with a wash solution or water, which washes the dissolved metal out of the cake by being drawn by the suction or vacuum through the slime cake into the interior of the leaf, to run into a suitable tank to which the leaf is connected; after which the wash solution is removed and the cake is discharged. In some of these filters the cake is formed and washed by direct mechanical pressure on the pulp and wash solution, and not through atmospheric pressure by the production of a suction or vacuum in the interior of the leaf. The best-known vacuum and pressure leaf filters are the Butters, Moore, Kelly, Burt, Ridgeway, and Oliver.

The development and use of leaf filters has marked a period in which fine-grinding and all-sliming of ore, crushing in cyanide solution, and the treatment of silver and other ores heretofore giving a low extraction has been rapidly developed, especially in America. Fine-grinding had to some extent been practiced in connection with the use of the plate-and-frame filter press, the tube mill having been introduced by Diehl and by Sutherland in Australia in 1896 and 1898. Crushing in solution had been carried on with questionable success at a few plants since early in the history of the cyanide process, the first attempt being by Paul in northern California in 1891, and later by others in the Black Hills. But it is only since the introduction of leaf filters that fine-grinding and crushing in solution has become generally practicable and desirable.

Dry-crushing for cyanidation was rapidly developed during the early days of the process, but, since the introduction of leaf filters and fine-grinding, has fallen into disuse, except where coarse crushing is permissible. Roasting as a preliminary to cyaniding has practically disappeared, except for sulphotelluride ores, to which it was first applied in 1895. The discovery that bromine together with cyanide as a bromocyanide was a more active solvent or a supersolvent of the precious metals, and its application, date from 1892. It has been extensively and successfully used in the treatment of sulphotelluride ores in Australia, but its use elsewhere has been almost unknown.

The electrical precipitation of the dissolved metals from cyanide solution was introduced in South Africa in 1893, and great importance was attached to it. However, it has been almost entirely abandoned, though used in a few isolated installations to-day. Electrical precipitation without removing the solution from the pulp has never been a commercial success.

The use of zinc dust as a precipitant in place of shavings dates from 1894. It has been in favor in many large plants, but only during the last few years has there been a tendency to consider it preferable to the standard zinc shavings, and then mainly in America.

Some of the early experiments made by MacArthur-Forrest, and of the first work in actual practice, were done upon sulphide or mill concentrate, but the cyanidation of this material may be said to be only partly developed, except as the general improvements of the cyanide process have been applied. It is a fertile field for improvement.

The fundamental chemistry of cyanidation was fairly well worked out, considering the numerous and complex reactions that take place, during the early years of the process. The investigation of the chemical side of cyanidation has been comparatively slow during the latter years, due to the high technical and scientific ability requisite in doing such work, and to the fact that investigations into the physical and mechanical side have been much more profitable individually, and have so completely occupied the time and attention of investigators and operators that time for scientific research has not been available.

The physical and mechanical side of cyanidation has been in a state of continuous development since the first introduction of the process, and the field is now wider and better than ever. The proof of this is to be seen in the widely varying methods of leaching, agitating, filtering, and other details in the same locality, and the still wider variations in the different gold-silver regions of the world, also in the constant introduction of new devices. The field of cyanidation has been and is constantly widening through its encroachment upon amalgamation, concentration, and smelting.

CHAPTER II

NATURE AND PROPERTIES OF CYANIDE

Definition of Cyanide and Cyanogen. — *Cyanogen* is the compound radical CN, the carbon (C) and nitrogen (N) constituents of hydrocyanic acid (HCN), which is composed of hydrogen (H), carbon, and nitrogen, and is often called prussic acid. *Cyanide* is a compound of the cyanogen radical CN with usually a metallic substance, as potassium (K) or sodium (Na), forming potassium cyanide (KCN) or sodium cyanide (NaCN). A radical in chemistry may refer to a single element, in which case it is a simple radical, but more often refers to a group of two or more elements, which, once united, thereafter combine in chemical union or break the chemical bonds with other elements or compounds as if they were a single element incapable of being disassociated into two or more elements. The radical cyanogen (CN), or cyanide radical, is composed of one atom of carbon (C) and one of nitrogen (N), and in all the phases of the cyanide process and its chemistry this chemical union is never broken. To do so would be to lose the solvent action on the metals, for neither carbon nor nitrogen has any such dissolving effect. Neither is there much tendency for the two elements to disassociate. While the chemical symbol for cyanogen, or the cyanide radical, is CN, it has become a custom to write it Cy, a contraction of cyanide.

Properties and Reactions of Cyanide and Cyanogen. — Cyanogen is a colorless gas and does not exist free to any extent, consequently it must be fixed by being combined with a metal or other substance to hold it. It is a most active radical, especially in combining with the metals, with which it forms several hundred compounds, thus increasing the difficulty in isolating and determining the properties of each, more especially under working conditions, — a fact that has hindered the investigations of the chemistry of the cyanide process. Cyanogen, or the cyanide radical, is related to the cyanides as chlorine is related to the chlorides, and iodine to the iodides. As the acid radical SO_4 of sulphuric acid (H_2SO_4) unites with iron (Fe) to form an iron sulphate ($FeSO_4$), and the acid radical Cl of hydrochloric acid (HCl) unites with iron to form an iron chloride ($FeCl_2$), so does the cyanide radical CN unite with iron to form primarily an iron cyanide ($Fe(CN)_2$), and similarly with other metals. The result of the chemical combination of the radical CN with a base or metal is to form a salt, such as potassium cyanide (KCN or KCy) or sodium cyanide (NaCN or NaCy). The chemical principles involved in the formation of common table salt, sodium chloride (NaCl), are the same as those involved in the formation of sodium cyanide (NaCN).

Cyanogen combines to form simple or single cyanides, which may be regarded as metals replacing the H of HCN, as:

$$HCN + K = KCN + H;$$

and to form double cyanides which may be considered as a combination of two single cyanides, as:

$$Zn(CN)_2 + 2\,KCN = K_2Zn(CN)_4,$$

in which the zinc cyanide ($Zn(CN)_2$) first formed between zinc and cyanide and the potassium cyanide are the single cyanides, and the potassium zinc cyanide ($K_2Zn(CN)_4$) finally formed is the double cyanide. Other and more complex cyanogen compounds form and are found under working conditions.

The metal or base with which cyanogen is combined to form a cyanide is easily replaced by one for which cyanogen has a greater affinity. Thus in a KCN solution the K is replaced by gold (Au), forming the simple cyanide AuCN, and finally the double cyanide $KAu(CN)_2$, because cyanogen has a greater affinity for gold than for potassium (K). When the solution containing a gold cyanide is brought into contact with zinc (Zn), the gold is replaced by the zinc owing to the greater affinity of cyanogen for zinc than for gold, the reaction being:

$$2\,KAu(CN)_2 + Zn = K_2Zn(CN)_4 + 2\,Au.$$

It is the action of this principle that makes the cyanide process for gold and silver extraction possible.

Cyanogen Used in the Cyanide Process. — The sources of cyanogen in the cyanide process are potassium cyanide and sodium cyanide, the simple cyanides of the alkaline metals potassium and sodium. There are other simple cyanides of the alkaline earths and metals, such as ammonium cyanide (NH_2CN), barium cyanide ($Ba(CN)_2$), calcium cyanide ($Ca(CN)_2$), magnesium cyanide ($Mg(CN)_2$), and strontium cyanide ($Sr(CN)_2$). These have solvent powers similar to those of potassium and sodium cyanide, but are not used in the ordinary cyanide process, mainly for economic reasons. The double cyanides have considerable solvent power in some cases, but are too stable and hold their cyanogen too firmly to be a source of it, except so far as it is possible to utilize that formed in working solutions. The other and complex cyanogen compounds have little or no dissolving effect.

Potassium cyanide is a white salt with the usual salty taste. It gives an alkaline reaction and is easily dissolved and very soluble in water. One part of boiling water will dissolve 1.2 parts of the salt. Exposed to the atmosphere, especially in the presence of moisture, there is a slight decomposition into hydrocyanic acid sufficient to give the characteristic odor similar to that of an almond or peach kernel and irritating to the mucous membrane. It is an irritant to the skin externally and a deadly poison internally. Sodium cyanide has almost identical properties. The salts are made by fusing nitrogenous substances, as horns, hoofs, dried blood, old leather, etc., with alkali and iron, followed by a refining or eliminating process, leaving the desired salt; or by synthetic processes fixing nitrogen from the atmosphere or ammonia by passing them over heated alkaline salt and carbon to form a union of the carbon, nitrogen, and sodium or potassium as sodium or potassium cyanide.

Difference between Sodium and Potassium Cyanide. — Practically the only difference between sodium and potassium cyanide, and in the main with the other simple alkaline cyanides, is the dissolving strength, which depends upon the amount of the CN radical. The atomic weight of potassium is 39.1, of sodium is 23, of carbon is 12, and of nitrogen is 14. Consequently the weight of a molecule of potassium cyanide is:

$$\overset{\text{K}}{39.1} + \overset{\text{C}}{12} + \overset{\text{N}}{14} = 65.1,$$

of which the cyanogen (CN) represents 26 of the total 65.1 parts by weight, which is 40 per cent or 40 parts CN in 100 of the salt. The weight of a molecule of sodium cyanide is in a similar way:

$$\overset{Na}{23} + \overset{C}{12} + \overset{N}{14} = 49,$$

of which the cyanogen represents 26 parts of the total 49 parts by weight, which is 53.06 per cent or 53.06 parts CN in 100 of the salt. If, in equal weights of the salt, potassium cyanide contains 40 parts CN and sodium cyanide contains 53.06, then the dissolving strength of sodium cyanide is 1.3265 times that of potassium, or it is 132.65 per cent strong when the pure potassium salt is considered as 100 per cent strength. This method of considering pure potassium cyanide as 100 per cent strong and marking all cyanide, whether potassium or sodium and pure or impure, according to its strength or amount of CN radical as compared with pure potassium cyanide at 100 per cent, is now in practice everywhere.

It is impossible to say which of the two salts is the better for use in cyanide work. Potassium cyanide was at first used entirely, apparently because it was the only salt available. In recent years sodium cyanide has been extensively used and has met with considerable favor at some plants, while others have found it unsatisfactory and have preferred to return to the use of potassium cyanide. Sodium cyanide, whether in the solid form or in solution, appears to be less stable and consequently to decompose faster than potassium cyanide. In wet climates it absorbs moisture faster and gives some trouble in this way through deliquescing. Its base, sodium, forms more soluble compounds than the potassium of potassium cyanide, and may give trouble by precipitating them in the zinc boxes. Commercial cyanide is generally far from being pure, owing to alkaline constituents that are introduced in the process of manufacture. Potassium cyanide may often contain considerable of the stronger sodium cyanide, introduced for the purpose of bringing it up to the branded strength. The effect of these impurities, like any difference between pure potassium and sodium cyanide, is not well understood, but is being studied, and probably will result in requiring cyanide of a certain purity and composition. This will be an improvement of the method in the past of purchasing the most economical salt as determined by calculations based on the branded strength and the cost of the cyanogen (CN) delivered at the plant; the higher strength salt often being more economical owing to the indirect saving in transportation.

CHAPTER III

DISSOLUTION OF GOLD AND SILVER

Reactions. — It is generally accepted that gold is dissolved by a cyanide solution in accordance with the equation first brought to public attention by Elsner and known as Elsner's equation:

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

the gold (Au) combining with potassium cyanide (KCN), oxygen (O), and water (H_2O) to form a gold potassium cyanide ($KAu(CN)_2$) and caustic potash (KOH). The simple gold cyanide (Au(CN)) is probably first formed to be changed into the double gold cyanide ($KAu(CN)_2$), as:

$$2\,Au + 2\,KCN + O + H_2O = 2\,Au(CN) + 2\,KOH.$$

$$Au(CN) + KCN = KAu(CN)_2.$$

Silver is dissolved in a way similar to gold, as:

$$2\,Ag + 4\,KCN + O + H_2O = 2\,KAg(CN)_2 + 2\,KOH.$$

One part of potassium cyanide should dissolve 1.51 parts of gold or .83 part of silver according to the above formulæ.

Necessity and Source of Oxygen. — It is seen from the above equations that oxygen is necessary in dissolving gold and silver. This has been confirmed in experiments and practice. The necessary oxygen is supplied by the air or oxygen which the solution has absorbed through being exposed to the atmosphere, and by that absorbed or held by the ore itself. Oxygen may also be supplied by pumping air through the charge or solution, or by chemical oxidizers. However, it has been abundantly proven in practice that the attempt to supply oxygen artificially soon reaches a point where it is uneconomical. Consequently the necessary supply of oxygen is relied upon to be had by the use of a sufficiently large volume of freshly precipitated and aerated solution, by aerating the ore through draining and drawing the atmosphere into the interstices between the grains of pulp, by bringing the pulp into contact with the atmosphere when agitating, by agitating with compressed air, and in exceptional cases by pumping or drawing air through the charge. The necessity of providing much oxygen by stress on these means is small with a clean, gold ore, but increases with the quantity of sulphide or baseness of the ore, and with most silver ores, since the metallic compounds of these ores decompose or oxidize to form new compounds, thereby utilizing or abstracting the oxygen necessary in the dissolution process. The best-known chemical oxidizers that may be used are sodium peroxide (Na_2O_2), potassium permanganate ($KMnO_4$), and manganese dioxide (MnO_2). While these hasten the dissolution, they have never been found to be of economic value, as they invariably do not give any increased extraction over that which can be obtained by using a little more time or more aeration of the charge and solution.

Bromocyanide and Mercury Salts as Supersolvents. — The dissolving power of cyanide solution has been increased by the addition of chemicals which have something of an oxidizing effect, but act mainly through the liberation of cyanogen in a nascent state — ready and strongly desirous of uniting with a substance replacing the H of HCN. Bromine in connection with cyanide as bromocyanide is the only chemical that has been used to any extent for this purpose, and then only on telluride and sulphide ores that will not give a good extraction otherwise. Its use requires such care and expense that it is undesirable for treating ores from which a good extraction can be secured otherwise by the usual processes. The reactions that occur in the use of bromocyanide have never been solved, but the supersolvent qualities are presumed to be due to the liberation of nascent cyanogen and to some oxidizing effect, since the bromine does not enter into combination with the gold.

Mercurous (Hg_2Cl_2) or mercuric chloride ($HgCl_2$) has been added as a chemical in addition to the cyanide used. Its effect appears to be due to the affinity of mercury for cyanogen, forming a mercuric cyanide ($Hg(CN)_2$) or a double mercuric cyanide of potassium ($K_2Hg(CN)_4$), by decomposing such stable compounds as the ferrocyanides and ferricyanides ($K_4Fe(CN)_6$ and $K_3Fe(CN)_6$) in addition to the simple cyanides and easily-decomposed double cyanides, thus removing the interference of the ferrocyanides. The double mercuric cyanide dissolves gold and silver without requiring oxygen, as:

$$K_2Hg(CN)_4 + 2\,Au = 2\,KAu(CN)_2 + Hg.$$

From the silver sulphide the dissolution may proceed as:

$$K_2Hg(CN)_4 + Ag_2S = 2\,KAg(CN)_2 + HgS.$$

The mercuric sulphide (HgS) formed being stable and insoluble, and therefore not a detriment as the alkaline sulphide (K_2S) formed when the silver sulphide is acted upon by KCN, as:

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

The alkaline sulphide being an abstracter of oxygen unless altered into an insoluble and stable sulphide, as the mercuric sulphide (see Alkaline Sulphides and Sulphocyanides). The mercury salts have been used to a slight extent in this way in the working of sulphide silver ores.

Strength of Solution Required. — The strength of solution required varies. Experimental work in the laboratory will indicate the most advisable strength for starting a new ore or plant. But this strength is invariably reduced in the course of time, due to the desirableness of starting new operations with a sufficiently strong solution, and that a cyanide solution after being in use for some time is found to contain cyanogen compounds that are not apparent in the usual test for cyanide strength, but which have a direct or indirect solvent effect.

Experiments have shown that both pure gold and silver dissolve most rapidly in a solution of .25 per cent (5 pounds per ton of solution) KCN, that between .1 per cent (2 pounds) and .25 per cent (5 pounds) the dissolving rate is nearly constant, but grows less with solutions above or below these strengths. The lessened efficiency of cyanide solutions stronger than .25 per cent (5 pounds) KCN in the above experiments is ascribed to the fact that the amount of oxygen soluble in a solution grows less as its cyanide content becomes greater. When oxygen is supplied as needed, a stronger solution will dissolve the metals faster than a weak one, but the weaker the solution the more highly efficient equal amounts of cyanide will be, and the less will be the cyanide consumption per ton of ore treated or unit of precious metals dissolved.

The present practice in leaching an ore containing gold that is easily dissolved is to use about a .1 per cent (2 pounds) KCN solution, and seldom higher than .2 per cent (4 pounds); while in the agitation treatment of such ores, solutions from .05 per cent (1 pound) to .1 per cent (2 pounds) are generally used. On silver ores a .1 per cent (2 pounds) to .3 per cent (6 pounds) solution is usual in agitation, and from a .25 per cent (5 pounds) to .5 per cent (10 pounds) in leaching practice. For treating concentrate by agitation, solutions ranging from .15 per cent (3 pounds) to .4 per cent (8 pounds) are in use, and from .2 per cent (4 pounds) to .75 per cent (15 pounds) for leaching. To enable the stronger solutions to act most efficiently and get the increased advantage of their higher strength, it is necessary to supply plenty of oxygen, which the method of operating does by aerating the solution and ore frequently.

The use of a stronger solution causes a higher consumption of cyanide mainly by the increased effect of the stronger solution upon the base metals and cyanicides. This action upon the base metals is slow, and while they dissolve according to their slower rate of solubility at the time the gold and silver is being dissolved, their dissolution continues as vigorously after the comparatively quick dissolution of gold and silver is made; consequently the strong or dissolving solution should be withdrawn as soon as the precious metals are dissolved, and the weak solutions used thereafter for washing. The so-called " selective action " of cyanide in dissolving the precious metals is not a true selection of these in preference to the base metals, but is due to the somewhat quicker dissolution of the precious metals under equal conditions, and to the fine state of division and small amount of them in comparison with the base metals — conditions which allow comparatively quick and easy dissolution of the precious metals.

Cyanicides. — The solubility of gold and silver in cyanide solutions is reduced by the presence of " cyanicides " in the ore. A " cyanicide " is any substance outside of the precious metals and those involved in the working of the process — as the zinc — that will unite chemically with the cyanide or tend to decompose it, thereby destroying the cyanide or rendering it inert for dissolving purposes. Such substances may be a metal, as copper (Cu) or iron (Fe) when in a condition to be acted upon by a cyanide solution, such as in the form of a salt, producing in the case of copper a potassium cuprous cyanide ($K_2Cu_2(CN)_4$) or in the case of iron a potassium ferrocyanide ($K_4Fe(CN)_6$). Acids are active cyanicides, forming hydrocyanic acid (HCN) with the cyanogen. Cyanicides hinder the solubility of gold and silver in two ways: first, by destroying or neutralizing the cyanide so that it is not available for dissolving the gold and silver; and second, by going into solution to such an extent that the solution becomes inactive towards the dissolution or precipitation of gold and silver, in which condition it is said to be "foul." The action of cyanicides is met by removing them from the ore by concentration, water-washing, etc.; by neutralizing them into inert salts by the use of the proper quantity of lime, etc., — including aeration — introduced into the ore or solution; by the passage of the solution through the zinc boxes, which often appears to cleanse it of the influences which retard its dissolving effect; and by keeping the solution so low in cyanide strength that it will, by its greater affinity or dissolving influence on gold and silver, — its selective action towards them — dissolve these metals and leave the base metals and cyanicides unacted upon as much as possible.

Heating the Solution. — Laboratory experiments often indicate that a higher extraction can be obtained by a heated solution than a cold one. In practice it has generally been impossible to notice any difference between the normal extraction and that made by heating the ore and solution, or that obtained during the heat of summer or the frigid weather of winter. A few cases have been reported in which some virtue has been found in heating the solution. However, it is the more general experience that no additional extraction is obtained, or at least nothing sufficient to warrant the cost of heating the solution and the additional consumption of cyanide due to its decomposition, and the increased decomposition and resulting activity of the base metals and cyanicides; though the decomposition of the base metals, compounds, and alloys is beneficial in liberating the precious metals, that they may be more easily dissolved. One reason that militates against the use of a heated solution is that as the solution is heated it is unable to retain the oxygen dissolved in it.

Effect of Size of Metal Particles. — The size and shape of the particles of gold and silver have an important influence on their rate of solubility. Where the metal is in very fine particles, it will be quickly dissolved and require only a weak solution to get the maximum dissolution within a reasonable length of time. When the particles are large, a larger surface is exposed to the action of the solution, causing a large amount of metal to go into solution in a given time. Presuming that the metal is in spheres, the solution is constantly removing a film of metal and reducing the spheres to smaller diameters, consequently larger spheres or metal in thick particles will require considerable time for dissolution over that necessary when the same amount of metal exists in thin plates or in a larger number of smaller particles. In practice, ore containing the metal in a comparatively coarse state, not removed by amalgamation or concentration, needs to be treated with a strong dissolving solution to reduce the time of dissolution to that of an ore containing finely-divided metal treated with a weak and more slowly solvent solution.

Effect of Crushing and Form in which Metal is Held. — The crushing must either liberate the gold and silver from its matrix or expose a face of it, that the cyanide solution may act upon it. Though with porous ore, that the solution can penetrate and be withdrawn from, it is not so essential that the precious metal be liberated. Following the above principle, various factors affect the degree of fineness to which the ore must be crushed. Where the gold is finely divided it naturally follows that fine crushing will be required to liberate it, more especially with hard dense ores, which the solution cannot penetrate. When the gold is comparatively coarse, a face may be exposed by coarse crushing which will allow dissolution to continue slowly inward, until the entire amount of metal is eaten out. Sulphides especially require fine grinding to liberate the gold they mechanically hold. Many ores contain the gold on the breaking or parting planes, from which it is liberated in the crushing process which naturally splits the grains along these planes, or the solution easily penetrates the fractures. Fine-grinding, especially of a sliming nature, besides liberating the particles of precious metal, also breaks them up or hammers them into thin particles so that they are quickly dissolved by cyanide. Whereas, by a system of crushing that only liberates the metal, considerable time for dissolution would be required by the larger particles. Sliming in solution is especially efficacious, for the metals are ground fine in a large volume of well-aerated solution under an agitation that is very favorable for causing the metals to go into solution. When the metals exist as compounds, as gold combined with tellurium, and silver with sulphur, chlorine, etc., fine-grinding is necessary to get the highest efficiency of the cyanide solution in attacking the compounds and removing the precious metals, it being very much on the same principle as gold and silver mechanically held or more or less alloyed with base metals.

Some ores may only require crushing to $\frac{1}{4}$- to $\frac{1}{2}$-inch cubes, such as low-grade, porous, friable ores. Others may give good extractions only when crushed to a 200-mesh or even finer. The rule may be given that the harder, denser, higher-grade, more sulphuretted, and baser the ore is, the finer-grained and more tightly held mechanically, combined, or alloyed the gold and silver is, the finer will be the crushing required. That the softer, more leachable, oxidized, friable, porous, less base, and lower grade the ore is, the coarser and freer that the gold and silver is mechanically and otherwise held, the coarser will be the crushing permissible to obtain an economic extraction. This rule is subject to the effect the solution will have upon the cyanicides, the cost of crushing and grinding, and the trouble or expense and the efficiency in handling the slime, all of which increase with finer crushing.

Volume of Solution Required and Method of Application. — The volume of solution and method of application should be such that the solution at all times contains sufficient oxygen and cyanide strength for efficient dissolution. An ore containing cyanicides that destroy the cyanide or coarse metal that consumes it in the dissolving process will require leaching or agitation with a large volume of solution or, less preferably, the addition of more cyanide during the agitation; that a solution of reasonable dissolving strength may be always available about each particle of metal, and yet that there be not the loss due to the use of an inordinately high cyanide strength. An ore containing reducing or deoxidizing agents rapidly fouls the solution towards dissolving the metals by abstracting the necessary oxygen, to remedy which the charge must be aerated or, as is sometimes more convenient and desirable, the fouled solution is replaced by a freshly-aerated one. Where the cyanicides and reducing agents exert themselves in this way, a large volume of solution is required and is applied by being continuously leached through the percolation charge, or by a large amount of solution in comparison to the dry pulp in an agitation charge, or by replacing with fresh solution through settling the pulp and decanting off the old solution. The distinction between that solution required to dissolve the metals and that required to wash these dissolved metals out of the ore must be clearly borne in mind, though in practice the functions of both may be considered as more or less united. With an ore containing easily dissolved gold treated by percolation, $\frac{1}{3}$ of a ton of solution may dissolve the metals and $\frac{2}{3}$ of a ton may wash them from a ton of the ore. Other ores in which the metal is slowly dissolved, as those of silver, may require many tons applied continuously, so that dissolution and washing continue together until no longer profitable.

Time Required for Dissolution. — The time required for dissolving the gold and silver depends upon the nature of the ore and its treatment; the size and thickness of the metal particles; the mechanically-held, chemically-combined, and metallically-alloyed condition of the metals; the action of cyanicides and reducers; the cyanide strength of solution; and the volume of solution as referring to keeping a solution that is an active dissolver always in contact with the ore. With some ores containing extremely fine gold practically all the dissolvable gold will be in solution by the time the pulp leaves the tube mill, when crushing and sliming in cyanide solution. Where the metal is in coarse, thick particles, chemically combined or mechanically covered or alloyed, or only a small part of its area or cross-section is exposed to the cyanide activity, the dissolution must be comparatively slow. The presence of cyanicides consuming the cyanide and oxygen render the dissolution slow, owing to the inability to get these to the dissolving metal as fast as needed, or to their more or less complete destruction. Weak solutions are more slowly solvent than strong ones in the presence of sufficient oxygen, consequently:

Maximum dissolution = Strength of solution × Dissolution period.

In which either or both of the factors, "strength of solution" or "dissolution period," may be varied within certain limits to produce a corresponding result in their products — the "maximum dissolution" or "rate of dissolution." That is, a strong solution will dissolve the same amount of metal as a weak one in a less length of time, but a weaker solution given a longer contact will generally dissolve the same amount at a slower rate than a strong one. However, this rule is subject to the chemical law of mass action.

Gold ores will usually require a contact of 12 hours to 3 days for dissolution of the gold by leaching, or 3 to 18 hours by agitation. Silver ores may require a contact of 4 to 12 days by leaching, or 18 hours to 3 days by agitation. Concentrate may require a contact of 10 to 30 days by leaching, or 12 hours to 10 days by agitation. The fineness to which the ore is ground, outside of leaching *vs.* agitation, is usually the most important factor in the rate of dissolution in practice for reasons that have been referred to.

Silver Ores. — Silver ores require to be treated with stronger solutions and for a greater length of time than gold ores, for the metal in a gold ore is usually as native gold or a gold-silver alloy in small particles, while the silver in a silver ore is more often as a compound with sulphur, bromine, chlorine, antimony, arsenic, etc., usually the first. When it occurs as native silver the mass is relatively large as compared with the mass of the gold of a gold ore, and in all cases the mass or amount of the silver is comparatively large. It has also been shown that the rate of solubility of silver is about two-thirds that of gold, while almost double the amount of cyanide is required in the formation of the silver potassium cyanide as the gold potassium cyanide.

The solution of silver from the silver sulphide (Ag_2S) — the principal silver mineral worked by cyanidation — proceeds through the decomposition of the sulphide by cyanide, as:

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

In which 1 part of cyanide dissolves .414 parts of silver, or one-half as much as of native silver and about one-quarter as much as of native gold. Unfortunately the soluble or alkaline sulphide (K_2S) formed is not stable, and in a weak solution the silver sulphide may not be decomposed, or if it is the alkaline sulphide may tend, especially if the solution becomes weakened, to reverse the reaction by which it was formed and reprecipitate the silver as a sulphide. Consequently the solution must be strong to decompose the sulphide and hold the silver in a dissolved condition. The alkaline sulphide if not precipitated as an insoluble sulphide by some metal in solution, as lead, zinc, or mercury, will unite with the cyanide and with oxygen to form a sulphocyanide or thiocyanate, or will be oxidized into a sulphate, which is one of the reasons for using a large volume of freshly-aerated solution in the treatment of silver ores. The subject of alkaline sulphides is more fully treated under Alkaline Sulphides and Sulphocyanides.

The dissolution of silver from the other ores must necessarily be slow and consume much cyanide and oxygen, owing to the necessity of breaking down the silver compounds in removing the silver from them.

CHAPTER IV

SUITABILITY OF AN ORE FOR CYANIDATION

It may be stated tentatively that all ores of gold and most silver ores can be cyanided. Modifications and special treatment may be required with refractory ores, but those ores for which some successful treatment system cannot be devised are few. The refractory qualities of an ore toward cyanide are of two kinds, physical and chemical. What may be termed refractory physical qualities refer to the necessity of crushing the ore extremely fine to liberate the metal to the solvent action of cyanide solution, and to the trouble encountered in treating a very slimy material; these will be considered elsewhere. To speak of an ore as being refractory to treatment generally refers to its chemical or physical-chemical nature whereby the precious metals cannot be economically dissolved owing to their being chemically combined or mechanically alloyed with some substance that prevents the metals from being dissolved by cyanide, also to some constituent of the ore that combines with the cyanide in large quantities or may retard the dissolution of the precious metals from the ore, or their precipitation from the solution. These undesirable constituents are spoken of as " interfering substances." The results of an analysis of an ore containing interfering substances or refractory compounds will not indicate whether it can be cyanided or not, for a quantity of mineral or element which may not interfere in one ore, may, in a somewhat different form in another, partly or completely prevent successful treatment. With what success an ore can be cyanided can only be judged and determined by laboratory tests, preferably followed by small working tests, which will be discussed in a later chapter.

Classification of Ores. — Ores may be divided into three classes. Clean ores having no sulphides or base metals and their compounds; base ores containing unaltered sulphides or base-metal compounds; and oxidized ores as a result of the decomposition of original base or sulphide ores. Clean ores are easily treated, as they consume but little cyanide, do not introduce fouling substances into the solution, and the precious metals readily dissolve. Ores containing fresh, unaltered sulphides or base metal compounds cyanide less easily, depending partly upon the tendency of the sulphides and compounds to alter and decompose, thus consuming cyanide and oxygen, and introducing substances that may foul the solution toward further dissolution of the gold and silver and its precipitation; also upon the ability of the cyanide to unlock the mechanical or chemical combination in which the gold and silver is held by the base metals. Oxidized ores act similarly to decomposing base ores, with the exception that their more complete decomposition has allowed a large part of the troublesome constituents to be removed, though that which remains is in a better condition to be acted upon by cyanide and has better liberated the precious metals for easy dissolution. Chlorides, carbonates, and partially oxidized ores, as representing an intermediate or final stage in the transition of unaltered sulphides to fully oxidized or weathered ores, are the hardest to treat, owing to the activity with which cyanide attacks these soft and easily acted upon base metal compounds.

Iron. — Metallic iron is so much less soluble than gold and silver in cyanide solution and dissolves so slowly that the fine iron and steel introduced into the pulp in the process of crushing and grinding has no noticeable deleterious effects, but many of the iron compounds are readily acted upon. Limonite ($2\,Fe_2O_3.3\,H_2O$), the soft, yellow iron oxide, which is found in oxidized ores as one of the final stages in the oxidation of iron, is practically unacted upon by cyanide solution, though it causes trouble mechanically through its tendency to slime. Pyrite, the brass-colored, cubical iron sulphide and the one most generally found, is slowly acted upon by cyanide, but does not easily decompose. Marcasite, the white iron pyrite, which is also quite common, is much less soluble in cyanide solution than pyrite, but readily decomposes. Metallic iron never enters the solution sufficiently to be harmful, neither does metallic iron oxidize to an extent that is injurious in ordinary cases. It is the products of the decomposition of the sulphides, of which iron is one of the principal constituents, that are harmful; these are ferrous sulphate and oxide, and sulphuric acid, consuming the cyanide through the iron entering the solution as a potassium ferrocyanide

($K_4Fe(CN)_6$) and the cyanide being neutralized by the acid. The interference is met by the use of water washes removing the soluble compounds, and the neutralizing properties of lime and alkalinity as more fully given under Alkalinity and Lime.

Sulphur. — Sulphur does not react directly with cyanide, but, as the principal constituent of the sulphides, it forms alkaline sulphides when the metallic sulphides are decomposed by cyanide or alkali. The alkaline sulphides react with cyanide and oxygen to form sulphocyanides or thiocyanates, or with oxygen to form sulphates, the abstracting of the oxygen being especially harmful. They also precipitate silver under certain conditions and possibly gold. The interference is met by precipitating the soluble alkaline sulphides by means of zinc, lead, or mercury into inert sulphides. The subject is fully treated under Alkaline Sulphides and Sulphocyanides.

Copper. — Cyanide has a great affinity for copper and will strongly act upon metallic copper, especially when finely divided. Copper compounds in a physically hard state, such as unoxidized pyrite or sulphide, are more slowly acted upon. When the compounds are soft, porous, and spongy, the dissolution is fast, resulting in an excessive consumption of cyanide, a precipitation of copper in the zinc boxes, and, with much copper in solution, trouble in dissolving and precipitating the gold and silver. Chalcopyrite, the principal copper sulphide found in gold ores, gives but little trouble when unoxidized. Malachite and azurite, the carbonates of copper, are easily dissolved and a very small amount will give trouble. An assay of the copper content of the ore will not indicate if it can be profitably worked, for the extent of the action of cyanide depends mainly upon the form or state the copper is in. In some cases, as with oxides and carbonates, a few pounds of copper per ton of ore may prohibit cyaniding by ordinary methods, whereas an ore containing several per cent of unaltered chalcopyrite may perhaps be amenable to treatment. Modifications for treating copper ores involve the use of low strengths of solution and special methods or treatment of precipitation, the use of ammonia as a preliminary dissolver of the copper or to give a greater selective action to the cyanide solution on the gold, and the preliminary removal of the copper by concentration or a dilute acid wash.

Lead. — Lead is strongly acted upon by cyanide, but galena (PbS), the sulphide of lead, is only slightly attacked and very slowly dissolved. The cyanidation of ores containing galena is generally attended with no excessive consumption of cyanide or any trouble except that which but seldom occurs in the zinc boxes from the precipitation of a large amount of lead. Finely-ground concentrate containing 36 per cent of galena has been successfully cyanided.

Arsenic. — Arsenic is very slightly acted upon by cyanide, but may sometimes be found in small quantities in the zinc precipitate resulting from the treatment of arsenical ores. Mispickel or arsenopyrite ($FeS_2.FeAs_2$), containing 46 per cent of arsenic, a silver-white to steel-gray pyrite, is abundant in gold ores. Concentrate containing 65 to 72 per cent of mispickel has been successfully cyanided. Realgar (AsS), the sulphide of arsenic, containing 70 per cent of arsenic, as a prominent constituent of ore has been successfully cyanided, as at Mercur, Utah. Arsenical ores in most cases give a good extraction by ordinary methods, in others they require fine-grinding, the decomposing effect of strongly alkaline solutions, roasting, or the increased dissolving effect of bromocyanide. In some cases the decomposition of the arsenical pyrite or sulphide causes a high consumption of cyanide. In all cases there is a strong reducing action through the formation of alkaline sulphides which must be met by aerating the solution and charge, and in most cases by treatment for the removal of the alkaline sulphides by other means. Arsenical ores have a reputation of being refractory, but in most cases are amenable to successful treatment.

Antimony. — Ores containing antimony are generally hard and in some cases impossible to treat. Antimony does not appear to react with cyanide, but the decomposition of stibnite (Sb_2S_3), the sulphide of antimony, decomposes cyanide, probably by forming alkaline sulphides and the resulting sulphocyanides, and hydrocyanic acid. There is a reducing action similar to that with arsenical ores, but so pronounced as to make it impossible to treat the ores in some cases. With stibnite the precious metals appear to be firmly held by the antimony so that they cannot be dissolved out. Strongly alkaline solutions will decompose the antimony to allow the precious metals to be better liberated, but may increase the reducing or deoxidizing power and cyanide-consuming tendencies of the ore to such an extent as to prohibit cyanide treatment. Preliminary treatment with hot alkaline solutions has been suggested. Roasting has been very successful. Fine-grinding will better liberate the value, but cannot remove the evil effects on the solution.

Tellurium. — Tellurium does not react with cyanide or only slightly in alkaline solutions, but holds the gold in a chemical combination that is unaffected by cyanide under ordinary conditions, so that the usual processes give little extraction. Fine-grinding with the increased solvent action of bromocyanide is used in Australia to extract the precious metals. The method in America has been to break the combination between the precious metals and the tellurium by roasting, which volatilizes or oxidizes the tellurium, allowing the gold and silver to be readily dissolved, but new chemical methods analogous to the use of bromocyanide have been introduced.

Mercury and Cinnabar. — Metallic mercury is so slowly dissolved by cyanide that it introduces no difficulties into amalgamating in cyanide solution, or treating tailing containing metallic mercury. The mercury in old tailing is generally altered to an oxide or chloride which is easily dissolved to be precipitated in the zinc boxes like that dissolved when amalgamating in solution. The dissolution of a small amount of mercury is beneficial in precipitating the obnoxious soluble or alkaline sulphides as an insoluble and inert sulphide of mercury (HgS), and in forming a galvanic couple with the zinc in the zinc boxes which assists the precipitation.

Cinnabar (HgS), the sulphide of mercury, is practically the only mineral of mercury found in ores of the precious metals, and may be said to be unaffected by cyanide. In fact, mercury by special treatment and gold by the usual cyanide treatment have been produced from the same ore, the production of mercury ceasing as its percentage fell, but the cyaniding for gold continuing.

Zinc. — Metallic zinc is more easily dissolved than gold, but sphalerite or zinc blende (ZnS), the sulphide of zinc containing 67 per cent of zinc, is probably less acted upon by cyanide than iron pyrite. The products and compounds formed by the oxidation or decomposition of zinc blende, as the carbonate and oxide, act similarly to other decomposing sulphides and their products for whose metallic base cyanogen has a strong affinity, consuming much cyanide and alkali.

Nickel and Cobalt. — Nickel and cobalt have an effect in cyanide treatment similar to that of copper.

Manganese. — Cyanide reacts with manganese and while it may increase the cyanide consumption, it does not appear to harm the solution, being removed as an insoluble compound by oxidation or the use of lime. Manganese in silver ores usually prevents a good extraction by locking up the silver in a manganese compound so as to prevent the solution from dissolving it. With the silver in a soluble form and the manganese separate from and uncombined with the silver, there is no difficulty in getting a good extraction and no fouling of the solution. No practical method of unlocking the silver from the manganese compound has yet been devised, but it would appear that fine-grinding, roasting, the use of a supersolvent as bromocyanide, or a treatment by acid or alkali to decompose and break up the compound are the points from which experiments should be started. Treatment with a weak acid solution preliminary to cyaniding has given excellent results in laboratory experiments.

Carbon and Carbonaceous Matter. — Vegetable matter, charcoal, and other organic material in an ore or tailing give trouble both in the dissolution and in the precipitation, owing to their deoxidizing or reducing power, their cyanide-consuming properties, their tendency to reprecipitate the metals in the ore, and to a too vigorous evolution of hydrogen in the zinc boxes. Considerable alkalinity and aeration should be used when they cannot be removed. Lime itself may be the source of the trouble when carrying carbon introduced in the lime-burning process. Graphite gives trouble mechanically by causing a scum or froth and chemically by reprecipitating the metals. The remedy appears to be roasting, which may not always be successful, drying and agitating in that condition with air, or by sun drying and weathering.

Aluminum and Magnesium. — These metals as sulphates resulting from the oxidation and decomposition of sulphides may interfere by uniting with the cyanide to form hydrocyanic acid, as:

$$MgSO_4 + 2\,KCN + 2\,H_2O = Mg(OH)_2 + K_2SO_4 + 2\,HCN.$$

These metals may also be dissolved and precipitated in the zinc boxes. The presence of sufficient lime or alkalinity will remedy this, as:

$$MgSO_4 + Ca(OH)_2 = Mg(OH)_2 + CaSO_4.$$

The calcium sulphate ($CaSO_4$) and magnesium hydrate ($Mg(OH)_2$) are harmless and insoluble.

Silver. — The amenability of silver ores to cyanide treatment varies with the nature of the mineral or the silver compound. Argentite (Ag_2S), the silver sulphide; cerargyrite ($AgCl$), the silver chloride; bromyrite ($AgBr$), the bromide; embolite ($Ag(BrCl)$), the chlorobromide; and native silver when in a fine state of division, are readily acted upon and dissolved by cyanide and give a good extraction. Proustite (Ag_3AsS_3), the light-red ruby silver; pyrargyrite (Ag_3SbS_3), the dark-red ruby silver; and stephanite (Ag_5SbS_4), the brittle silver, do not so readily respond to cyanide treatment; these are the arsenical and antimonial ores of silver. Coarse native silver and that in galena (PbS), the sulphide of lead; in tetrahedrite ($Cu_8Sb_2S_7$), the gray copper sulphide; and in sphalerite or zinc blende (ZnS), the sulphide of zinc, cannot be successfully treated by the cyanide process. Ores containing manganese, in which the silver is locked with the manganese in an embrace that cannot be broken by cyanide, are not amenable to treatment. The treatment of refractory silver ores includes fine-grinding to liberate the precious metals and allow the mineral to be attacked and decomposed in general; in the use of a strong alkalinity in the cyanide solutions to assist in decomposing the sulphides and metallic compounds; and in the use of lead and mercury salts to precipitate the alkaline sulphides.

CHAPTER V

CHEMISTRY OF CYANIDE SOLUTIONS

Classification of Cyanide Tests. — The tests and experiments made in connection with a cyanide plant may be divided into three classes:

(1) Those made daily by the man on shift, which consist of titrating the solution for its strength in "free cyanide" and possibly in "total cyanide," and its alkaline strength in "protective alkalinity," as a guide to the amount of cyanide to be added for dissolving purposes, and the amount of lime for neutralizing.

(2) Those made daily by the assayer, which consist of assays of the ore before and after treatment, and of the solutions used in treatment, also of the slag, matte, and bullion produced. These are made for the purpose of determining if the routine of treatment is being carried out with maximum efficiency, and for checking the bullion returns.

(3) Those made by or under the direction of the chemist or superintendent of the plant, which consist of laboratory extraction and sizing tests on the ore, and a determination and investigation of certain or most of the constituents of the ore, solution, etc. These are made for the purpose of studying the ore and its treatment with a view to increasing the extraction, decreasing the costs, and meeting some unsolved problem or any difficulties that may arise unusual to the daily routine. The nature and extent of these experiments will vary with each plant and operator. The education, training, and activity of the operator is the principal determining factor, the necessity of investigation usually coming second. Because a plant operates smoothly and an ore treats easily is no reason for not making a thorough chemical and physical investigation, for it is the facts brought out by such an investigation, and corroborated from time to time by renewed and further investigation, that enable one to operate a plant successfully and quickly locate and remedy trouble when it arises.

The analytical chemistry of cyanide solutions is an exhaustive subject, due to the numerous and complex reactions that occur and their influence when working on mill solutions, which always contain a multitude of complex substances. The methods to be given are the standard and more simple methods in general use and adapted to ordinary mill practice, for further methods involving a thorough familiarity with analytical chemistry, the student is referred to "Clennell's Chemistry of Cyanide Solutions," second edition, and the articles listed in the Classified Bibliography. Helpful information of a less direct nature will be found in the works on analysis by Sutton, Fresenius, and others, also in the chemical journals.

A. Free Cyanide

The commonest and most important test is for the strength of the solution in cyanide as a guide to the quantity of solid potassium or sodium cyanide that must be added to the solution, by the man on shift, to bring it up to the working strength used in the plant. This has been called the test for "free cyanide." The result is taken to indicate in terms of potassium cyanide, all the cyanogen (CN) or cyanide radical which is present in the solution as potassium cyanide (KCN), sodium cyanide (NaCN), and all the other simple cyanides of the alkalis and alkaline earth metals. The test for free cyanide in both theory and practice does not give all the cyanide strength that has a solvent action on gold and silver — not that of the double cyanides — but gives that strength obtained when making up new solution, or that additional strength obtained by adding the solid cyanide to a solution already in use and in good condition.

Standard Silver Nitrate Test. — A standard silver nitrate ($AgNO_3$) solution is made by dissolving 6.5232 grams pure crystallized silver nitrate in water and making up with pure water to one liter (1000 c.c.). To test the cyanide solution, 10 c.c. is measured by a pipette into a small beaker of three or four times that capacity. If the cyanide solution is turbid or muddy, it should be filtered before being measured out. To the 10 c.c. in the beaker is added as an indicator a few drops of a 3 to 5 per cent solution — 3 to 5 grams of the salt or substance dissolved in water and made up to 100 c.c. — of potassium iodide (KI), though the use of an indicator is not highly important and

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it is therefore often omitted. The standard silver nitrate solution is then run from a burette into the cyanide solution in the beaker; at first very fast, and finally drop by drop, shaking the beaker that the white cloud, formed as each drop of the standard silver nitrate is run in, may be dissolved until a faint, indistinct white cloudiness appears; then cautiously adding the drops until there is a permanent opalescence, slightly yellowish. Each c.c. of the silver nitrate solution used on 10 c.c. of the cyanide solution indicates one pound of potassium cyanide or its equivalent in a ton of solution. Thus, if 2.3 c.c. of silver nitrate were required, the solution contains the equivalent of 2.3 pounds potassium cyanide per ton.

In titrating weak solutions where close results are required in connection with experimental work, as much as 50 c.c. of the solution may be used for a test. Excessively strong solutions may be diluted with a little water after the portion for titration has been measured out, to prevent the formation of a granular precipitate of single cyanide, which does not so readily dissolve as the fine, white cloud of it first formed and quickly dissolved in weak solutions. It is customary to titrate against a black background, holding the beaker on a level with the eye and looking through a thick body of the liquid, that the end reaction may be better observed. Though the exactness of the end reaction may be observed with pleasing satisfaction in the case of solutions newly made up for test purposes, the exact end point cannot be so easily distinguished with complex mill solutions. The errors that may occur in this way in the daily routine of plant work or through the limitations of the method are generally unimportant, for mill solutions should be of such a strength that an accidental reduction of 10 to 20 per cent in their working strength for a short period should not be harmful. As the strength of solution used in a plant is eventually determined by the results obtained in practice and various experiments relating to the daily practice, the careful operator instructs his shift men to carry the end reaction to the same point that he does, thereby keeping the error — usually a case of carrying the reaction too far and overestimating the strength — constant and minimizing the danger in this direction.

Reactions in Standard Silver Nitrate Test. — The test depends upon the following reactions:

A. $AgNO_3 + KCN = AgCN + KNO_3$,
B. $AgCN + KCN = KAg(CN)_2$,
C. $AgNO_3 + KAg(CN)_2 = 2\,AgCN + KNO_3$,

in which the single silver cyanide (AgCN) formed in *A*, as indicated by the temporary white precipitate or cloud, redissolves in an excess of cyanide to form the soluble double silver cyanide ($KAg(CN)_2$) in *B*. After all the cyanide has been converted into the double silver cyanide, an additional drop of the silver nitrate will cause a precipitate of the single silver cyanide as in *C*, which is insoluble and does not dissolve in the absence of free cyanide, but forms a permanent opalescence.

The amount of silver nitrate to be used in standardizing may be computed by combining *A* and *B* into one equation, since the silver nitrate converts the cyanide into the double silver cyanide, as:

$$\underset{\underset{169.89}{\left\{107.88 + 14.01 + 48\right\}}}{Ag\ N\ O_3} + 2\ \underset{\underset{130.22}{\left\{39.10 + 12 + 14.01\right\}}}{K\ C\ N} = KAg(CN)_2 + KNO_3.$$

If 169.89 parts $AgNO_3$ combine with 130.22 parts KCN, then 1 part $AgNO_3 = \frac{130.22}{169.89}$ or .766496 parts KCN. If 1000 c.c. solution contains 6.5232 grams $AgNO_3$, 1 c.c. will contain .0065232 grams, which is equal to .766496 × .0065232 or .005 grams KCN. Consequently if 1 c.c. of $AgNO_3$ solution is required on 10 c.c. of the cyanide solution, the 10 c.c. contains .005 grams KCN, equal to .05 per cent or 1 pound KCN per ton solution.

Reactions of the Potassium Iodide. — The use of a few drops of a neutral 3 to 5 per cent solution of potassium iodide (KI) is said to correct or reduce the liability of errors in titrating complex mill solutions, but is added mainly as an "indicator" to make the end reaction more distinct. The reactions occurring are somewhat similar to those in the case of cyanide:

A. $AgNO_3 + KI = AgI + KNO_3$,
B. $AgI + 2\,KCN = KI + KAg(CN)_2$,
C. $AgNO_3 + KI = AgI + KNO_3$,

in which any single silver iodide (AgI) as first formed in *A* is dissolved by an excess of cyanide to form the double silver cyanide, while the iodide again becomes potassium iodide, as in *B*. When no more free cyanide is present, the single silver iodide forms and remains as a permanent precipitate as in *C*, giving a yellowish color to the solution. The tendency is for the iodide in a cyanide solution to be precipitated in preference to the cyanide, consequently the final permanent precipitate where the potassium iodide indicator is used becomes more the final potassium iodide reaction of:

$$C.\ AgNO_3 + KI = AgI + KNO_3,$$

than the final cyanide reaction of:

$$C.\ AgNO_3 + KAg(CN)_2 = 2\,AgCN + KNO_3.$$

Testing Strength of Solid Cyanide. — The strength of solid cyanide may be tested or determined by this method through weighing out a sample of the salt — from the interior of the cakes where the surface has absorbed moisture through exposure to the atmosphere — and making it into a solution of a certain theoretical strength, and then titrating a sample of the solution. Thus 5 grams of cyanide may be dissolved in water and made up to 500 c.c., making a 1 per cent (20 pounds) solution. Should the solution titrate .97 per cent (19.4 pounds per ton of solution) strong, then it is "97 per cent pure" or "strong" potassium cyanide. Should sodium cyanide be taken, it may be found to indicate 1.2 per cent (24 pounds per ton of solution), showing that the cyanide is 120 per cent strong when computed in terms of potassium cyanide.

B. HYDROCYANIC ACID AND ACIDITY

Occurrence. — The hydrocyanic acid existing in cyanide solution is formed mainly by the decomposition of cyanide by acids in the ore, as:

$$H_2SO_4 + 2\,KCN = 2\,HCN + K_2SO_4.$$

Test for Hydrocyanic Acid. — The hydrocyanic acid may be determined in cyanide solutions by first estimating the free cyanide in the usual way. Another sample of the solution is then taken, to which is added an excess of a solution of potassium or sodium bicarbonate ($KHCO_3$ or $NaHCO_3$). The solution is then titrated with the standard silver nitrate solution without the indicator, and the difference between this titration and that for free cyanide is taken as the hydrocyanic acid.

The addition of the solution of potassium or sodium bicarbonate causes the following reaction:

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

Since the HCN is titrated as KCN and the atomic weight of hydrogen (H) is 1, of potassium (K) is 39.1, of carbon (C) is 12, and of nitrogen (N) is 14:

$$\frac{\overset{1}{\mathrm{H}}\;\overset{12}{\mathrm{C}}\;\overset{14}{\mathrm{N}}}{\underset{39.1}{\mathrm{K}}\;\underset{12}{\mathrm{C}}\;\underset{14}{\mathrm{N}}} = \frac{27}{65.1} = .415.$$

The above titration less that for free cyanide indicates directly the pounds of potassium cyanide to which the cyanogen of the hydrocyanic acid is equivalent, or when multiplied by .415 the result indicates the amount of hydrocyanic acid.

Acidity Test for Hydrocyanic Acid. — Another method of determining the hydrocyanic acid, one which may more properly be defined as giving the " acidity " of the cyanide solution in the absence of any protective alkalinity, consists in neutralizing the cyanide, rendering the zinc innocuous, and determining the acidity with standard alkali. For this test one-half more to double the amount of standard silver nitrate required in the *total* cyanide test is added to 10 c.c. of the cyanide solution, to convert all the free cyanide and any other easily-decomposed cyanides into the double silver cyanide salt neutral to acidity or alkalinity. To this is added about 5 c.c. of a 5 per cent solution of potassium ferrocyanide ($K_4Fe(CN)_6$) or an excess over that required to precipitate the zinc in the solution as a potassium zinc ferrocyanide ($K_2Zn(Fe(CN)_6)$) inert in the test. Phenolthalein is now added as an indicator of alkalinity and if the solution turns red, indicating that there is a protective alkalinity, it should be titrated with the standard decinormal acid solution for the amount of protective alkalinity. But if the solution does not turn red, it is either neutral or acid and should be titrated with standard decinormal alkali until the solution becomes alkaline by turning red, for the amount of acidity. (See Protective Alkalinity for the preparation and use of standard acid and alkali solutions and indicators.) The acidity may be reported in the number of pounds of caustic soda (NaOH) or lime (CaO) that would be required per ton of solution to neutralize the acidity, the number of pounds of hydrocyanic acid it represents, or the cyanide that has been decomposed. Using 10 c.c. of cyanide solution, each c.c. of the standard decinormal alkali equals .04 per cent (.8 pound per ton of solution) caustic soda (NaOH) or .028 per cent (.56 pound) lime (CaO). Or each cubic centimeter of the decinormal alkali used equals .027 per cent (.54 pound) hydrocyanic acid (HCN), as indicated by the equation:

$$\begin{array}{ccccccc}
(1 + 12 + 14) & + & (23 + 16 + 1) & = & (23 + 12 + 14) & + & ((2 \times 1) + 16) \\
\mathrm{H \quad C \quad N} & + & \mathrm{Na \quad O \quad H} & = & \mathrm{Na \quad C \quad N} & + & \mathrm{H_2 \quad O.} \\
27 & & 40 & & 49 & & 18
\end{array}$$

A decinormal caustic soda solution contains 4 grams NaOH in 1000 c.c., or .004 grams NaOH in each c.c. In the equation 40 parts NaOH neutralizes 27 parts HCN, or 1 part NaOH equals .675 parts HCN, therefore 1 c.c. decinormal alkali equals .675 × .004 grams or .0027 grams HCN. If 1 c.c. decinormal alkali is equal to .0027 grams HCN in 10 c.c. of cyanide solution, it is equivalent to .027 per cent HCN or .54 pound per ton. Since, as shown before, the cyanogen in .415 pound HCN is equivalent to that in 1 pound KCN, .027 per cent or .54 pound HCN equals .065 per cent or 1.3 pounds KCN per ton solution. To tabulate:

One c.c. decinormal alkali taken in 10 c.c. solution

= .04 per cent NaOH or .8 pound per ton solution.
= .028 per cent CaO or .56 pound per ton solution.
= .027 per cent HCN or .54 pound per ton solution.
= .065 per cent KCN or 1.3 pounds per ton solution.

Nature of Acid Cyanide Solutions. — Cyanide solutions do not often become acid with careful manipulation, operators universally aiming to have at least a slight protective alkalinity. Though in working gold ores a few cases have been reported where a slight acidity of the solutions seemed beneficial, apparently owing to the retarding influence of excessive alkalinity on the dissolution of gold and the deleterious effects of the substances resulting from the decomposition of sulphides and base compounds by alkali, mainly the alkaline sulphides produced, generally a solution that is acid causes a waste of cyanide by the new acid that appears, a poor extraction through the fouling of the solution by base metals which alkali would render inert, and a poor precipitation in the zinc boxes, especially with a weak solution, when the tendency for a white precipitate of zinc cyanide to be formed in the boxes is great, owing to the inability of the weak solution to hold the zinc cyanide in a dissolved or soluble state.

C. TOTAL CYANIDE

Definition. — The test for total cyanide is considered to give, in terms of potassium cyanide, all of the cyanogen or CN radical present in the form of simple cyanides as determined by the test for free cyanide, which is the potassium and sodium cyanide and the other single cyanides of the alkalis and alkaline earths metals, and additionally that contained in the hydrocyanic acid (HCN) and the double cyanide of zinc ($K_2Zn(CN)_4$) and possibly some other easily-decomposed double cyanides.

Test with Standard Silver Nitrate and Alkali. — For determining the total cyanide, from 10 to 50 c.c. of the cyanide solution to be tested are taken, a few drops of the potassium iodide indicator are added, and then a few cubic centimeters of a normal caustic soda (NaOH) or caustic potash (KOH) solution — sufficient to make the solution excessively alkaline. It is then titrated with the standard silver nitrate solution as in the case of free cyanide, carrying the titration past a white turbidity until a permanent yellow color is obtained. The number of cubic centimeters of silver nitrate solution used in 10 c.c. of cyanide solution indicates in terms of potassium cyanide the number of pounds of total cyanide in a ton of solution. It should be observed if increasing the amount of alkali added will increase the total cyanide obtained, the highest result being taken.

The normal caustic alkali solution is made by dissolving 4 grams caustic soda or 5.6 grams caustic potash in water and making up to 100 c.c. Any strength of caustic alkali solution may be used in a sufficient quantity.

Reactions in Total Cyanide Test. — The above test depends in the case of hydrocyanic acid upon the production of a cyanide on the addition of an alkali to hydrocyanic acid, as:

$$HCN + KOH = KCN + H_2O.$$

In the case of the zinc potassium cyanide ($K_2Zn(CN)_4$) and other easily-decomposed double cyanides, the addition of an alkali causes, or apparently causes, a formation or regeneration into the simple cyanide, as:

$$K_2Zn(CN)_4 + 4\,KOH = 4\,KCN + Zn(KO)_2 + 2\,H_2O.$$

Action of Double Cyanides. — This is an important test, but in most cases is only made occasionally by the chemist in charge. The importance of the test arises from the fact that it has been found both in laboratory experiments and plant practice, that zinc potassium cyanide is an active solvent of gold and silver, and more active when apparently regenerated into KCN or a simple cyanide by the addition of alkali, as shown in the last equation. The following from tests by W. H. Virgoe* is typical:

Test No.	Solvent.	Strength.	KCN Consumption.	Extraction.
		Per cent	Lbs.	Per cent
1	KCN	.22	1.0	87
2	$K_2Zn(CN)_4$	.22	0.4	45
3	$K_2Zn(CN)_4+CaO$	.22	0.2	75

The practice that the above test would indicate has been adopted in many silver plants by adding the lime used for neutralizing and for giving a protective alkalinity in such a quantity that the tests for free and total cyanide closely approach each other or are practically the same, thus performing in the plant practice what is experimentally performed in the total cyanide test. The following adapted data by L. N. B. Bullock regarding his cyanide practice at Copala, Sonora, Mexico,† is an excellent illustration of this principle put into practice. The ore is valuable for its silver, which occurs as a sulphide and carries from 12 to 20 ounces of the metal per ton. When treatment was first commenced the protective alkalinity in the working solution was carried at .04 per cent NaOH (.56 pound lime (CaO) per ton solution), the use of 5.2 pounds lime per ton of ore being sufficient. The cyanide consumption varied from 4.3 to 4.5 pounds per ton of ore. With the object of ascertaining what results could be secured by decomposing the zinc cyanide

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* *Journal Chemical, Mining, and Metallurgical Soc. of S. A.*, Vol. 4, Aug., 1903.

† *Mining and Scientific Press*, June 8, 1907. Recent Cyanide Practice, pp. 264.

or double cyanides and regenerating cyanide through increasing the alkalinity, the amount of lime used was increased and kept gradually increasing until the solutions tested .2 per cent NaOH (2.8 pounds lime per ton) in protective alkalinity. The working solution for slime treatment, which was carried at .125 per cent (2.5 pounds) KCN, at once began to gain in strength, and kept gradually growing stronger until it showed .3 per cent (6 pounds) KCN; the alkalinity then being allowed to fall to .09 per cent NaOH (1.26 pounds lime per ton), the cyanide strength also fell. After many experiments with various strengths, it was found that .135 per cent NaOH (1.9 pounds lime per ton) was the least protective alkalinity that would give the desired regeneration of cyanide, and consequently the protective alkalinity has been kept at that figure since. As a result of this regeneration the cyanide consumption has not exceeded 1.5 pounds per ton treated for more than five months with no indication of any increase; as a matter of fact, no cyanide was added to the slime treatment solutions for nearly 13 weeks, and the amount used to bring the leaching plant solutions up to strength was small; this, of course, was due to the large excess of zinc cyanide existing in the system.

The use of an excessive protective alkalinity, while in many cases effecting a considerable saving in cyanide and by decomposing the base metal compounds to some extent better liberates the precious metals, decreases the solubility of native gold and silver, lessening the extraction or increasing the time required for dissolution. Consequently an excessive alkalinity is not used in treating gold ores and only on silver ores when found beneficial.

The amount of double cyanides in a plant solution should be watched and an attempt made to gauge their dissolving influence. Where there is a considerable amount, it is well to report the strength of the solution in both free and total cyanide. A sudden material reduction in the amount of double cyanide may indicate the need of a solution stronger in free cyanide, and that some important change is taking place in the solution that should be investigated. The zinc potassium cyanide is formed by the passage of the solution through the zinc boxes and in connection with the precipitation. It is customary to consider the double cyanide or the difference between the free and total cyanide as zinc potassium cyanide only.

D. PROTECTIVE ALKALINITY

Definition. — The "protective alkali" of a cyanide solution is taken to mean those substances, excepting the simple cyanides and the easily-decomposed double cyanides determined by the total cyanide test, that are alkaline to an indicator. the theory being that this protective alkalinity will be neutralized or destroyed by any acidity which the solution may encounter, before the cyanide is consumed or destroyed; that hydrocyanic acid will be prevented from forming or that which is already formed will be regenerated into free cyanide; and that many cyanicides will be rendered inert and harmless by direct or indirect reaction with the alkali.

Test for Protective Alkalinity. — The protective alkalinity is determined by taking 10 c.c. of the solution to be tested. To this is added about 5 c.c. or sufficient of a 5 per cent solution of potassium ferrocyanide ($K_4Fe(CN)_6$) that any zinc in solution as the double cyanide or otherwise may be rendered neutral by conversion into zinc potassium ferrocyanide, as:

$$K_4Fe(CN)_6 + K_2Zn(CN)_4 = K_2ZnFe(CN)_6 + 4\ KCN.$$

The cyanide liberated in this equation is converted by an excess of silver nitrate into a double silver cyanide, which, like the zinc potassium ferrocyanide, is neutral and inert to indicators in the test. Standard silver nitrate is added for this purpose to the extent of perhaps one-half more or double that required in the determination of *total* cyanide, to insure all the cyanide being converted into the neutral double silver cyanide. A few drops of phenolthalein solution are then added, which will turn the solution a bright red if any protective alkalinity is present. The solution will be unchanged if it is neutral or acid, in which case it should be titrated for its "acidity" with decinormal alkali as described in the test for hydrocyanic acid. If the solution is alkaline, it is titrated with decinormal acid until the red or pink shade of phenolthalein just disappears. Each cubic centimeter of the decinormal acid used on 10 c.c. of solution indicates the following equivalents:

.04 per cent caustic soda (NaOH), or .8 pound per ton solution.
.056 per cent caustic potash (KOH), or 1.12 pounds per ton solution.

.028 per cent unslacked lime (CaO), or .56 pound per ton solution.
.037 per cent slacked lime ($Ca(OH)_2$), or .74 pound per ton solution.

In mill work the results should be reported in pounds of the neutralizer used — invariably unslacked lime (CaO) — per ton of solution. Percentage may be used for technical purposes, and the kind of alkali in terms of which the results are stated, preferably as $NaOH$ or CaO, should always be given. The test may be conveniently, though somewhat less accurately, made in mill work by adding the silver nitrate in excess after the usual free cyanide test, then with or without potassium ferrocyanide, adding the indicator and titrating.

Preparation of Indicators. — The phenolthalein solution is prepared by dissolving 5 grams of phenolthalein in 1 liter (1000 c.c.) of a solution one-half pure water and one-half alcohol. Methyl orange may also be used as an indicator, except with oxalic acid. It is prepared by dissolving 1 gram of the powder in 1 liter of water. Litmus is seldom used.

Theory of Standard Acid and Alkali Solutions. — A normal (N) solution of a substance contains in 1 liter of the solution, the molecular weight in grams of the substance divided by the number of atoms of the active element (hydrogen or hydrogen equivalent) in the molecule of the substance. One c.c. of such a solution prepared from any acid will be exactly neutralized by 1 c.c. of such a solution of any alkali, by the active base of the alkali replacing the hydrogen equivalent of the acid, as:

$$\begin{cases} HCl + NaOH = NaCl + H_2O. \\ H_2SO_4 + 2\,NaOH = Na_2SO_4 + 2\,H_2O. \end{cases}$$

A half-normal $\left(\frac{N}{2}\right)$, fifth-normal $\left(\frac{N}{5}\right)$, or decinormal $\left(\frac{N}{10}\right)$ solution is one-half, one-fifth, or one-tenth as strong as a normal (N) solution.

Preparation of Standard Decinormal Acid and Alkali Solutions. — The standard acid solution may be prepared from nitric acid (HNO_3), hydrochloric acid (HCl), sulphuric acid (H_2SO_4), or oxalic acid ($C_2H_2O_4.2\,H_2O$), usually from the latter two. A decinormal $\left(\frac{N}{10}\right)$ acid solution made up to 1000 c.c. with pure water contains 4.9 grams H_2SO_4, 3.646 grams HCl, 6.3 grams

HNO_3, or 6.3 grams $C_2H_2O_4.2\,H_2O$. The quantities being arrived at by dividing the molecular weight of the acid by the number of hydrogen atoms in the chemical proper (not including the water of crystallization), and again by 10 to give the amount for a decinormal solution, as:

$$\begin{array}{l} H_2 \quad S \quad O_4 \\ 2 + 32 + 64 = 98 \div 2 \div 10 = 4.9. \end{array}$$

$$\begin{array}{l} H \quad Cl \\ 1 + 35.46 = 36.46 \div 1 \div 10 = 3.646. \end{array}$$

$$\begin{array}{l} H \quad N \quad O_3 \\ 1 + 14 + 48 = 63 \div 1 \div 10 = 6.3. \end{array}$$

$$\begin{array}{l} C_2 \quad H_2 \quad O_4. \quad 2\,H_2O \\ 24 + 2 + 64 + 4 + 32 = 126 \div 2 \div 10 = 6.3. \end{array}$$

For mill work the solution may be prepared by weighing out, or by measuring from a burette, the required amount of the highest-grade, chemically-pure acid of a reputable maker, and diluting with pure water up to the necessary amount. When the acid is measured from a burette the amount must be calculated from its specific gravity, for as water has been given a specific gravity of one as a standard and 1 c.c. has been taken as weighing 1 standard gram, so does a cubic centimeter of sulphuric acid of 1.845 specific gravity weigh 1.845 grams, and similarly with the other acids. For more exact work the acid solution must be standardized against a standard alkali solution of high accuracy, though solutions prepared from oxalic acid are very accurate without standardizing. Pure sodium carbonate (Na_2CO_3) is heated without fusing, to drive off the absorbed moisture, in a porcelain or platinum dish until the dish assumes a dull-red color, being kept at that heat for about 15 minutes. It is cooled under a dessicator and 5.3 grams ($Na_2CO_3 = 2(23) + 12 + 3(16) = 106 \div 2 \div 10 = 5.3$) weighed out quickly,— to prevent absorption of moisture,— dissolved in water, and made up to 1000 c.c. This is an exact decinormal alkali solution of which 25 or 50 c.c. are taken, the indicator added, and titration made with the decinormal acid solution, which must be diluted with water or strengthened with acid until a cubic centimeter of the standard acid will exactly neutralize a cubic centimeter of the standard alkali.

Decinormal alkali for rough mill work is prepared by dissolving 4 grams of caustic soda ($NaOH = 23 + 16 + 1 = 40 \div 1 \div 10 = 4$), or 5.61 grams caustic potash ($KOH = 39.1 + 16 + 1 = 56.1 \div 1 \div 10 = 5.61$) and making up to 1000 c.c. with water. The pure chemical should be used from a well-stoppered bottle in which the tendency to deliquesce or absorb moisture, which is especially great with caustic potash, is at a minimum. The solution in any case will not be very accurate and should be adjusted to a standard acid solution.

When using 10 c.c. of the cyanide solution, each cubic centimeter of the decinormal acid used indicates a protective alkalinity, or each cubic centimeter of the decinormal alkali used equals an acidity equal to the following:

.04 per cent caustic soda ($NaOH$), or .8 pound per ton solution.
.056 per cent caustic potash (KOH), or 1.12 pounds per ton solution.
.028 per cent unslacked lime (CaO), or .56 pound per ton solution.
.037 per cent slacked lime ($Ca(OH)_2$), or .74 pound per ton solution.

The above values may be computed from the quantities in any of the standard acids or alkalis, since they exactly equal or neutralize each other. Taking caustic soda as the illustration, the decinormal solution contains 4 grams in 1000 c.c., or .004 gram in each cubic centimeter. The .004 gram $NaOH$ or its equivalent in acid used in titrating 10 c.c. of solution is equal to .04 per cent of the 10 c.c., showing the solution to contain the equivalent of .04 per cent $NaOH$. The value of the caustic potash can be figured out in a similar way, using 5.61 grams as required in the decinormal solution. In the case of unslacked lime (CaO), one atom of calcium (Ca) replaces two of hydrogen as indicated in the formation of calcium sulphate by sulphuric acid and lime:

$$H_2SO_4 + Ca = CaSO_4 + 2\,H.$$

Consequently $CaO = 40 + 16 = 56 \div 2 \div 10 = 2.8$ grams in 1000 c.c. of decinormal solution or .0028 gram in each cubic centimeter. The .0028 gram or its equivalent in acid used in titrating 10 c.c. of solution is equal to .028 per cent of the 10 c.c., showing the solution to contain the equivalent of .028 per cent

CaO. The value of slacked lime ($Ca(OH)_2$) may be figured in a similar way: $Ca(OH)_2 = 40 + 2\,(16 + 1) = 74 \div 2 \div 10 = 3.7$ grams required in 1000 c.c. $\frac{N}{10}$ solution, and so on.

The amounts of the chemicals required in 1000 c.c. of solution to make up a $\frac{N}{10}$ solution may be tabulated:

4.9	grams sulphuric acid, 2.66 c.c. (specific gravity of 1.845).
3.646	grams hydrochloric acid, 3.04 c.c. (specific gravity of 1.20).
6.3	grams nitric acid, 4.44 c.c. (specific gravity of 1.42).
6.3	grams oxalic acid. (Exists in solid form.)
4	grams caustic soda. (Exists in solid form.)
5.61	grams caustic potash. (Exists in solid form.)
5.3	grams sodium carbonate. (Exists in solid form.) (For standardizing.)

It is convenient in plant practice to make the solutions so that 1 c.c. of the standard solution when used on 10 c.c. of cyanide solution will indicate a protective alkalinity or an acidity equal to 1 pound CaO (unslacked lime) per ton of solution. For this purpose 1000 c.c. of the standard solution must contain its chemical in the following quantity:

8.75	grams sulphuric acid, 4.74 c.c. (specific gravity of 1.845).
6.51	grams hydrochloric acid, 5.42 c.c. (specific gravity of 1.20).
11.25	grams nitric acid, 7.92 c.c. (specific gravity of 1.42).
11.25	grams oxalic acid. (Exists in solid form.)
7.143	grams caustic soda. (Exists in solid form.)
10.02	grams caustic potash. (Exists in solid form.)
9.464	grams sodium carbonate. (Exists in solid form.) (For standardizing.)

The action and use of alkalinity have been indirectly referred to before and will be more fully treated under Alkalinity and Lime.

E. TOTAL ALKALINITY

The "total alkalinity" of a cyanide solution is that alkalinity which is visible in the presence of an alkaline indicator. It may be said to be that of the protective alkalinity and additionally of the cyanides — simple, double, and otherwise.

The test is conducted and computed exactly the same as in the determination for protective alkalinity, except that the cyanide and zinc are not rendered neutral and inert by the addition of silver nitrate and potassium ferrocyanide.

F. FERROCYANIDES AND FERRICYANIDES

Definition and Occurrence. — Iron in a metallic form is attacked by cyanide with extreme slowness, but most of the iron compounds are more readily affected and dissolved. Iron may be introduced into the solution in this way by contact with corroded pipes and iron tanks, or iron originating in the milling process and subsequently altered to a compound susceptible of being readily dissolved by cyanide solution. Also through iron that is a constituent of the ore as an oxide, sulphate, etc., from the previous decomposition of the iron or other pyrite, or in the case of a fresh unweathered pyrite by the slow alteration when it is attacked by the cyanide or alkali of a solution, especially in the presence of oxidation.

The combination of iron (Fe) and cyanide forms an infinite variety of compounds under the head of ferrocyanides and ferricyanides, in the simplest form, as:

$$FeSO_4 + 6\,KCN = K_4Fe(CN)_6 + K_2SO_4.$$

In which the ferrous salt, iron sulphate ($FeSO_4$), when brought into contact with cyanide forms potassium ferrocyanide ($K_4Fe(CN)_6$) and potassium sulphate (K_2SO_4), while the ferrocyanide may eventually be changed by oxidation to ferricyanide ($K_3Fe(CN)_6$). Iron in these combinations is one of the principal foreign constituents of a cyanide solution. The ferrocyanides have been considered as reducers through utilizing the oxygen to form ferricyanides, while the ferricyanides have been considered as oxidizers by the reversal of the method of their formation. The more practical view is that they are reducers and thereby harmful. They occur in the solution in proportion as the iron found in the ore in a state subject to being acted upon by cyanide has not been removed or oxidized into the innocuous ferric oxide or hydrates by water-washing, alkaline treatment, and aeration. When occurring in small quantities their influence is unnoticeable, but when present in large amounts their effect is very harmful, reducing the percentage of extraction or retarding the dissolution of the precious metals, and hindering the precipitation chemically and also mechanically through the precipitation of ferrocyanide compounds in the zinc box, such as zinc potassium ferrocyanide and similar. When solutions become highly charged with ferrocyanides and ferricyanides so that they cannot be made to effect the extraction made in laboratory tests with clean solutions, they should be discarded, which may be after only a few months of use. However, this periodical discarding of the solutions used in working a decomposed highly pyritic ore can usually be avoided by a proper removal, neutralization, or alteration of the iron by water-washing, alkali, or aeration. Or the solution may often be brought to a healthy state again by aeration and possibly by the addition of alkali. Mercurous or mercuric chloride has been added to such solutions with the effect, through the activity of mercury in combining with the cyanogen of the simple and double cyanides and including the ferrocyanides and ferricyanides, to form a potassium mercuric cyanide which is an active solvent even without oxygen, and probably regenerates the cyanogen of ferrocyanides and ferricyanides into the active mercuric cyanide. The presence of ferrocyanides is second only to the formation of alkaline sulphides in fouling working solutions, and the part that each plays in such a fouling effect is an interesting and difficult study.

Determination. — The simplest method of determining the ferrocyanides and ferricyanides is to consider all the iron in solution as a ferrocyanide. The cyanogen is decomposed by evaporating a measured quantity of solution with HNO_3, taking up with H_2SO_4, evaporating almost to dryness, and taking up with water, when the iron may be determined by any of the usual methods for determining iron. The amount of iron found is multiplied by 6.6 to give the potassium ferrocyanide as $K_4Fe(CN)_6$, as:

$$\frac{\underset{K_4}{4\,(39.1)} + \underset{Fe}{55.85} + \underset{(CN)_6}{6\,(12+14)}}{\underset{55.85}{Fe}} = \frac{368.25}{55.85} = 6.6.$$

G. ALKALINE SULPHIDES AND SULPHOCYANIDES

Definition and Occurrence of Alkaline Sulphides. — Alkaline or soluble sulphides, as potassium sulphide (K_2S), sodium sulphide (Na_2S), etc., are often formed in a cyanide solution, or may be contained in small quantities in the cyanide used. They are considered to occur principally through the decomposition of a metallic sulphide by cyanide or alkali. In the case of treating the silver sulphide (Ag_2S) or the iron pyrite (FeS_2) by cyanide, as:

$$\left\{\begin{array}{l} Ag_2S + 4\,KCN = K_2S + 2\,KAg(CN)_2. \\ FeS_2 + 6\,KCN = 2\,K_2S + K_2Fe(CN)_6. \end{array}\right.$$

The decomposing effect of alkali with the formation of an alkaline sulphide is shown by the equation:

$$FeS_2 + 2\,KOH = K_2S + Fe(OH)_2 + S.$$

The general effect of an alkali in attacking a metallic sulphide is to form an alkaline sulphide with the sulphur and a hydrate (as $Fe(OH)_2$) of the base remaining. While the alkalis do not easily act upon all the metallic sulphides, they undoubtedly have some solvent or decomposing effect in all cases.

Alkaline Sulphides and Sulphocyanides or Thiocyanates. — The alkaline sulphides reduce the dissolving power of the solution by abstracting the oxygen present, probably in two ways. First, by the oxidation of the alkaline sulphide into an alkaline sulphate, as:

$$K_2S + 4\,O = K_2SO_4;$$

and second, by forming a sulphocyanide or thiocyanate (KCNS), as:

$$K_2S + KCN + H_2O + O = KCNS + 2\,KOH.$$

In which both oxygen and cyanide are utilized and rendered useless for dissolving purposes. The double reaction, first into the sulphide and then into the thiocyanate, may be stated as:

$$\left\{\begin{array}{l} Ag_2S + 5\,KCN + H_2O + O = 2\,KAg(CN)_2 + KCNS + 2\,KOH. \\ FeS_2 + 8\,KCN + 2\,H_2O + 2\,O = K_2Fe(CN)_6 + 2\,KCNS + 4\,KOH. \end{array}\right.$$

Action and Removal of Alkaline Sulphides. — The alkaline sulphides are unstable and tend to reverse the equation made by their formation, precipitating the silver as a sulphide, and possibly doing the same to a slight extent with the gold, or at least retarding its dissolution. The tendency of the silver to be reprecipitated in this way increases as the solution becomes weaker in cyanide in obedience to the law of mass action; which is, that in the case of one substance acting chemically on another, the action will proceed until the mass of the acting substance is overcome by the mass of the active substance formed, at which point equilibrium is established, and if the mass of the active substance formed is then increased or that of the acting substance decreased, there will be a reversal of the chemical reaction until chemical equilibrium is again established. Consequently, the mass or strength of the cyanide must be high in the case of considerable alkaline sulphides, to keep them from reversing the reaction and reprecipitating the silver. This is one of the reasons for the stronger solutions used in silver plants. The principle can be shown by dissolving silver sulphide (Ag_2S) in strong cyanide solution, and then diluting the solution until a precipitate of the Ag_2S forms. In plant practice too weak a solution either does not dissolve the silver or may allow it to be reprecipitated; while an accidental wash of very weak solution or water before dissolution is completed, appears to stop further extraction with strong solution, possibly by coating the metal or mineral with a hard, insoluble film of silver sulphide which is extremely difficult to dissolve.

The quantity of alkaline sulphides formed in treating a gold ore is small, probably increasing as the amount or percentage of pyrite or metallic sulphide (concentrate) increases and as stronger cyanide or alkaline solutions are used. The alkaline sulphides are probably removed as fast as formed in ordinary gold ores by being precipitated and discharged in the pulp residue as an insoluble zinc sulphide (ZnS), through reacting with the zinc potassium cyanide ($K_2Zn(CN)_4$) formed in the passage of the cyanide solution through the zinc boxes, as:

$$K_2Zn(CN)_4 + K_2S = 4\,KCN + ZnS.$$

Though the zinc in solution, reacting in a way similar to that of silver when reprecipitated, is a valuable ally in this way, it may not as completely remove the sulphides as desirable, or be able to cope with the large quantities produced in treating sulphide ores, especially the sulphide ores of silver. A lead compound is more active for this purpose, precipitating the sulphur as an insoluble and inert lead sulphide (PbS). Lead acetate ($Pb(C_2H_3O_2)_2$) has principally been used for this purpose. Litharge (PbO), the oxide of lead, has also been employed, but being insoluble cannot be conveniently used. The reaction in the case of using lead acetate is:

$$Pb(C_2H_3O_2)_2 + 4\,KOH = K_2PbO_2 + 2\,K(C_2H_3O_2) + 2\,H_2O.$$
$$K_2PbO_2 + K_2S + 2\,H_2O = PbS + 4\,KOH.$$

Mercury dissolved in amalgamating in cyanide solution or added as a soluble salt is even more active in precipitating the alkaline sulphides than lead compounds, and mercurous (Hg_2Cl_2) or mercuric chloride ($HgCl_2$) has been used for this purpose. Oxidation of the alkaline sulphides into sulphates or thiocyanates, by aeration of the solution and charge, will cause a solution containing alkaline sulphides to regain its solvent ability. It is this tendency of the alkaline sulphides to oxidize, this strong reducing action in utilizing the oxygen that is necessary in dissolving the gold and silver, that causes the poor or retarded extraction from ores giving rise to the alkaline sulphides. The sulphides are often precipitated in the zinc boxes as a zinc or silver sulphide, the sulphur of which may give trouble in the clean-up. There is a possibility that some of the influence that running a solution through the zinc boxes has on cleansing it and making it a more active solvent is due to the precipitation of the soluble sulphides by the zinc dissolved and other influences in the passage through the boxes.

Application of Lead Acetate. — Lead acetate for the purpose of precipitating the alkaline sulphides is not often used in gold plants, for small amounts of the sulphides are not harmful and are removed by the zinc. It is used in most silver plants, though it does not appear necessary unless working on sulphide ores. Since the alkaline sulphides are generally oxidized or changed to the thiocyanates in a short time, solutions in which they have formed do not often show them, but rather the resulting thiocyanates, the determination of which — since only a part of the alkaline sulphides are changed to the thiocyanates — does not appear to be of practical value in the matter of removing the soluble sulphides. Consequently, the amount of lead acetate or other precipitant to be used can only be determined in an empirical way, by attempting to learn its influence on the extraction and by examining the solution for the alkaline sulphides before time has been allowed them to oxidize. The tendency of the solution to foul against further dissolution of the precious metals until aerated, its reducing power, and its efficiency in competition with newly-made solutions are studied in this connection. Most plants treating sulphide silver ores use a half-pound or less of lead acetate per ton of ore treated; some use as high as a few pounds. It is prepared as a solution to be added to the agitation charge or sprinkled throughout the sand to be leached, or less often added to the solution in some convenient manner.

Test for Alkaline Sulphides. — The presence of alkaline sulphides may be determined by agitating 200 c.c. of the solution with a small quantity of lead carbonate ($PbCO_3$). A black precipitate of lead sulphide will indicate the presence of alkaline sulphides. Another method of testing consists of preparing a solution of nitroprusside by adding a little nitric acid to a solution of ferro or ferricyanide of potassium. Add a few drops of the nitroprusside solution to the cyanide solution. If alkaline sulphides are present, even in minute quantities, the solution will assume a brilliant purple color.

H. Available Cyanide

Definition. — " Available cyanide " has been given as an indefinite term referring to the " solvent ability " of a cyanide solution to dissolve the precious metals. This ability is due and proportional to the amount of the free or simple cyanide mainly, to some extent to the easily-decomposed double cyanides and perhaps the hydrocyanic acid, and to little if any extent to the other cyanogen compounds. It is indirectly affected by many other things, as the protective alkalinity, the amount of oxygen available, the quantity and nature of the foreign constituents, and the substances in combination with the cyanogen. This will indicate that an estimation of the constituents and characteristics usually determined will not clearly indicate the solvent ability or efficiency of a solution. Consequently the available cyanide or the solvent ability cannot be reduced to any terms. It can only be compared with a so-called standard, for which a new or freshly-made-up solution may be taken.

Test for Available Cyanide. — The usual method of estimating the available cyanide or dissolving efficiency is to run comparative laboratory tests on two portions of the same sample of ore, treating one with the mill solution to be tested and the other with a new solution. Care should be taken that the amount of sample taken, the volume of solution used, and the strength of solution, etc., is exactly similar in the duplicate tests. Where these are made frequently a large sample of 50 to 100 pounds may be prepared and check tests made on it with fresh solution, after which tests may be made on portions of the sample with mill solutions whenever desired. This will give the results with only one test, but most operators will prefer the first method, taking for their sample the discard from the sample for assay taken of a working charge, and thereby checking the working extraction by laboratory tests.

I. CYANATES AND TOTAL CYANOGEN

Cyanic acid has the composition HCNO. In cyanates the H is replaced by a metal or base as KCNO. The decomposition of cyanide through oxidation in a solution may be considered as into a cyanate, which is without effect in the practical working of the process.

" Total cyanogen " is a term that has been used to indicate or refer to all the cyanogen or CN radical present in a solution, in any form whatever, such as in the simple and double cyanides, the hydrocyanic acid, and the ferrocyanides, ferricyanides, thiocyanates, cyanates, etc.

J. REDUCING POWER

Cyanide solutions vary in their reducing power, in the tendency for substances in the solution to oxidize and thus abstract the dissolved oxygen that should be available for the dissolution of gold and silver. The reducing power can be determined in a comparative way by preparing a decinormal $\left(\frac{N}{10}\right)$ solution of potassium permanganate ($KMnO_4$), containing 3.16 grams of the chemical in one liter of water. Any convenient but standard amount of the cyanide solution may be taken and acidulated by a standard amount of sulphuric acid, to which the standard potassium permanganate solution should be added, until the color no longer disappears. The results, as stated in the amount of decinormal potassium permanganate solution used on a constant amount of cyanide solution, are recorded and studied in a comparative way, to learn the advisability or necessity of aerating the solution and ore, and any changes that may take place in the reducing power of the solution or ore.

K. Assay of Metals in Cyanide Solution

Classification of Methods for Gold and Silver. — The usual methods of assaying cyanide solution for gold and silver fall naturally into four classes:

A. Evaporation of the solution in a tray or boat of lead foil, followed by the cupellation of the residue and lead tray.

B. Evaporation of the solution with litharge, addition of suitable fluxes, fusion in the assay furnace, and cupellation.

C. Precipitation of the precious metals, filtration, incinerating the filter and its precipitate, fluxing the residue, melting, and cupeling.

D. Precipitation of the precious metals with a large amount of lead, removing precipitate from the solution, and cupeling without fusion — the Chiddy method.

A. **Lead Tray Evaporation.** — A block of wood about $\frac{3}{4}$ inch thick, $1\frac{3}{4}$ inches wide, and $2\frac{1}{2}$ inches long is prepared. The lead foil is cut into strips to be folded about this block into a tray $\frac{3}{4}$ inch deep. A test tube is graduated by a file mark or a sticker to hold 1 assay ton of solution as indicated by 29.166 c.c. of solution run in from a burette. By means of this measuring tube, 1 assay ton of the solution is placed in the tray, which is set on a piece of asbestos board on a hot plate to be evaporated to dryness. After evaporation the tray is folded into a compact mass and placed in a cupel to be cupeled. This method does not give extremely accurate results, through the spitting of the solution while evaporating and through the impurities in the solution affecting the subsequent cupellation. It has the further disadvantage of requiring considerable time and enabling only a small amount of solution to be taken for assay — though larger trays taking 3 assay tons may be conveniently used. However, being a simple method it is often employed.

B. **Evaporation with Litharge, etc.** — Any measured quantity of the solution may be taken, usually 5 to 10 assay tons. This is placed in a porcelain evaporating dish, covered with from 20 to 40 grams of litharge, to lessen the tendency to spit, and evaporated without boiling for the same purpose. The residue after evaporation is transferred to an assay crucible, to which is added the fluxes necessary to produce the usual assay fusion, which may consist of 15 grams bicarbonate of soda, 5 grams borax glass, 4 grams silica or more of powdered glass, and 1 gram flour as a reducer. The flux may be varied in any way that will give a satisfactory fusion. The amount of silica or powdered glass used should be just sufficient to prevent the charge from attacking the crucible. The amount of flour or reducer should be varied to give a button of the desired size. The lead button obtained from the fusion is cupeled, etc., in the usual way. Evaporation with litharge is supposed to give the most exact results of the different methods in use. It has been, and will continue to be, the method by which all other methods will be checked, but owing to the time and labor involved is not used in ordinary work.

C. **Precipitation, Incinerating, Fusing, etc.** — This method involves the precipitation of the precious metals from the solution by the addition of a metal, metallic salt, or other substance; the filtering off of the precipitate followed by incinerating it and the filter; and the fluxing and fusing of the residue as in the usual fire assay. Numerous reliable methods have been used, each of which has its advocates. They are the methods that were formerly used in plant practice, but which have largely been superseded by the Chiddy method. The following is one of the simplest and most practical of these methods. Take 10 assay tons of the cyanide solution in a beaker. Add 4 grams of zinc dust on point of spatula. Stir vigorously. Allow to stand for a few minutes and again stir. Finally add about 10 c.c. of commercial H_2SO_4 and stir. After action has ceased, add more H_2SO_4 if needed, until sulphuric acid is in excess and zinc is all dissolved, which will be indicated by no more action when a small quantity of the acid is added. Filter and incinerate filter and precipitate by placing in an assay crucible and setting in muffle or furnace. Add suitable flux after incinerating and cooling, which may consist of 10 grams litharge, 10 grams bicarbonate of soda, 3 grams silica or more of powdered glass, and 1 gram flour. Fuse, cupel lead button, etc.

D. **Precipitation with Direct Cupellation — the Chiddy Method.** — The Chiddy method with various modifications is now generally used, as it is an easily-handled, quick, and reliable method for gold and silver. Place 5 to 10 assay tons of the solution in a beaker. Heat nearly to boiling. Add before or during the heating, 10 c.c. of a clear saturated solution of lead acetate ($Pb(C_2H_3O_2)_2.3\ H_2O$) and .5 gram zinc dust. One part of lead acetate will dissolve in 2 parts of water, while any excess will remain undissolved in the bottom of the bottle or vessel holding the solution. Stir well and bring nearly to boiling. Allow to heat for several minutes. Stir again and add 15 c.c. of commercial HCl, and continue heating. After effervescence has ceased, add more HCl until the absence of action shows that the zinc is dissolved and the acid is in excess. The lead has now settled into a sponge which should be tapped together and pressed into a mass with a glass rod, the solution poured off, and the lead washed once or more by decantation. The lead is pressed into a compact mass with a glass rod or the fingers to remove the water. It is placed on a piece of lead foil $1\frac{1}{2}$ inches square, which is folded to allow the steam to escape at the top, and is placed in a hot cupel for cupellation.

The principal trouble encountered in using this method is the tendency of the lead to break up instead of agglomerating, and thus become lost in the decanting. This may be prevented to some extent by not allowing the solution to come to a boil. Another method is to introduce into the solution a piece of aluminum from 1 to $1\frac{1}{2}$ inches square and $\frac{1}{16}$ to $\frac{1}{8}$ inch thick, which will more readily collect the lead, and can easily be removed before cupellation. Where the lead breaks up, it may be finally transferred to the point of a small filter paper, which should be dried before cupeling. A simple expedient is to transfer or wash the lead into a small lead foil tray an inch or more square, drain the moisture through a folded corner, and dry tray before cupeling. Where zinc dust is not available a somewhat larger amount of the shavings may be used. This method gives good results with very weak solutions, but it is advisable to have some strong solution on hand and bring very weak solutions up to not to exceed .5 per cent (10 pounds) KCN.

L. ASSAY OF BASE METALS IN SOLUTION

The base metals in a cyanide solution, such as iron, copper, lead, zinc, etc., may be determined in any of the usual ways by first decomposing the cyanogen. This may be performed by adding to a measured quantity of solution, as 100 to 300 c.c., from 5 to 10 c.c. HNO_3 and the same amount of H_2SO_4, evaporating to dryness, and taking up with a few cubic centimeters of H_2SO_4 diluted with water. This gives the metals as sulphates for determination in the usual manner.

CHAPTER VI

ALKALINITY AND LIME

Definition and Properties of Lime. — Lime (CaO) is an alkaline earth, an oxide of the metal calcium (Ca). Neither calcium nor its oxide (lime) occurs free in nature. Lime is prepared by burning or calcining limestone, which when pure is calcium carbonate ($CaCO_3$) or carbonate of lime, thereby driving off the carbonic acid (CO_2) and leaving lime (CaO) in unfused lumps in the form of the original stone. In this state it is called burnt lime, unslacked lime, quicklime, caustic lime, calcium oxide, or dehydrated or anhydrous calcium oxide or lime. When this lime is exposed to the atmosphere, it attracts moisture and falls into a powder with a rapidity dependent upon the amount of moisture in the air and the quality of the lime, more rapidly as the quality of the lime or absence of impurities becomes higher and the calcining process has been carried to the proper point. This process is called air-slacking.

When lime (CaO) is brought into contact with water (H_2O), it decomposes the water with the evolution of much heat — a process called slacking — to form $Ca(OH)_2$, known as slacked lime, or calcium or lime hydroxide or hydrate; it is the same product as is formed by air-slacking. The term " lime " in connection with the cyanide process always refers — unless otherwise noted — to the equivalent of unslacked lime (CaO), whether the lime is slacked before use or not.

Lime after being slacked readily mixes with water to form a smooth and liquid paste called " milk of lime." A filtered or clear saturated solution of lime is called " lime water." This, as representing the maximum solubility of lime in water, contains about 1 part unslacked lime (CaO) in 800 parts of water, equal to .125 per cent or $2\frac{1}{2}$ pounds per ton of water. This is equal to 1 part of slacked lime ($Ca(OH)_2$) in 600 parts of water, but the results are invariably stated as unslacked lime. By the use of an excess of lime, cyanide solutions can be made to show a protective alkalinity of higher than .125 per cent ($2\frac{1}{2}$ pounds) CaO, but probably not more than .15 per cent (3 pounds); this is due to the alkalinity of other substances, more especially those resulting from chemical reactions.

Uses of Lime and Alkalinity in the Cyanide Process. — Lime and alkalinity have no solvent action upon the precious metals, but enter into numerous reactions occurring in the cyanide process. Through these they protect the cyanide from being destroyed or decomposed, mainly by entering into the combinations that cyanide would otherwise enter; by liberating or regenerating cyanide through replacing it in compounds; and by precipitating or rendering inactive substances that may interfere, thus keeping the solution clean and in excellent condition for dissolving the precious metals out of the ore and precipitating them in the zinc boxes. A further use of lime and alkalinity is as a solvent upon base metals and compounds, thereby better liberating the precious metals from the chemical combination or mechanical alloy or covering for easy dissolution. In doing this lime acts as an alkali, and any alkali could be used with more or less advantage for this purpose, but lime is almost exclusively used owing to its cheapness and that its properties and reactions are preferable to those of the other alkalis which have been used, principally caustic soda and to a slight extent caustic potash. In illustrating the chemical reactions of the cyanide process it is customary to use caustic potash (KOH) as the alkali formed or decomposed, probably because the K of the alkali is convenient to add to the CN as KCN. The student should bear in mind that the use of caustic potash in this way is as a generic or class term referring to alkalis in general and not to caustic potash in particular, consequently K, Na, and Ca may be considered to have similar properties when united to CN as a cyanide or to OH as an alkali, and likewise to a more or less extent with the other alkaline earths and metals.

Lime also acts physically in addition to chemically. It is used in connection with slime treatment for the purpose of causing the light, feathery, suspended slime to agglomerate and settle rapidly for decantation or pulp-thickening purposes. This subject is treated under Slime Treatment and Agitation.

Neutralization of Metallic Salts. — The main use of lime is as a neutralizer of the acid and metallic salts formed in the ore through the oxidation and decomposition of the metallic sulphides. The principal stages in the decomposition of iron pyrite may be adapted from those given by W. A. Caldecott,* as:

(1).	FeS_2.	Iron pyrite.
(2).	$FeS + S$.	Ferrous sulphide and sulphur.
(3).	$FeSO_4 + H_2SO_4$.	Ferrous sulphate and sulphuric acid.
(4).	$Fe_2(SO_4)_3$.	Normal ferric sulphate.
(5).	$2\,Fe_2O_3.SO_3$.	Basic ferric sulphate.
(6).	$Fe_2O_3.xH_2O$.	Ferric hydrate.
(7).	Fe_2O_3.	Ferric oxide.

A fresh unoxidized metallic sulphide, as an iron pyrite, is but little acted upon by cyanide solution. But in a weathering and partially decomposing iron pyrite this reaction probably occurs:

$$FeS_2 + H_2O + 7\,O = FeSO_4 + H_2SO_4.$$

The sulphuric acid (H_2SO_4) formed, if brought into contact with the cyanide, will destroy it through forming hydrocyanic acid (HCN), as:

$$H_2SO_4 + 2\,KCN = 2\,HCN + K_2SO_4.$$

The decomposition into hydrocyanic acid may be prevented by washing the soluble sulphuric acid out of the ore before the application of cyanide, by a neutralizing alkali wash, or less perfectly by a protective alkalinity in the cyanide solution, as:

$$H_2SO_4 + Ca(OH)_2 = CaSO_4 + 2\,H_2O.$$

The calcium sulphate ($CaSO_4$) formed being an insoluble salt harmless to the cyanide. If caustic soda (NaOH) is used, the resulting sodium sulphate (Na_2SO_4) is very soluble. The potassium sulphate (K_2SO_4) resulting from the use of caustic potash (KOH) is also very soluble. Any hydrocyanic acid will be regenerated in the presence of lime or protective alkalinity into a cyanide, as:

$$HCN + KOH = KCN + H_2O.$$

The ferrous sulphate ($FeSO_4$) formed in the first equation of the previous paragraph will destroy cyanide, as:

$$FeSO_4 + 2\,KCN = Fe(CN)_2 + K_2SO_4.$$
$$Fe(CN)_2 + 4\,KCN = K_4Fe(CN)_6.$$

In which the simple iron cyanide ($Fe(CN)_2$), if it is formed at all,

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* Proc. Chemical and Metallurgical Soc. of S. A., Vol. 2.

is immediately changed into the species of double cyanide, the ferrocyanide ($K_4Fe(CN)_6$). Further complex reactions between the ferrocyanide and the ferrous sulphate may result in the formation of Prussian blue ($Fe_4(Fe(CN)_6)_3$), which will give a blue color to the solution and deposit small quantities of the dark-blue compound in the ore, tanks, and piping, acting as a sign that the neutralization has been poorly carried out.

The ferrous sulphate is very soluble and may be washed out or may be neutralized into the harmless insoluble calcium sulphate by means of lime, or into other harmless salts by other alkalis, as:

$$FeSO_4 + Ca(OH)_2 = CaSO_4 + (FeO + H_2O \text{ or } Fe(OH)_2).$$

The ferrous oxide (FeO), if it does form, is hydrolized into the ferrous hydroxide or hydrate ($Fe(OH)_2$), a white precipitate which turns a dirty green, is insoluble, and is easily oxidized into the insoluble ferric oxide (Fe_2O_3), thus acting as a strong reducer or deoxidizer in the solution and ore. The ferrous hydroxide or hydrate is acted upon by cyanide, as:

$$Fe(OH)_2 + 6\,KCN = K_4Fe(CN)_6 + 2\,KOH.$$

It is probably the principal source of loss through the reaction between iron and cyanide, and the hardest iron interference to remove. The loss of cyanide is reduced by affording every means for the ferrous hydrate ($Fe(OH)_2$) to be oxidized into the ferric oxide (Fe_2O_3), which is unacted upon by cyanide. The use of aeration or oxygen besides reducing the cyanide consumption, increases the extraction of the precious metals by supplying the oxygen that would otherwise be abstracted by the iron from the solution or ore. To meet such conditions it may be necessary to pump air through the charge before or during treatment.

If the ferrous sulphate ($FeSO_4$) is not removed when formed, it will oxidize into the poorly soluble normal ferric sulphate ($Fe_2(SO_4)_3$), and from that into the insoluble basic ferric sulphate ($2\,Fe_2O_3.SO_3$), both of which will react with cyanide and thereby cause a high consumption in the same manner as ferrous sulphate, but in the presence of alkali these are probably converted or oxidized into the hydrous ferric oxides, as:

$$\left\{\begin{array}{l} Fe_2(SO_4)_3 + 6\,KOH = Fe_2O_3.3\,H_2O + 3\,K_2SO_4. \\ 2\,Fe_2O_3.SO_3 + 2\,KOH = 2\,Fe_2O_3.H_2O + K_2SO_4. \end{array}\right.$$

The hydrous ferric oxides or ferric hydrates have the formula $Fe_2O_3.xH_2O$ in which x, representing the number of molecules of H_2O, is variable and indeterminate. The ferric hydrates are formed from the ferrous and ferric salts — as the sulphates — by the action of alkali. Some of them are to some extent soluble and act upon cyanide, others do not. Taken as a whole, they may be considered to be insoluble and harmless to cyanide, more especially in the presence of alkalinity and aeration which increases these properties or carries the compound nearer to the inert ferric oxide (Fe_2O_3) — ferric hydrate less its water of combination. The red or blood color sometimes noted in solutions is usually due to a soluble iron or manganese compound which the addition of alkali or a continued aeration will precipitate as an inert ferric or other hydrate. The decomposition of sulphide ores may result in the formation of sulphates of other metals, such as magnesium and aluminium, which would interfere in a way similar to the iron salts, but the use of lime or alkalinity converts them into hydroxides that are harmless.

Lime and Alkalinity in Zinc Precipitation. — Lime and alkalinity play an important part in the zinc precipitation of the precious metals. In weak cyanide solution the zinc in solution may be considered to take the form of a hydroxide ($Zn(OH)_2$), or the single zinc cyanide ($Zn(CN)_2$), both of which are insoluble in water and only slightly soluble in weak cyanide solution. In strong cyanide solution the zinc may be considered to exist as the double zinc potassium cyanide ($K_2Zn(CN)_4$) which is highly soluble, and in alkaline solutions of the alkalis as a zincate ($Zn(KO)_2$) which is also very soluble. With weak solution the zinc shavings will be more or less coated with zinc hydroxide and zinc cyanide, so that little dissolution of the zinc takes place under conditions to replace the gold in the double gold potassium cyanide ($KAu(CN)_2$), or to chemically set up the electric currents that assist in depositing the gold. This results in poor precipitation and the formation of the white precipitate of the zinc boxes, consisting mainly of zinc hydroxide and zinc cyanide. The use of a strong cyanide solution dissolving the white precipitate of hydroxide and simple cyanide to form the soluble double cyanide and actively dissolving the zinc, keeps it clean and promotes good precipitation. Lime or alkalinity may be used in the same way as cyanide for this purpose, with the exception that the soluble zincate is formed instead of the double cyanide. The subject is further discussed and the reactions given under Precipitation.

While a protective alkalinity is usually necessary for good precipitation, at least with weak solution, too high an alkalinity must be avoided as it causes an excessive action on the zinc, as:

$$Zn + 2\,KOH = Zn(KO)_2 + 2\,H,$$

resulting in an increased consumption of zinc and the production of much hydrogen, which may cause an undesirable disturbance in the zinc boxes and poor precipitation, by the entangled hydrogen bubbles preventing good contact between the solution and the zinc. Too high a protective alkalinity may cause a deposition of lime salts in the zinc boxes, covering the zinc with a flocculent or hard precipitate which interferes with precipitation, and increases the bulk of the clean-up and melt, even with sulphuric acid treatment, for the calcium sulphate formed is very insoluble.

Lime as a Neutralizer of Carbonic Acid. — Carbonic acid (CO_2) decomposes cyanide into hydrocyanic acid, as:

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

The hydrocyanic acid liberated is regenerated into cyanide by any alkali present, or the carbonic acid is itself neutralized, as:

$$\left\{\begin{array}{l} HCN + KOH = KCN + H_2O. \\ CO_2 \quad + KOH = KHCO_3. \end{array}\right.$$

The source of carbonic acid or carbon dioxide may be the atmosphere, the air used in agitation and aeration, organic matter, carbonate ores, etc. The air used in agitation, besides perhaps increasing the cyanide consumption to a small extent through its carbonic acid, increases the consumption of alkali very materially, forming insoluble calcium carbonate ($CaCO_3$) in the case of lime. This coats or clots the filtering canvases of the leaf filters so that they must be treated with dilute hydrochloric acid to dissolve out the calcium carbonate as a soluble calcium chloride. A high protective alkalinity is not used in many plants for this reason alone. The use of a high protective alkalinity may also give trouble in this way, by coating and gradually closing the solution pipes with a deposit of calcium carbonates and other alkaline earths.

Dissolving Effect of Alkalis upon Metals. — Alkalis have an important influence in cyaniding, through their tendency to act upon the base metals to convert them into oxides and hydrates. This action is noticeable in connection with nearly all the base metals. By altering the metals into the softer oxides and hydroxides, and by decomposing and breaking down and altering the compounds, it causes a greater action in many cases — in those in which it does not act to produce, or to the extent of producing inert salts and oxides — between the substance and the cyanide. By breaking down the compounds it better liberates for easy dissolution the precious metals that they have chemically and mechanically imprisoned. These effects are to a large extent undesirable in treating gold ores, since gold is invariably in a native or metallic form and any increased liberation of gold — which would be from the sulphides in most cases — would be overcome by the deleterious effects of the greater reactions between the base metals or compounds and the cyanide, the mechanical liberation by fine-grinding being preferable. With silver ores this effect is in many cases highly desirable, for silver is generally both chemically combined and mechanically alloyed or held with other substances. The effect of high protective alkalinity in treating gold ores is to lessen the extraction and increase the time of dissolution. High alkalinity or lime decreases the solubility of both native gold and silver in cyanide solutions, and where sulphides and base metal compounds are found, will cause more alkaline sulphides to form in the solution (see Alkaline Sulphides and Sulphocyanides) and the base metals to be more acted upon. For this reason a high protective alkalinity when treating sulphide gold ores is especially undesirable, but of value in the working of sulphide silver ores in which the silver and sulphur are chemically combined. Cases have been noted in working sulphide gold ores where solutions slightly "acid," showing no protective alkalinity, have given a higher extraction than those having a protective alkalinity, though at the expense of an increased cyanide consumption.

Amount of Lime or Protective Alkalinity Required. — The amount of lime or protective alkalinity used will vary with the material treated, the method of cyanidation used, and the ideas of the metallurgist in charge. Even the metallurgist may vary the amount greatly, as the result of careful study of the plant practice and laboratory experiments. Where it is necessary to assist the settling of the slime by using lime, the results when using varying quantities are noted, and that amount used which will give good settling results with a reasonable quantity. The amount used for neutralizing purposes depends upon the nature of the ore, and indirectly upon how the lime is applied. In the treatment of gold ores it is aimed to have only a slight protective alkalinity. It will be attempted in the average gold plant to keep the protective alkalinity of the solutions at some standard between .005 per cent (.1 pound) and .025 per cent (.5 pound) lime (CaO). Except where necessary for settling purposes, solutions containing an average protective alkalinity of more than .04 per cent (.8 pound) CaO will seldom be found in gold plants. On the other hand, in the treatment of silver ores a high protective alkalinity is generally used, ranging from .025 per cent (.5 pound) to a maximum or saturated solution of lime, .125 per cent (2.5 pounds), and even a little higher where the pulp or solution contains other alkaline constituents. Some plants treating silver ores use that protective alkalinity in the solution or that quantity of lime which will cause the mill solution titrations for free cyanide and total cyanide to approximately coincide, — usually a case of a protective alkalinity of about .1 per cent (2 pounds) CaO — the principle and some discussion of which has been given under Total Cyanide. Plants treating gold ores will usually use from 1 to 5 pounds CaO per ton of ore treated, and silver plants from 3 to 10 pounds, though as high as 20 pounds are being used.

Methods of Adding Lime. — The lime is added in cyanide practice in various ways. It may be added dry to the ore in the bins, or fed dry or wet as a milk of lime into the crushing and grinding machines. Feeding the lime into the fine-crushing machinery is an excellent method when crushing in solution or when the mill water is circulated for reuse, as it allows early action upon the cyanicides of the ore, and yet there is no mechanical loss. When used in this way, the pipes returning the water or solution for reuse in crushing should be large and easily taken down, for they may be gradually lined with a deposit of lime and alumina salts that must be periodically removed. These pipes have sometimes been replaced with open wood troughs. Lime often causes trouble by coating the mill screens and blinding their openings, especially with woven-wire screens. A change to the punched or slotted-plate type may remedy the difficulty. Lime has in some cases given trouble on the amalgamating plates, but this can usually be overcome, especially if a thick bed of amalgam, free from substances that the alkali may alter, is kept on the plates.

A favorite method of adding the lime is after the crushing, but just before the pulp reaches the cyanide tanks, supplying the lime freshly wet-crushed from a single-stamp battery or grinding pan fed by an automatic feeder; or from a small tank fitted with agitator blade, ascending current of water, or other method of agitation, into which lime is dumped at intervals. Lime is sometimes added in unslacked form directly to an agitation charge, but should first be slacked and then added as a milk of lime. When leaching vats are filled with dry ore, the lime is distributed, invariably slacked in a dry, powdery form, in small lots to be well mixed with the charge; this is the most effective way of adding lime. Where the sand vats are filled with wet, flowing pulp, the necessary lime is usually added crushed wet into the pulp stream as stated before. Much of the finer and the quickly-dissolved portion of the lime overflows the vat and is lost, unless the mill water is reused or crushing in solution is practiced. Small quantities may be added by the inconvenient method of sprinkling it over the top of the charge and raking it in. A method which can be resorted to is to add the lime to the stock solution tanks, thus making the cyanide solution a strongly saturated solution of lime water. This method is open to criticism as temporarily giving an inordinately high protective alkalinity where a constant low one is desired. Yet on ore containing many cyanicides, a solution low in cyanide and strong in alkalinity is often used first on a leaching charge, that the cyanicides may be cheaply neutralized before adding the stronger cyanide solution.

The method of adding the lime deserves careful consideration. If milk of lime is added to the pulp flowing to leaching vats, the overflow of which runs to waste, much of the lime will be lost. If added in granules, they will sink with the grains of pulp, become imbedded in the charge, and gradually dissolve and give off their alkali, thus doing effective work. The size of the granules of lime required will vary with the nature of the ore and its treatment. They should be of such size that the lime will dissolve and give off its neutralizing power as fast as needed. This will be indicated by the solution issuing from the vat. It should at all times have a small protective alkalinity, but at no time an inordinately high amount, yet on the principle of economy the lime should be all dissolved and used by the time the treatment is finished. The cyanicides of some ores show themselves very fully at the start, others, especially heavily sulphuretted ores, gradually undergo oxidation and develop cyanicides and alkali and cyanide-consuming compounds during the whole time of their treatment. Consequently, each case is a separate problem of both how to add the lime to the best advantage in view of its influence on the dissolution and precipitation of the precious metals and the consumption of cyanide, and how to get the maximum efficiency of the lime. Between adding lime in unslacked coarse granules and as a milk of lime is a wide variation.

Lime v. Caustic Soda. — Caustic soda (NaOH) was at one time used extensively for neutralizing purposes, but lime was generally found to be superior as the salts or compounds of lime — mainly calcium sulphate — are very insoluble, while those of caustic soda are extremely soluble. In this way the lime salts are precipitated in the ore where formed instead of entering the solution, where they may exert some influence and perhaps be precipitated in the zinc boxes as sometimes occurs in using caustic soda. The excessive use of caustic soda as a neutralizer will cause trouble with the zinc precipitation much quicker than the use of lime. One advantage of caustic soda — when such an advantage is desired — is its extreme solubility; it will dissolve in an equal weight of water, while 1 part of CaO requires 800 parts of water. Pure lime has 1.43 times the neutralizing strength of caustic soda and is much cheaper.

Determination of Causticity of Lime, etc. — Commercial lime is always more or less impure, containing varying amounts of sand, clay, iron, carbon, etc. These were contained in the original limestone or are due to the fuel. The available or useful alkali may be roughly estimated by taking a weighed and powdered sample of the unslacked lime, mixing it with water to form a very dilute and liquid emulsion containing a definite per cent of the commercial CaO, and titrating it, as in the test for protective alkalinity. The comparative efficiency of the lime when dissolved or slacked in cold water and in hot water should be tested by titrating a similar lime-test solution that has been boiled, for grinding or thoroughly disintegrating the lime in hot water will usually give a higher alkalinity. For more accurately obtaining the causticity or available CaO or alkali, weigh out 2 grams of the powdered unslacked lime, make up to 1000 c.c. with water and 20 grams of pure cane sugar, shake at intervals over a period of 12 hours, and finally remove an aliquot part and titrate with the decinormal acid and phenolthalein.

If it is feared that the lime contains reducing agents that will destroy the cyanide, a clean, new, cyanide solution should be made up for test purposes, titrated for its strength, and lime added in varying quantities with titrations to determine if the cyanide strength is reduced. Lime sometimes gives trouble by the carbon and carbonaceous matter in it or introduced in the burning process, precipitating the gold and silver. Agitating lime with metal-bearing solution, either in the laboratory or in the gold stock tanks, with assays of the solution before and after, will indicate regarding this.

CHAPTER VII

ORE TESTING AND PHYSICAL DETERMINATIONS

In making laboratory tests on ore for the extraction of its gold and silver by cyanide, the principles governing and the points arising in the actual working of the ore must be borne in mind, rather than an attempt to imitate the exact details of plant practice. The experienced cyanide operator in making tests on ore, while employing careful laboratory methods, observation, and study, visualizes the sample of a few pounds into a full-sized working charge, correlating each detail occurring in the sample to that which would take place in a working charge.

Facts to be Determined. — The principal things to be ascertained or to be examined into are: nature and composition of the ore; nature of the metal and the condition in which it is held in the ore; special treatment which may be necessary, as roasting, water-washing, aeration, removal of cyanicides, amalgamation, concentration, etc.; amount of lime or other neutralizer required; strength of cyanide to be used, with probable quantity required, and consumption that will take place; time required for dissolving the gold and silver; fineness of crushing required for an economically high extraction, and the variation due to crushing to varying degrees of fineness, including the results of sizing tests showing the amount of different mesh material produced, and the value of each before and after treatment; tendency of ore to slime, and quick or slow settling effect; also how it will percolate.

Methods of Testing. — To work these tests out fully requires a great amount of labor and time. It is customary to start with small bottle tests of perhaps a few ounces or pounds, and after the characteristics of the ore have been learned to increase to 25 to 100-pound lots; and finally, especially if the ore is a silver one or shows any refractory tendencies, to test at a custom testing plant or in a small experimental plant at the rate of 500 pounds or a few tons per charge. Tests in bottles are easily made and will quickly exhibit to the experienced operator the lines along which the metallurgical system is to be developed. They are insufficient to build a plant on, even presuming that the same dissolution would be effected in the plant as in the preliminary tests, for cyanide and crushing plants and processes are not fully standard to all classes of material, and consequently should be designed to meet the necessities of the ore, which must be learned by extended experimental work.

Securing Samples. — The first, and probably the hardest thing, unless under the direction of an experienced man alive to the dangers, is to get proper samples of the ore. The tendency is to take samples that are too well oxidized, for the ore prominently exposed during the early days of a mine is the oxidized ore near the surface. Often the samples are taken from dumps that have long weathered. Tests on this nature of ore will invariably indicate a high extraction with coarse crushing and either without or with but little concentration. Whereas, when the harder, unoxidized, baser ores are worked, there is a lower extraction obtained, a finer crushing required, and after a lapse of considerable time the operators awake to the fact that concentration or closer concentration by a more elaborate concentrating plant, or changes to give greater attention to the sulphide content of the ore, will increase the profits to a large extent. In some cases the samples represent too fine a material, in others too coarse. This may result in increasing or decreasing the apparent value of the material to be treated or the amount of some constituent in it, or, by giving undue proportions between the slime and sand, may cause the installation of an unsuitable system. When the mine contains different classes of ore, as oxidized and unoxidized, clean "free milling" and base, high and low grade, those that slime and those that are hard and dense, and separate shoots or ledges containing copper, zinc, lead, etc., each class should be tested separately and not averaged together. This distinction should be borne in mind after the plant is in operation, for a plant using cyanidation may, like a smelter, find it desirable to mix certain ores in some cases and keep them separate in others, to get a certain proportion of sand and slime, or to treat an easily worked ore differently from a refractory one.

Physical Examination of Ores. — The ore to be tested should first be closely examined and studied, for its characteristics, conditions under which it is found, the methods employed upon similar ore in the same or other districts, and the results from assays, pannings, microscopical examination, etc., will indicate the nature of treatment that will probably be required. If it is a soft, porous ore which a solution can easily penetrate, coarse crushing such as is done in a dry-crushing mill may be sufficient. If it is a hard, dense ore which the solution cannot penetrate, fine crushing will be required to liberate the metal to the solvent action of the cyanide. The metal may lie on the breaking or parting planes of the ore or on the faces of the crystals, in which case extremely fine crushing will not be required to expose it; whereas when it is embedded in the crystals or sulphide, very fine crushing is required. Where the metal is in coarse grains, it must be removed by amalgamation or careful concentration, or a long contact with a strong solution will be required to dissolve it, unless it is ground into small particles or thin scales by a tube mill or other slimer. Where the metal is in an extremely fine state of division or in very thin scales, a low strength of solution will dissolve it quickly. If the ore contains limonite, kaolin, alunite, talc, etc., that becomes a colloidal, slow-settling, and unmanageable slime, only the leaf filter can handle it to an advantage. Where the ore contains sulphides, it becomes a question whether to remove them or not before cyanidation. It is customary to remove them by concentration when they represent a considerable proportion of the value, or introduce interfering substances into the ore. A sulphide or base ore will generally necessitate finer grinding to liberate the metal. Telluride ore is invariably treated by roasting or bromocyanide. Ore containing copper may require the removal of the copper by concentration or by leaching with very dilute sulphuric acid before applying the cyanide, while other copper ores may be successfully worked by using low-strength solutions that will act less strongly upon the copper. Ores containing antimony, and to a less extent manganese or arsenic, may give low extractions and require aeration, etc. Sulphide ore may require considerable aeration. Clean and unoxidized ore will require little lime or other neutralizer of the acidity, while oxidized and base ore will require a large amount.

Free Acidity. — The first test that may be made is for free acidity or that which is soluble and can be washed out of the ore. Take 20 grams or more of ore ground to the mesh expected to be used. Add the same number of cubic centimeters of water and agitate for ten minutes or longer. Filter and titrate 10 c.c. of the filtrate with decinormal caustic soda solution until the solution becomes alkaline as described under the test for hydrocyanic acid and protective alkalinity. This will give the number of pounds of caustic soda or lime required to neutralize the soluble acidity in one ton of water, and if the same weight of water as of ore was taken, the results indicate the caustic soda or lime required per ton of ore to neutralize the free or soluble acidity.

Latent Acidity. — Latent acidity, or that which is insoluble, is determined by washing a weighed sample or that used in determining the free acidity until the washings show no acidity, then adding some water to the ore and the alkaline indicator, and titrating with the decinormal alkali solution until the ore solution becomes alkaline. The results are figured in the same way as for free acidity, except that the titration is computed for the number of grams of ore used. A method giving higher and more correct results consists of diluting with the necessary amount of water and adding standard caustic soda solution in some excess of that required to make alkaline, agitating for half an hour or longer, filtering and washing ore until no more alkalinity, then titrating to neutrality by standard acid. As a standard acid and alkali exactly equal each other, the difference between the alkali added and the acid required to neutralize the excess, will give the latent acidity.

Total Acidity. — The total acidity, that due to both the free and latent, which is what is usually required, is found as for latent acidity without first water-washing. Where the ore to be tested is wet, it should not be dried, for more acidity will be generated from the iron, pyrites, etc.; but the tests should be made first, after which the ore is dried and weighed for calculating the results.

These tests for acidity will indicate with little accuracy concerning the amount of lime required, for more will be used in practice, owing to the large proportion of impurities in the lime, — which should always be borne in mind, — to the lime not dissolved or uneconomically used, and to further acidity which may be generated. A closer approximation of the amount can be obtained through tests made by the use of bottles, introducing 20 grams or more of ore into each, together with the same number of cubic centimeters of a fresh cyanide solution of working strength and varying weighed quantities of lime, as .05 (1 pound per ton of ore), .15 (3 pounds), or .25 per cent (5 pounds) or more of the weight of ore, agitating for 30 minutes or longer, filtering and testing 10 c.c. of each for cyanide consumption and protective alkalinity. That solution which indicates a protective alkalinity below .3 per cent (.6 pound) CaO will probably show a low consumption of cyanide, and indicates the amount of lime to be used on clean ores, though as the ore becomes baser the lime and cyanide consumption cannot be estimated from such short contacts or so generally — the actual practice must be imitated.

Extraction Tests with Bottles. — The most important thing in all cyanide tests is to learn the highest dissolution of gold and silver that can be effected. Bottle tests on gold ores will usually give this as closely as a working charge of several hundred tons, if the sample is representative, in the same way as a small portion taken for assay represents a carload or a day's run of ore. As the ore becomes baser and with silver ores, bottle tests cannot be relied upon so strongly, for on account of the comparatively slow dissolution of the precious metals in such ores the maximum dissolution may not be effected. Bottle tests, besides giving the maximum extraction, indicate less accurately the strength of cyanide to be used, the consumption that will take place, the time required for dissolution, and the amount of lime to be used; also, by testing the solution, the nature of the cyanicides.

For making simple bottle tests on gold ores, take wide-mouthed bottles and introduce into each 2 assay tons of ore ground to the mesh expected to be necessary, together with the amount of lime or neutralizer that the bottle test before described shows to be necessary to give a small protective alkalinity. In the absence of having made this test, use an amount in excess of that indicated by the test for total acidity. Add 120 c.c. of a new cyanide solution to each bottle, making the first .1 per cent (2 pounds) KCN, the second .175 per cent (3.5 pounds) KCN, and the third .25 per cent (5 pounds) KCN. If it is a clean ore containing the gold in a finely-divided state, a charge should be tried using a strength of .05 per cent (1 pound) KCN. With a base gold or an easily-worked silver ore a charge of .375 per cent (7.5 pounds) KCN should be tried, while with a base silver ore or a very base gold ore, the test should be made with solutions from .1 per cent (2 pounds) to .6 per cent (12 pounds). These bottles should be agitated for 24 hours, or left stand with occasional shakings for 48 to 72 hours. If the ore contains the gold in a finely-divided state and little sulphide, the value will be dissolved within 12 hours. If the ore is very base, it may require more than 72 hours' contact, unless continuous agitation is given. Some silver ores may require a few days' continuous agitation or contact for a week with strong solution. The bottles should be uncorked at times to aerate. At the end of the period, filter each charge and test for cyanide consumption and protective alkalinity, which can be reduced to tons of ore, since each assay ton of ore was treated with approximately 2 assay tons of solution. Wash the ore by decantation or on a filter for some time after all alkalinity is removed,—a thorough washing is most essential. Finally dry and assay. At the same time as making these tests it would be well to grind a sample of the pulp to 200-mesh and treat it with a strong solution (.375 to .5 per cent — 7.5 to 10 pounds — KCN) with a few decantations and additions of new solutions, that the results of the residue assay may be taken as the maximum extraction under ideal conditions for comparison with the regular bottle tests. With silver ores and sulphides, lead acetate at the rate of 1 pound per ton of ore should be added to the charge or solution to remove any alkaline sulphides formed. If the ore reaches the cyanide plant wet and without drying, it should be treated that way in the test, for a preliminary drying would oxidize the ore and probably give a higher extraction than the plant could in actual practice. The difference between the assays of the sample before and after treatment gives the extraction; this may be checked by drawing off 30 c.c. or any aliquot part of the solution and assaying. Or the effect of further contact with the solution may be tested by drawing off or removing the aliquot portion of the solution and assaying the same, followed by retreating the ore with fresh solution before washing and drying it for assay. The bottles may be continuously agitated by being attached to some suitable moving device, as a wheel, in which case the extraction will take place quickly, just as agitation in actual practice causes the value to go into solution in a comparatively short period.

Agitation tests with lots of 2 or 3 pounds of ore may be conveniently made in large acid bottles. The amount of solution used should be from two to four times that of the dry ore by weight, or even more. An air-agitating tank may be constructed in various ways, such as by cutting off the bottom of a large wine or other bottle having a long, sloping neck. This is placed in an upright position with neck down. A $\frac{3}{4}$-inch glass tube is suitably supported from near the bottom of the neck to within 2 inches of the top. Through the cork in the neck of the bottle passes a $\frac{1}{4}$ or $\frac{1}{8}$-inch glass tube delivering air into the bottom of the $\frac{3}{4}$-inch glass tube. A charge sufficient to nearly submerge the central tube is placed in the agitator and a slight amount of air turned on, resulting in circulating the pulp up the central tube and down the outside of it. Any modification of this — the Pachuca air-lift tank principle — may be used.

In agitation tests, samples should be taken hourly or every few hours by means of a tube, preferably of glass, inserted to the bottom of the charge, the upper end closed with the finger and quickly withdrawn, or some of the pulp may be syphoned off. Care must be exercised to get a true sample of the pulp and not one containing too little of the coarser material, nor should the amount removed be sufficient to vitiate the final sample. The sample should be tested for cyanide strength, protective alkalinity, and gold and silver. If the consumption of cyanide or alkalinity is large, it may be necessary to add more during the agitation.

Percolation Tests. — Percolation tests more closely imitate leaching practice and consequently are often used. These are made in glass percolators or by using large acid or smaller bottles as such by cutting their bottoms off, the discharge of the percolator being fitted with a rubber tube and a pinchcock. A filter bottom of muslin or light canvas is arranged on a platform in the bottom of the percolator, after which the charge of ore containing the proper amount of neutralizer is added; this may be as small as 2 pounds if crushed medium fine. There is set above the percolator a vessel containing the cyanide solution to be used, which may be drawn off by a small cock or syphoned out by a rubber tube, the stream of which is regulated by a pinchcock. If it is desired to measure the solution, an amount by weight equal to twice the amount of ore is convenient. Thus if 600 grams of ore have been taken, 1200 c.c. of solution may be used. The ore in the percolator is covered with this solution, after which the discharge cock is opened to allow a drip sufficient to drain the entire amount of solution through the charge in the allotted time, which is usually three days with simple gold ores, and longer with very base and silver ores. The solution in the reservoir being allowed to drip into the percolator at a rate sufficient to keep the charge covered. Occasionally the entering drip should be shut off to allow the charge to drain and to aerate for a short time, after which the charge should be covered with solution and percolation started. After final percolation and drainage, the charge should be thoroughly washed past the point where the washings show not even a trace of alkalinity, when it is dried and assayed. The solutions and washings are saved, measured, titrated for cyanide strength and protective alkalinity, assayed for gold and silver, and the results figured out. By using a good sized charge and sampling at regular periods when drained by a sampler resembling a cheese drier, by plunging a tube into the charge, or by removing the charge, the progress of the dissolution of gold and silver can be noted. These samples must be well washed without any delay. By running several percolation tests together on parts of the same sample, using different strengths of solution and sampling at regular intervals, the necessary data regarding strength of solution to be used, consumption of cyanide, and time required for dissolution can be learned, but not the degree of fineness to which the ore should be crushed.

Fineness of Ore Required. — To learn the degree of fineness to which the ore should be crushed, the simplest forms of bottle tests that use a solution sufficiently strong and a long enough application to get the maximum extraction is all that is required. The ore should be crushed and ground to those of the following sizes which may be deemed necessary, 4, 10, 20, 30, 60, 100, 150, and over 200-mesh. The results of cyaniding those sizes that give a good extraction should be shown in a comparative way by plotting, etc., and studied, for that degree of fineness which gives the highest extraction may yield less profit than when crushing to a coarser size, on account of the increased cost of crushing finer and cyaniding the finer material.

Sizing Tests. — The last tests to be made are sizing tests. These occupy some time and are usually not made until the preliminary tests are well worked out, though it is an advantage to make them at all times. A sample of the crushed ore to be tested by cyanide amounting to about 2 pounds is taken and weighed. It is first concentrated and reconcentrated until all sulphide is removed. It is then stirred up with the water used in concentrating and the muddy water poured off, care being exercised that no fine sand passes off with the water, when it is again stirred up with water and the muddy water poured off. This is repeated and repeated until the sand is washed entirely free from slime that is of a light, flocculent, feathery nature, that agglomerates together, makes water muddy, and does not readily settle; while the slime product contains no sand or granular matter, however fine. The slime is allowed to settle, the water poured off, and the sludge either dried in a pan or run onto a filter paper and dried after draining, when it is weighed. The sand and concentrate are also dried, after which they are sized through screens. To divide the concentrate into two sizes and possibly three is good. The sands should be sieved into at least four sizes, the coarsest size containing not more than 5 or 10 per cent of material that may be considered as a coarse oversize, — as that which failed to be crushed to the desired mesh,— while screens should be used that will divide the remainder about evenly. Taking the case of material crushed in a mill through a 40-mesh screen, the sizing test should produce the following sizes: a concentrate or a coarse and fine concentrate, a slime, an oversize of 5 or 10 per cent held on a 40-mesh laboratory screen, a held on 60, 100, and 150-mesh, and a passed 150-mesh sand; these sizes should be weighed, assayed, and the results tabulated. If the ore contains free gold liberated by the crushing, this sizing test is of little or no value unless the gold is removed by amalgamation or panned out with the concentrate, preferably removed by amalgamation.

The remainder of the sample from which the sizing test and the head assay sample were taken is now cyanided in the laboratory until the maximum extraction is obtained, which is best performed by introducing the ore and solution into a large acid bottle and agitating it intermittently or continuously as usual. After the charge is washed free of dissolved gold and silver, it is sized as before, the results being compared with those of the ore before cyaniding, and both with other tests in which the ore was crushed finer or coarser. The sizes obtained of the sample before cyaniding may be assayed, then cyanided separately, and the residues assayed, or the coarser sizes after cyaniding may be recrushed and recyanided and the results observed. The whole purpose of sizing tests is to show in what part of the ore the value lies before and after cyanidation, and the effect of coarser or finer crushing on the extraction. While the straight cyanide tests made on the ore as crushed to different degrees of fineness will show the increase or decrease in extraction, so that the most economical size can be selected, the sizing tests are necessary to a true diagnosis of conditions. To act without them is too much like a physician prescribing for a sick man without learning the nature of the complaint. It may be that by crushing only the oversize an increased economical extraction can be had, or all sizes of the sand may respond to finer crushing, or the sulphide or coarser sulphide only may require finer crushing. Knowing exactly where the trouble is, the metallurgist can provide to meet that point. When studying sizing tests or the effect of crushing to different degrees of fineness, the metallurgist must consider more than the conditions directly affecting cyanidation. He must also consider the crushing devices at his disposal or which can be reasonably installed. To make a practical success he must bring such knowledge and study to bear that he can adjust the conditions necessary to secure a high extraction by cyanidation to those necessary to obtain a high tonnage at a reasonably low cost from the crushing machinery, and find the economic mean of the two. While many in making sizing tests include the flocculent slime in the finest sand, the remaining sands should always be washed free of the adhering slime, this slime to be added to the finest size. The following will indicate the method of tabulating the results.

Sizing Test

Head Sample — Charge No. 76. Jan. 10, 1911.
Silver at $0.50 per ounce.
Assay: Gold, $3.40; silver, 4.6 ounces, $2.30; total, $5.70.

	Weight.		Gold Value.		Silver Value.			Gold and Silver.		
	Grams.	Per cent.	Per Ton This Size.	Represents in Ton of Ore.	Per Ton This Size.		Represents in Ton of Ore.	Per Ton This Size.	Represents in Ton of Ore.	Per cent Value of Ton of Ore.
					Ounces.	Dollars.				
			Dollars.	Dollars.			Dollars.	Dollars.	Dollars.	
Held on 60-mesh	29.5	5	3.00	.15	4.11	2.06	.10	5.06	.25	4.6
Held on 100-mesh	88.5	15	1.50	.225	1.80	.90	.135	2.40	.36	6.7
Held on 150-mesh	71.	12	1.20	.145	1.60	.80	.095	2.00	.24	4.5
Held on 200-mesh	106.	18	1.30	.235	2.00	1.00	.18	2.30	.41	7.6
Passed 200-mesh	59.	10	1.30	.13	1.84	.94	.095	2.24	.23	4.3
True slime	207.	35	1.80	.63	2.62	1.31	.46	3.21	1.09	20.3
Concentrate on 100-mesh	17.75	3	30.80	.925	36.34	18.17	.545	48.97	1.47	27.3
Concentrate passed 100-mesh	11.75	2	38.60	.79	52.10	26.05	.54	64.65	1.33	24.7
	590.5	100		3.23			2.15		5.38	100.0

Sizing Test

Tail Sample — Charge No. 76. Jan. 18, 1911.
Silver at $0.50 per ounce.
Assay: Gold, $0.80; silver, 1.5 ounce, $0.75; total, $1.55.

	Weight.		Gold Value.		Silver Value.			Gold and Silver.		
	Grams.	Per cent.	Per Ton This Size.	Represents in Ton of Tailing.	Per Ton This Size.		Represents in Ton of Tailing.	Per Ton This Size.	Represents in Ton of Tailing.	Per cent Value of Ton of Tailing.
					Ounces.	Dollars.				
			Dollars.	Dollars.			Dollars.	Dollars.	Dollars.	
Held on 60-mesh................	28.75	4.5	1.20	.05	2.12	1.06	.05	2.26	.10	6.9
Held on 100-mesh...............	96.	15.	1.00	.15	1.14	.57	.085	1.57	.235	16.2
Held on 150-mesh...............	64.	10.	.60	.06	.80	.40	.04	1.00	.10	6.9
Held on 200-mesh...............	127.75	20.	.60	.12	.66	.33	.065	.93	.185	12.7
Passed 200-mesh................	76.50	12.	.40	.05	.68	.34	.04	.74	.09	6.2
True slime.....................	210.75	33.	.20	.065	.48	.24	.08	.44	.145	10.
Concentrate on 100-mesh.........	19.25	3.	6.20	.185	10.24	5.12	.155	11.32	.34	23.4
Concentrate passed 100-mesh.....	16.	2.5	4.80	.12	11.10	5.65	.14	10.45	.26	17.8
	639.00	100.0		.80			.655		1.455	100.1

Comparison of "Head" and "Tail" Sizing Tests

Charge No. 76. Jan. 10–18, 1911.

	Extraction per Ton Each Size.				Extraction per Ton of Ore.
	Gold.	Silver.	Total.		
			Per cent.	Dollars.	
	Per cent.	Per cent.			Dollars
Held on 60-mesh	60.	45.5	54.1	2.74	.15
Held on 100-mesh	33.3	36.7	34.6	.83	.125
Held on 150-mesh	50.	50.	50.	1.00	.14
Held on 200-mesh	53.8	66.	59.5	1.37	.225
Passed 200-mesh	69.	64.	67.	1.70	.14
True slime	88.9	81.7	86.5	2.77	.945
Concentrate on 100-mesh	79.9	71.8	76.9	37.65	1.13
Concentrate passed 100-mesh	87.	78.1	83.8	54.20	1.07
General extraction by sizing test			72.95%		3.925
General extraction by assay			72.81		

Amalgamation Tests.* — For making amalgamation tests, two methods may be followed. The first is to place 6 or 8 assay tons of the ore crushed to the desired mesh in a large glass bottle with sufficient water to make a thin pulp, adding ½ ounce of mercury. The pulp is rolled around in the bottle, is lightly shaken, and is given a panning motion for some time, that all the free gold may be amalgamated. The contents are finally washed out of the bottle, panned and repanned until the amalgam is separated from the pulp, when the tailing is dried and assayed, the difference between the head and tailing assay representing the amount amalgamated. If mercury that contains no gold has been used in the test, the gold in the amalgam can be determined by boiling the amalgam in dilute nitric acid until only the pure gold remains, when it may be washed, dried, annealed, and weighed as usual in the gold assay; or the amalgam may have the mercury driven off by heating it in the open where there is no danger of salivation, and cupeling the resulting sponge, which will give both the gold and silver amalgamated. Mercury entirely free from gold can seldom be obtained, but can easily be prepared by dissolving it in dilute nitric acid, when the gold remains undissolved and can be filtered off, while the mercury

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* From "Practical Stamp Milling and Amalgamation," by the author.

can be precipitated by suspending a piece of copper in the solution.

A better method of making an amalgamation test is to work the ore as a thin pulp in a gold pan having an amalgamated bottom, assaying before and after treatment; the pan being used to separate any sulphide present at the same time. Laboratory amalgamation tests as a rule will not give as high an extraction as will be obtained in actual mill practice. This may be due to the fact that in preparing ore for such a test, it is screened frequently, resulting in an evenly-sized material, whereas in actual practice a large proportion is crushed much finer and should give a higher extraction. It is also possible that the dry-crushing may coat the gold with dirt or slime so that to some extent it resists amalgamation.

Tests akin to amalgamating in cyanide solution can be made by introducing the mercury into the bottle in which the cyanide test is being made and manipulating in the way usual to both tests, removing the amalgam when removing the cyanide solution. While this will probably give the same tailing as will be obtained in actual practice, it will undoubtedly give a lower extraction by amalgamation, due to the slowness with which the gold is brought in contact with the mercury and the long continuous though slow action of the cyanide solution upon the amalgam.

Tests on Concentrate. — In making tests on concentrate, it should be well washed and the proper amount of lime added to give the solutions a protective alkalinity. If the tests are to be by percolation, a .3 (6 pounds), .6 (12 pounds), and a .9 per cent (18 pounds) KCN strength of solution should be tried, while lead acetate equal to 1 or 2 pounds per ton of concentrate should be added to the solution to precipitate any alkaline sulphides formed. The solution for each test should be kept separate, should be run through zinc shavings, well aerated, restandardized, and reused. The amount of cyanide used on each test should be carefully noted, and that remaining in the solution after the test should be determined, thus enabling the cyanide consumption to be arrived at. The leaching should be continuous and the charge drained at least two or three times daily to allow good aeration, for this is most necessary and should be done mechanically if possible. The treatment with strong solution should continue for twenty days and longer, until no further extraction can be secured, after which the charge should be removed, dried by air, and carefully sampled, when it may be returned to the percolator and a short retreatment tried.

The charge should be sampled daily by a cheese trier or tube for the first few days of treatment, after which the sampling periods may be extended to every three or four days. Agitation *versus* percolation should be tried, and agitation tests with various degrees of fine-grinding and strengths of solution should be experimented with. The agitation should be by air or air should be supplied to the charge. Fresh solution at intervals is also desirable. The solutions should be tested for alkaline sulphides, reducing power, nature of cyanicides, etc. The sulphide may be roasted and then leached or agitated, but such roasting should be a " dead " or " sweet " roast, or soluble and insoluble salts, acting as cyanicides, will form and remain.

Summary of Small Ore Tests. — The method of testing an ore may be summarized as follows: Test for free acidity to learn value of water-washing. For total acidity to learn probable amount of neutralizing agent required. Make bottle tests with cyanide solution and lime or other neutralizer in varying quantities and starting with a larger amount than that indicated by the total acidity test. Make bottle or percolation tests with different strengths of solution, also determining the progress of dissolution. Make bottle or percolation tests on the material crushed to different degrees of fineness, and preferably determining the progress of dissolution. Make a test of percolation against agitation on a sample crushed to the fineness found to give the highest economic extraction, to learn the comparative extractions and rates of dissolution. Make sizing tests before and after treatment to determine where the metal lies, and possibly the presence of cyanicides in some particular size. Compare sizing tests of ore crushed to different degrees of fineness. Separate out the coarser sizes and treat separately, one-half without further crushing and the other half after being ground finer. Make test to find the maximum of extraction under ideal conditions, by samples ground to a slime, given an oxidizing roast, well aerated during contact with solution and with much fresh solution and by adding 2 per cent of potassium chlorate ($KClO_3$) or sodium peroxide (Na_2O_2) to the charge as an oxidizer, using a stronger solution and a continued contact, by heating the solution, and by agitation.

The results of these tests will check very closely the extraction made in actual plant practice, but the amount of chemicals consumed in the laboratory test will be very much higher than those used in actual practice, especially with reference to the cyanide.

Tests on Large Scale. — Larger tests than in bottles or percolators should not be made until it has been determined to what degree of fineness the ore should be crushed, what strength of solution should be used, and how fast the dissolution takes place. In short, the larger tests should check the results of the more easily-made smaller tests. These larger tests are usually made in ordinary wood tubs or those obtained by cutting barrels in two. A filter bottom is placed in the bottom, also a cock for drawing off the solution. The tub is well coated on the inside with paraffin or waterproof paint to prevent absorption of the cyanide solution. The ore is charged into the tub or vat and the procedure conducted as with glass percolators and in imitation of actual practice. The charge is sampled periodically to learn the progress of dissolution, while the solution drawn from the ore is tested for gold, silver, cyanide, and protective alkalinity. It is well to make sizing tests in connection with tub tests. Agitation tests in lots of ore of 25 to 100 pounds should also be made if that method of dissolving the gold and silver is to be used. Air-lift agitation tanks for this purpose can easily be constructed. These tests worked out by an experienced metallurgist should be sufficient to warrant erecting a cyanide plant for an easily-worked gold ore, but it is more advisable and with base ores it is necessary to make larger tests. These may be made in a small experimental plant of 500 pounds' capacity or more, *conducted as a regular working plant,* treating many charges, reusing the solution over and over, and finally making a clean-up. Or several tons of the ore may be tested out in a custom ore-testing plant under the direction of a well-experienced metallurgist. Possibly it may be well to treat the ore in a testing plant using the same filtering and other devices as it is proposed to install.

Leaching Rate. — The leaching rate is the rate in inches per hour that the surface of a body of solution standing over a charge falls when the charge is allowed to percolate as freely as possible. The nearest that it can be arrived at is by preparing a pipe, such as a water pipe, of as large a diameter as is obtainable and with a length equal to that of the proposed depth of charge. One end of the pipe is fitted with a leaching bottom followed by a valve. The pipe is placed in an upright position and filled with ore to within 8 inches of the top, either dry-filled or wet-filled with dilute pulp, allowing the grains to settle and arrange themselves more compactly under water, as is the method to be used. The charge is then covered with water and percolation started, the rate of fall of the water being noted. The leaching rate should also be observed after the charge has stood and packed for a few days.

In many cases gold ores can be leached with 1 ton of solution to 1 ton of ore. This means that 32 cubic feet of water (1 ton) must pass through 20 cubic feet of ore (which may be taken as the approximate dimensions of 1 ton of ore laid down in a leaching vat under water). With a charge of ore 1 foot deep, a column of water $1\frac{3}{5}$ feet or 19.2 inches deep would have to pass through. If 4 days or 96 hours are allowed for this, the ore would have to leach at the rate of $\frac{1}{5}$ inch per hour; if the ore charge was 2 feet deep a column of water 38.4 inches would have to be passed through, requiring a leaching rate of $\frac{2}{5}$ inch per hour. From this the following table has been worked out to show the leaching rate in inches required per hour for passing 1 ton of solution through 1 ton of sand laid down under water (20 cubic feet) for different depths of charge and different lengths of time for percolation.

Leaching Rate in Inches Required per Hour to Pass One Ton of Solution Through One Ton of Sand

Hours for Percolation.	Depth of Charge in Feet.					
	4	6	8	10	12	14
96	.8	1.2	1.6	2	2.4	2.8
120	.64	.96	1.28	1.6	1.92	2.24
144	.53	.80	1.07	1.33	1.6	1.87
168	.46	.69	.91	1.14	1.37	1.6
192	.4	.6	.8	1.	1.2	1.4
216	.355	.53	.71	.89	1.06	1.24
240	.32	.48	.64	.80	.96	1.12

A study of the above table in connection with present practice in leaching indicates the correctness of the statement of Julian and Smart in their work on cyaniding, that a leaching rate of 3 inches per hour is good, 1½ inches per hour is fair, and ¾ inch per hour is bad and usually uneconomical. A leaching rate of 1½ inches per hour or better is aimed at in well-regulated plants.

Slime-settling Rate. — Tests regarding the amount of lime for settling and time required are made by taking a sample of the slime containing the solution and dry pulp in the proportion to be used, which is usually four of solution to one of dry pulp by weight. The homogenous slime is divided to a number of glass jars or cylinders, such as 1000 c.c. graduates. To the first no lime is added, to the others lime is added and well agitated in the proportions of .1 (2 pounds), .2 (4 pounds), .3 (6 pounds), .5 (10 pounds), and .7 per cent (14 pounds) of the dry pulp, and allowed to settle. At periods a few hours apart the amount of clear solution in each sample is noted and recorded. At the end of the test all clear solution is syphoned from the sample which has given the most economical results, after which the sample is weighed, dried, and reweighed, to learn to what percentage of moisture the pulp has settled. The results are usually reported as the wet pulp equal to 100 per cent; thus a pulp containing 45 per cent moisture would contain 55 parts of dry slime and 45 parts of solution by weight.

Determination of the Cause of Low Extraction. — The cause of low extraction or the nature and condition of the gold and silver remaining undissolved in the residue after cyanide treatment should be investigated, also the cause of any abnormal consumption of cyanide and other chemicals. The tests for acidity will show something of the nature of the cyanicides of the ore, and may indicate the need of water-washing, though such procedure is not practiced except on old pyritic tailing or sulphide that has oxidized. Sizing tests, etc., may show that the ore requires fine-grinding to liberate the value to the solvent action of cyanide, that the sulphide should be removed for more prolonged treatment with a stronger solution, or that the remaining value is distributed throughout all the sizes and resists dissolution, in which case some method of rendering the metal soluble must be looked for. Stronger solution and longer contact may be required. With very base and arsenical and antimonial ores it is advisable to test the reducing power of the solution, and for alkaline sulphides, for the cause of the low extraction may be the abstraction of the oxygen necessary for dissolution. In the case of telluride ore, roasting is necessary to liberate the precious metal to be dissolved, or bromocyanide may be used. Some sulphides, more especially the arsenical and antimonial ones, may require roasting. An investigation into the enclosed or encased condition of the precious metals remaining after treatment may be made by boiling the treated residue first in diluted nitric acid to remove the silver and then in diluted agua regia to remove the gold, with assays before and after such treatment.

Determination of the Cause of Cyanide Consumption. — The cause of cyanide consumption may be determined by taking 100 grams of the ore with 100 c.c. of new cyanide solution and agitating for 12 to 24 hours, then filtering off 50 c.c. and determining the metals in it which have combined with and destroyed the cyanogen; these will be mainly iron, copper, sulphur, etc. The hydrocyanic acid should also be estimated and the solution examined generally.

Precipitation Tests. — The solution from the tests may be run through zinc shavings in a beaker or other receptacle, care being used that all solution comes in good contact for some time with the zinc. The zinc may then be dissolved in a 10 to 15 per cent solution of sulphuric acid, the resulting gold slime washed, dried, fluxed as in an assay, melted, and the button used to check the results. Tests on precipitation in the laboratory are practically worthless, for only clean solutions are used, while in the plant they become charged with all kinds of compounds. Troubles with precipitation in actual practice can always be worked out, possibly excepting some cases where copper may interfere; but where copper would interfere to this extent, its action would be so pronounced in the laboratory tests as to cause a small plant working a charge of 500 pounds or more to be installed for thoroughly testing out the process preliminary to building the regular plant.

Tests During Plant Operation. — The methods spoken of do not apply alone to testing an ore preliminary to designing and building a plant, but even more so in connection with the operation of a plant. On starting a plant the metallurgist makes many sizing and other tests on the ore and continues to make them at different times. The purpose of these tests is to indicate how to increase the extraction, lower the cost and ease of operating, lessen the consumption of chemicals, and sometimes to increase the daily tonnage capacity of the plant without further equipment and construction. While these tests should be carried on with some regularity and without too great a time elapsing between new sets, they should always be made — and in connection with a watch over and investigation of the solution — when a change in the ore takes place, when a new departure in plant practice is being tried, when the assays of the residue go high, or when erratic results are being obtained. Other things may have an indirect influence, as conditions due to a change from summer to winter, from the dry to the rainy season, or even a change in mill employees who may make some alterations that pass by unnoticed. At some plants the head samples from the sand-vat or agitation charges have been treated regularly over long periods in the laboratory. The head and tailing residue samples are especially suitable for making sizing tests on. Laboratory tests in connection with plant practice have one end in view — what is the highest extraction that can be obtained under ideal conditions and, if the plant is not making such an extraction or duplicating the laboratory results, what is the reason, and can the plant practice be varied in a profitable way to do so? As these tests are made with new solutions they are a check upon the fouling of the mill solutions. It will be found convenient to make up a large quantity of solution, usually somewhat stronger than that used in the plant, in a crock or tub and use it for both dissolving and washing the value out of the ore, making the tests by agitation or percolation, preferably by agitation with suitable devices for easy manipulation.

Specific Gravity Determination. — A specific gravity determination of slime is necessary to compute the amount of dry slime or pulp and of solution in a charge. The specific gravity of any substance is the weight of that substance as compared with an equal volume of pure water at its maximum density (4° C. or 39.2° F.), which is assumed to be the standard and is given a specific gravity of 1.

To find the specific gravity of dry pulp or ore, take a 500-c.c. flask, fill with pure water to the mark, and weigh. Empty the flask, introduce a quantity of dry pulp, add sufficient water to bring to the mark, and weigh. Calculate the specific gravity of the dry slime (which usually ranges between 2.5 and 2.7) by this formula:

$$\text{Sp. gr. dry pulp} = \frac{\text{Weight of dry pulp placed in flask}}{\text{Weight of flask when filled with water} + \text{Weight of dry pulp placed in flask} - \text{Weight of flask filled with pulp and water.}}$$

As an example, if flask filled with water weighs 600 grams, the dry pulp used weighs 100 grams, and the flask when containing both pulp and water weighs 660 grams:

$$\text{Sp. gr. dry pulp} = \frac{100}{600 + 100 - 660} = 2.5.$$

The specific gravity of wet pulp, such as that in a slime treatment charge, is the ratio of its weight to that of the same volume of water. The measuring flask is filled to the mark with a sample of the wet pulp, great care being used that the sample introduced into the flask is of the same composition as that in the treatment charge. If the flask when empty weighs 100 grams, it will weigh 600 grams when filled to the 500-c.c. mark with water. If the flask filled with wet pulp weighs 660 grams, the weight of the wet pulp is 560 grams, and 560 : 500 :: X : 1, or X = 1.12, the specific gravity of the wet pulp, or:

$$\text{Sp. gr. wet pulp} = \frac{\text{Weight of wet pulp}}{\text{Weight of equal volume of water}}.$$

The specific gravity of wet pulp may also be obtained by means of a hydrometer, provided the ore particles remain in suspension and do not quickly settle out of the solution, as would be the case with a pulp containing much or coarse sand.

The percentage by weight of dry pulp in the wet charge is obtained by this formula:

Per cent dry pulp in wet slime

$$= \frac{100\,(\text{Sp. gr. dry pulp})\,(\text{Sp. gr. wet pulp} - 1)}{(\text{Sp. gr. wet pulp})\,(\text{Sp. gr. dry pulp} - 1)}.$$

In the above example:

Per cent dry pulp in wet slime

$$= \frac{100\,(2.5)\,(1.12 - 1)}{(1.12)\,(2.5 - 1)} = \frac{30}{1.68} = 17.9 \text{ per cent.}$$

This gives the per cent of moisture in the wet slime as 100 – 17.9, or 82.1.

The weight of dry slime in a wet charge is obtained by this formula:

Tons dry slime in charge

$$= \text{Cu. ft. in charge}\ \frac{(\text{Sp. gr. dry pulp})\ (\text{Sp. gr. wet pulp} - 1)}{32\ (\text{Sp. gr. dry pulp} - 1)}.$$

In the above example:

Tons dry slime in charge of 1 cu. ft.

$$= 1\ \frac{(2.5)\ (1.12 - 1)}{32\ (2.5 - 1)} = \frac{.3}{48} = \frac{1}{160}\ \text{ton} = 12.5\ \text{lbs.}$$

The weight of solution in a charge may be obtained by the formula:

Tons solution in charge

$$= \frac{(\text{Cu. ft. in charge})\ (\text{Sp. gr. dry pulp} - \text{Sp. gr. wet pulp})}{32\ (\text{Sp. gr. dry pulp} - 1)}.$$

In the example:

Tons of solution in charge of 1 cu. ft.

$$= \frac{1\ (2.5 - 1.12)}{32\ (2.5 - 1)} = \frac{1.38}{48} = .02875\ \text{ton or } 57.5\ \text{lbs.}$$

CHAPTER VIII

PERCOLATION

Definition. — Cyanide treatment by percolation consists in placing the suitably-prepared ore in a tank or vat and allowing the cyanide solution to pass through it by gravity in the process of dissolving and washing out the dissolvable gold and silver. The terms "percolation" and "leaching" are synonymous in cyanidation, though percolation refers more to passing a solution through a porous substance, as sand, and leaching to the dissolving and removing of some substance or substances from the material through which the solution is passing or percolating. The container holding the ore for treatment by percolation may be called a "vat" rather than a "tank," the word "tank" referring to agitation tanks for the treatment of slime and more especially to tanks holding solution.

Percolation of ore divides into two classes: the treatment of ore delivered dry into a vat, and the treatment of ore delivered wet and flowing into a vat. Under the first head comes ore from tailing ponds and deposits, and from dry-crushing mills; under the second head come ore and tailing delivered wet as a flowing pulp from the mill to the vats. The system of collecting in one vat and transferring the drained but still wet sand to another vat for percolation — the double-treatment system — may be said to be a combination of the two.

Treatment of Tailing Deposits. — Practically all of the old tailing deposits have been treated, but work of this nature will always be carried on to some extent, as new mills making a good extraction by amalgamation and concentration are not always in a position to put in a cyanide annex at once. At other places the value of the mill tailing may be considered too low to treat by cyanide unless the output be large, which may require years of development and small capacity to indicate. Mill tailing that has not been cyanided and that contains 80 cents or more in precious metal may be considered as offering opportunities for profitable cyanidation, if in sufficient quantity, and should therefore be stored if not treated at once. Even the discharged tailing residue from cyanide plants, when still carrying material value, may be stored with the hope that the passing of time will render the remaining precious metal soluble through oxidation or otherwise, or that some new process may be devised whereby the value may be obtained.

The principal problem in the treatment of tailing deposits is to get an easily-leached charge in the vats. Most tailings are carelessly banked, resulting in a segregation of the sand and slime in separate parts of the pond, the slime forming a leathery impermeable mass impossible to leach unless by manipulation in connection with the sand. The segregation of sand and slime is owing to the tailing flow being introduced at one point in the pond and the overflowing water being deducted at another point. The heavy sand naturally settles at the point of introduction, the sand diminishing in quantity and coarseness and the amount of slime increasing as the point of overflow is reached, about which is usually pure slime. The segregation in this manner can be prevented by having a number of inflow and outflow points spaced evenly about the pond, changing the flow points daily or oftener. This will result in throwing a layer of sand on a stratum of slime into which it will, to some extent, sink with some evenness over all the pond.

Slime which has settled in large deposits cannot be leached in connection with sand unless it is finely disintegrated. This has sometimes been accomplished by plowing and harrowing the slime and then mixing it with the sand. But this procedure is not always a success owing to the cost, the poor disintegration, and the inability to get a good admixture. Where the sand and slime have been settled in the pond with as thorough an admixture as possible, if the mere process of shoveling or scraping up the material to effect its removal to the leaching vat is not sufficient to disintegrate the slime and mix it with the sand for a good leaching product, plowing and harrowing in connection with sun-drying should do so.

The amount of slime that can be successfully leached with sand in a dry-filling percolation plant is large but variable. Slime, as referring to that part of mill pulp which muddies water and does not readily settle and not to any particular degree of fineness, is of a variable nature, but seems to be divided into two classes: a siliceous slime, a slime produced by the fine crushing of a hard siliceous rock, and a clay slime, resulting from a talcose, clayey ore. A siliceous slime is of a more granular, leachable, and quickly-settled nature. Ore making such a slime, when it is crushed through a 30 or 40-mesh screen and properly settled in a tailing pond, should leach well in a percolation vat. But the handling of a clay slime which is of a more flocculent and slowly-settling nature, and usually occurs in large quantities, is a more difficult problem.

Percolation vats are filled from tailing deposits by shoveling and carring, or more usually by wheel scrapers. The material, with reference to its coarseness or fineness, must be distributed evenly over the area of the vat, or the percolating solution, following the lines of least resistance, will leach through the coarse material, shunting the fine or slime material and producing unsatisfactory results. If the material must be varied in the same vat, it should be by a layer of even thickness over the whole area of the vat. No lumps or clots of slime should go into the vat, for they will absorb the solution and, from their unleachable character, will not allow it to be displaced; thereby causing a loss of cyanide and leaching capacity and giving practically no extraction.

Treatment of Dry-crushed Ore. — Ore that has been dry-crushed is delivered to the vats by cars or belt-conveyors, and is invariably homogenous, so that a uniform leaching rate all over the vat is to be had. Also, the sand and slime portions of the ore are so well mixed that together they form an easily-percolated material, allowing the cyanide solution to penetrate the ore and dissolve the metals, and then to be displaced and well washed out together with the dissolved metals. Ore treated by dry-crushing and percolation is seldom crushed finer than 40-mesh and is being less and less resorted to, for the ease, economy, and satisfaction of fine wet-crushing in comparison with fine dry-crushing, and the ability of the filtering devices to handle finely-crushed material, giving a high dissolution of the precious metals, is making dry-crushing uneconomical. The slimy material made in dry-crushing seldom gives any trouble in leaching, except in some cases where the clay slime, though evenly and thoroughly mixed with the sand, tends to absorb the solution and agglomerate, so that a thorough washing cannot be made. Such ores do not often occur, but when found should be filtered by the vacuum or pressure leaf-filter, for they settle too slowly and retain too much moisture to be treated by decantation, and cannot be washed to an advantage in the ordinary way in a plate and frame filter-press.

Direct-Filling of Vats with Wet Pulp.—In the wet-filling of vats, the pulp flowing from the wet-crushing mill is delivered to the vat by a revolving distributor; in some cases by a movable hose placed in different parts of the vat, a plan not to be recommended. This distributor, known as the Butters and Mein or garden-sprinkler type, Fig. 1, consists of a revolving basket or hopper placed over the exact center of the vat. Pipes of uneven lengths ending in pipe elbows radiate from the basket. The mill pulp is led into the basket or hopper by a launder and runs out through the pipes and elbows into the vat. The hydrostatic head of the pulp in the basket, in connection with the right angles of the elbows from which the pulp flows, causes the distributor to revolve as a garden sprinkler does, delivering the pulp to the vat in a series of various-sized rings about the center. As first used, the pulp without classification was run through the distributor into the vat, the surplus water and part of the slime overflowing

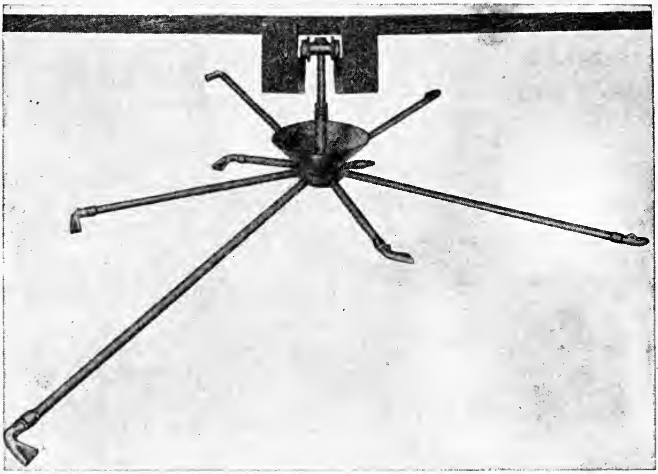


Fig. 1.—The Butters and Mein Automatic Sand Distributor (Pacific Tank and Pipe Co.).

the rim or edge of the vat into an annular launder; the overflow edge of the vat consisting of a tongue or strip of wood that could be moved up or down or planed off that the slime water might overflow evenly at all points about the circumference of the vat. This method resulted in the deposition of a large quantity of slime with the sand. The slime deposition was uneven from the center to the circumference of the vat, owing to the flow of water from the center towards the rim. These tended to cause uneven filling and poor leaching. As at present used, the mill pulp is passed through classifying cones, spitzkastens, or mechanical classifiers, which remove as much of the slime as possible. The sand flows through the distributor into the vat, from which the overflow carrying some slime and fine sand is over the rim into an annular launder or through small decantation pipes at the rim of the vat, or through a standpipe in the center of the vat and underneath the distributor. There has been used in vats collecting the sand for removal to the regular treatment vats a slat gate or window in the side of the vat extending from the top to the bottom, which is closed by a canvas curtain gradually unrolled and held in place by the sand as it rises in the vat.

It can be understood that there are objections to all of the above methods. To obtain good and satisfactory percolation, two things are necessary: first, a sand that is absolutely free from unleachable slime, a sand that is clean and sharp no matter how fine it may be; and second, an even distribution of that sand in the vat. Cone classifiers have in some cases made a very clean separation of sand and slime, but generally are not highly efficient. However, the Dorr classifier, as shown in Fig. 2, of the drag type of classifiers, gives an almost absolutely clean sand and a true slime, thereby fulfilling the first requirement. The usual methods of running sand distributors do not give the best distribution of sand in the vat. A large volume of water or very dilute pulp is used to operate the distributor; this creates a current from the center of the tank to the rim overflow, causing the fine sand to travel away from the point where it strikes the surface of the water filling the vat toward the overflow, and some even being carried out of the vat. Variations in the charge are also caused by the stoppage of the distributor at times. By arranging the distributor to be driven by power and running the sand delivered by the Dorr classifier down a steep grade, with as little water as possible, into the distributor and vat, an almost perfect leaching charge can be obtained. The Dorr classifier makes a clean separation of sand, while the small amount of water used to convey the sand through the distributor into the vat does not create a current, thus allowing even the fine sand to settle where it strikes the water in the vat, while the small overflow is almost clear water. This method of deposition allows the filling of a vat to be begun without first filling with water, to assist in the classification as must be done when using cone classifiers, and does not introduce difficulties by depositing a layer of slime when the mill is temporarily shut down.

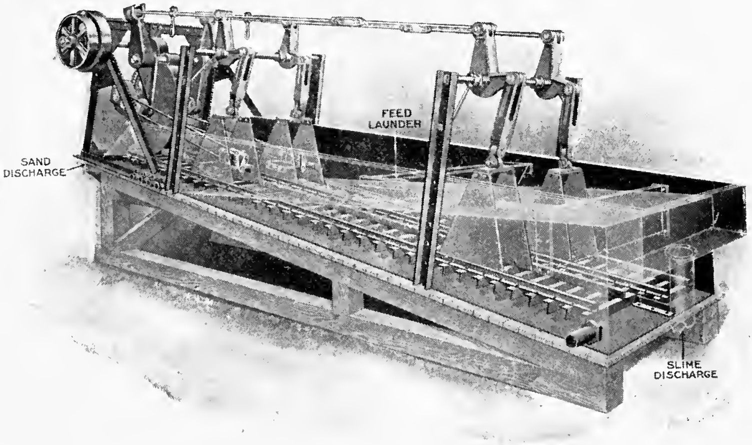


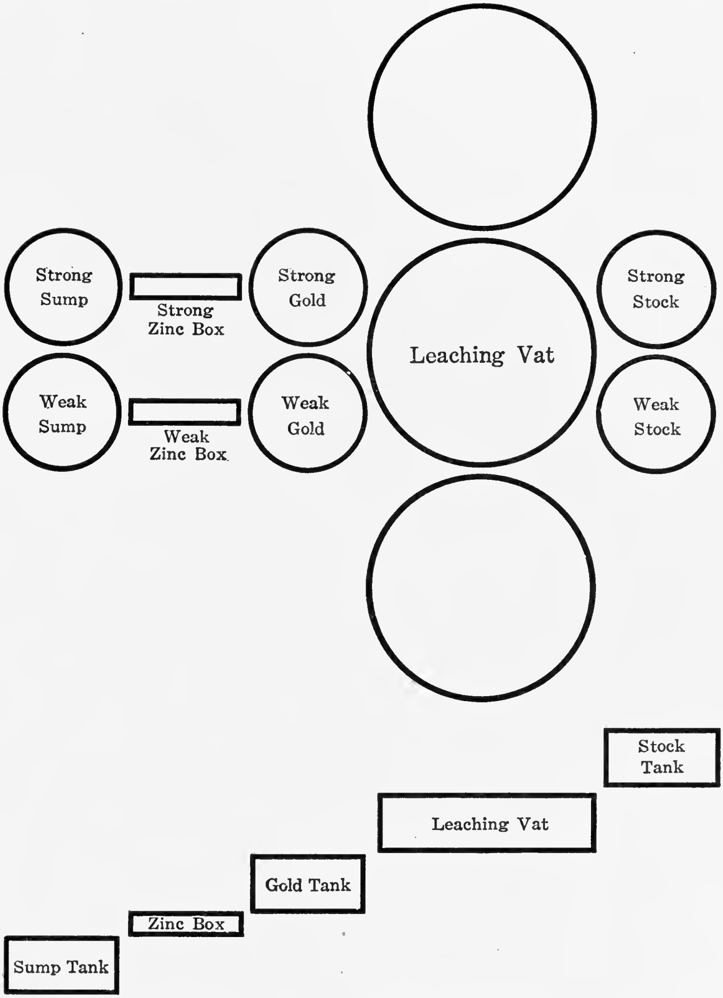
Fig. 2.—The Dorr Classifier.

The present practice is to prevent as much as possible any slime from getting into the sand charge, but in some plants the sand is rather crudely settled in collecting vats and transferred thence to final-treatment vats by hand or mechanical digging and conveying appliances. This transfer effects a thorough mixing of the coarse and fine sand and the slime; it also makes the charge more readily leached by increasing the voids or interstices between the grains of pulp. When sand is settled under water, the grains arrange themselves very compactly, thereby hindering percolation and aeration, but when a drained charge has been transferred to another vat it will occupy probably from 10 to 20 per cent more space (after it has subsided on the introduction of the first solution). Similarly, the ores from tailing deposits or dry-crushing mills when dry-filled into the vats will probably occupy from 10 to 20 per cent more space than when wet-filled — the increase is variable. The transfer from the collecting to the final-treatment vat also aerates the charge, though this can be well effected in other ways. By the use of a good method of wet-filling, all necessity of transferring the charge may be done away with.

Depth of Sand Charge. — It is hard to say what is the maximum depth of charge that can be leached. As the depth of the charge becomes greater, the pressure in the lower part of the charge from the ore above becomes heavier, causing the voids or interstices in the ore-charge to become smaller and thereby hindering percolation. This must especially be taken into account in treating ores containing slime, as from tailing deposits or dry-crushing mills, also finely-ground sand, for as the ore is ground finer or the amount of slime is increased, the voids or interstices become smaller. The higher the percentage of moisture retained by the drained charge, the lower will be the leaching rate and the less easily will the charge be leached. It is likewise hard to say what is the maximum fineness of ore that can be leached. No unsatisfactory results have been reported from leaching fine sand; it has been found that clean, sharp sand ground to 200-mesh can be leached. The question of leaching ore depends not so much on the fineness of the material, as on the amount of flocculent slime, the total absence of which gives ideal conditions for leaching. The leaching rate has been defined and the bearing on it of the volume of solution to be passed through the charge, which has a most important influence on the depth of charge permissible, has been discussed under Ore Testing and Physical Determinations. Six feet was the standard depth of leaching vats for a long time, but sand crushed very fine is now being leached in charges 10 feet deep, and vats 14 feet in depth are in use.

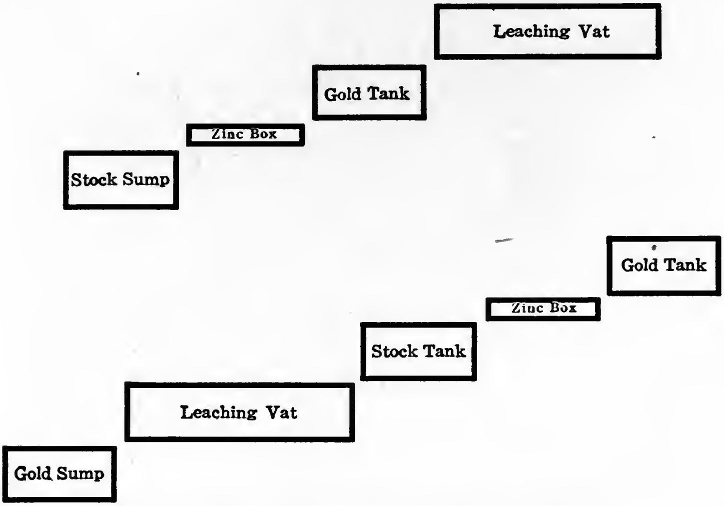
Arrangement of Leaching Plant. — Fig. 3 represents the usual plan of a leaching plant, while Fig. 4 shows its elevation. Figs. 5 and 6 show elevations of plant arrangement that are permissible, but not as convenient as in Figs. 3 and 4. However, the arrangement in Figs. 5 and 6 is often used where it is required by the surface configuration or the arrangement of the mill. The solution tanks supplying the leaching vats are always called

" stock tanks " or " storage tanks," preferably the former. The tanks receiving the enriched solution as it drains from the ore and which act as reservoirs for the zinc boxes are called " gold tanks." The tanks following the zinc boxes are called " sumps " or "sump tanks," when in the lowest part of the plant.



Figs. 3 and 4. — Arrangement of Leaching Plant.

Weak and Strong Solution and Their Separation. — It was formerly the custom in all plants to separate the solution into two parts, a "weak" and a "strong" solution, separate tanks and pipe systems being provided for each. The proportion of these solutions was equal or unequal, — usually equal, — as appeared best or convenient in the plant practice. Whatever proportions were used were kept constant by turning the solutions flowing from the treatment vats into the "strong gold tank" when they were high in cyanide strength, and into the "weak gold tank" when low in cyanide, keeping one as high in cyanide and the other as low as possible. The "weak solution" was in many cases used with a high protective alkalinity as the first solution on the ore, being a sort of an alkaline wash to neutralize the acidity of the ore, with a less consumption of cyanide than if strong solution was introduced before the cyanicides had been largely neutralized. And in all cases the weak solution was used as the final solution washes, thereby lowering the amount of cyanide to be displaced by the final water washes, and effecting a considerable saving in the amount of cyanide mechanically lost, through being discharged as the moisture in the tailing residue. It can be understood that there must be some action by the cyanide upon the metals and other substances in the ore every time the solution is brought in contact with the ore, and that this action is proportional to the strength of the solution; con-



Figs. 5 and 6. — Arrangement of Leaching Plant.

sequently the use of a weak solution instead of a strong one, for washing out the value after it is dissolved, must effect a saving in the cyanide chemically used and lost. The additional solid cyanide is always added to the strong or dissolving solution. When it was customary to use solutions of .2 per cent (4 pounds), and .25 per cent (5 pounds) for the dissolving, a material saving in the cost of cyanide was made by the division into two strengths of solution, but since it has become common to use solutions having a maximum strength of .1 per cent (2 pounds) on gold ores, the division has in many cases been abandoned. Especially where the problem of differentiating into a weak and strong solution has been complicated by the slime-treatment system or the necessity of having all the solution carry a substantial cyanide strength to get a good precipitation. Still it may be said that the division is practiced wherever it is possible and economical to do so.

Application of Solution to Treatment Vats. — The first solution run to treatment vats filled with dry ore is invariably from the bottom, being introduced underneath the filter bottom to gradually rise up through the ore and appear on the surface of the charge. There are several good reasons for this. The solution is usually started running long before the vat is filled and leveled off, the bottom of the charge thereby gets a much longer time of treatment than the upper part, a desirable thing, as the bottom of the charge is packed tighter by the weight above it and is seldom well drained, in consequence receiving less aeration and having a slower dissolving rate. The charge subsides to a very large extent — as much as 10 per cent — as the solution rises, thereby allowing the tanks to be filled with more ore than if the first charge of solution was run on from the surface. This subsidence is a gradual, easy movement of the particles of ore adjusting themselves, whereas, was the solution run on from the top, there would be a fissuring, channeling, and uneven packing of the charge. To the casual observer these would be of no moment, but they do cause irregular percolation and extra labor. It has been said that advantage should be taken of the air in the voids or interstices of the ore by running the solution on top of the charge, letting the air bubble up through. The theory of aeration in cyanide practice is sufficiently settled to say that if the solution had been allowed the usual opportunity to aerate, it would now absorb no more oxygen, and that the free air bubbling up could not assist in dissolving gold that was not yet in a condition akin to a nascent state and ready to go into solution through having been acted upon by the cyanide. The solution must, of course, be allowed to rise until it covers the whole surface of the ore charge, and with dry-filled ore until it is ready to overflow the vat, for the first solution passing through the charge and appearing on the surface must necessarily have much of its cyanide strength destroyed. The ore charge is carefully leveled off during the filling process if dry-filled, or after the water has disappeared from the surface of the charge for some time in the process of draining if wet-filled. The leveling being done by shoveling, hoeing, or raking the charge. High spots in a vat of dry-filled ore may be leveled at the time the solution appears on the surface, or at the culmination of the wet-filling of a vat, by pressing and working the high spots of ore with the flat surface of a hoe pressed vertically downwards, without harming the charge for good leaching purposes. The more perfect the method of filling, including adjusting the discharge from the pipes of the automatic distributor, the less leveling will be required.

Where the ore is wet-filled into the treatment vats, the first solution is run on from the top, for the reason that it is more convenient to displace the moisture in the ore in this way and draw it off from the bottom of the vat. The ore charge being perfectly settled and the grains of ore arranged closely and firmly together in the process of filling and settling under water, there is no subsiding, settling, channeling, or fissuring of the charge as would occur with dry-filled ore. Ore transferred wet into treatment vats from collecting vats usually receives its first solution on top of the charge, though it can be more advantageously introduced from the bottom if means are provided for drawing off the displaced moisture and weakened and diluted solution that appears above the charge.

Ore that has been filled into the vats dry must contain its soluble or free acidity that is removable by water-washing, and its insoluble latent acidity, the two forming the total acidity which an alkali would neutralize. The total acidity represents a large part of the cyanicides of the ore, to which may be added the other cyanicides which destroy cyanide and cannot be removed or neutralized by an alkali, such as copper and some of the base-metal compounds. When the ore is crushed in water and immediately conveyed to the leaching vats, nearly all the soluble or free acidity is washed out; but not the latent or insoluble acidity or the nonacid cyanicides, though the first is neutralized to quite an extent by any lime or alkali added. Where the crushed ore is allowed to stand in tailing ponds, new free or soluble acidity is developed. Where the ore is crushed in cyanide solution and delivered to sand vats, the free and latent acidity is largely neutralized and washed out by cyanide and lime or alkalinity in the solution.

A consideration of the above principles in connection with those given under Alkalinity and Lime gives a clue to the proper use of lime or other neutralizer, and as to the nature of the first solution to be used on the ore, whether high or low in cyanide or in protective alkalinity. If the dry-crushed ore contains an extraordinarily large amount of free acidity, it should first be water-washed to remove it. If the water wash or final water wash be alkaline, or lime slacked as a fine powder and well mixed has been added to the ore, the latent acidity will also be neutralized leaving only the nonacid cyanicides and new acidity which may be generated. Water washes to remove the acidity are now seldom used, except in treating concentrate or old pyritic tailing. Their use on dry-filled ore increases the amount of plant solution so that little or no water may be used for the final wash before discharging the tailing or residue, thereby causing a higher mechanical loss of cyanide. Consequently, under the above conditions, a solution low in cyanide and strongly alkaline is run on the ore. As soon as the solution appears at the surface of the charge, if it has been absolutely necessary to introduce it from the bottom, the introduction is changed from the bottom to the top of the charge, the drain valve is opened, and the weak solution is allowed to percolate through the charge, destroying the total acidity in the charge by its alkalinity and that of the lime added to the charge. The acid-destroying solution wash is allowed to run through the charge until the drainings show a slight protective alkalinity, when the solution is allowed to sink below the surface of the charge for some time, before being followed by the strong solution upon which the main dissolution of the gold and silver is relied. It can be understood that this method of using a first wash of weak solution is for the purpose of reducing the consumption of cyanide, and, as it involves some time and labor and may be otherwise undesirable, it is not often followed. In some cases with wet-filled ore this first wash or alkaline solution is treated as a separate and third nature of solution, to precede the strong solution as an alkaline wash and to follow the weak solution as a final water-wash, being precipitated only before use as a final wash. Laboratory investigations regarding the free and latent acidity and the nonacid cyanicides and the consumption of chemicals by each will indicate the procedure that may be advisable, but the results in actual practice by comparative tests and tests upon the solutions should determine the plant procedure. With silver ores it is generally advisable to start off with a strong solution that will dissolve and hold the silver in solution, and not so weak as to allow it to be reprecipitated, by the alkaline sulphides, about the remaining silver as a film of silver sulphide rather hard to redissolve.

Ore crushed in water and wet-filled into leaching vats may be taken as the type case of how leaching practice or treatment is conducted. Immediately upon the filling of one vat the pulp flow is turned to another. The drain cock of the filled vat is opened while all the water possible is syphoned from the top of the sand charge by a hose or decanting-pipes in the rim of the vat. As soon as the surface of the charge is sufficiently dry, it is leveled. When the charge is well drained of its moisture as indicated by the discharge from the drain valve, usually requiring from eight to eighteen hours, the first solution is run on top of the charge from the strong-stock or storage tank, presuming the strong solution is used first. This first solution is "standardized" or brought up to the standard of the maximum strength considered necessary to dissolve the precious metals. To do this the "solution man" in charge of the work for the shift under the plant superintendent, roughly mixes the contents of the strong-stock tank and drops a bottle attached to a string or pole into various parts of the tank to obtain an average sample of the solution. The solution is tested for its strength in free cyanide, and possibly for its total cyanide if that is being taken into account by the metallurgist in charge. The tonnage of solution is noted from a telltale and scale upon the side of the tank, or by plunging into the solution a measuring stick graduated to show the amount of solution in tons. The requisite amount of cyanide is now added to bring the solution up to the desired strength. It may be added by dissolving the solid salt in a small " dissolving tank " set above and emptying into the strong-stock tank, with tables and scales prepared that give the amount of concentrated solution required to bring the strong solution up to the standard. The cyanide may also be added to a perforated box or metal basket hanging in the flow into the strong-stock tank, or it may be placed in such a receptacle suspended in the solution and occasionally juggled up and down by the solution man. After the cyanide has been added and dissolved, the strong-stock solution is agitated to render the solution homogeneous in strength by a board attached to a pole as a hoe or by a very short air agitation.

In running solution on leaching vats, the whole surface of the charge should be covered as quickly as possible, for if more solution disappears at one part of the surface than another, that part is better treated by being washed freer of dissolved metals than the other part. Where a percolating charge is supplied by small solution pipes, this may cause serious trouble, but can to some extent be remedied pending the installation of larger pipes by closing the drain cock, saturating the charge, and flooding it with solution until the vat will hold no more, and allowing the solution upon draining to run into the vat until ready to disappear below the surface of the charge. The vat may be drained each time and then closed for a repetition of the above, or when the solution is ready to disappear from the surface of the charge the drain cock may be closed until the depth of the solution above the charge can be raised, or the outflow can be cut down to the inflow and a layer of solution kept at all times above the charge.

The solution running on a leaching charge to some extent presses out or displaces the solution or moisture below it and to some extent dilutes or is diluted by it. Dilution is especially the case with that moisture absorbed by the ore and adhering to and wetting the grains, such as is in evidence when the charge is drained. While that filling the interstices or voids between the grains when the charge is saturated, and which is less closely in contact with the grains, is mainly displaced by the solution above it while percolation is in progress.

The first solution having been run on top of a well-drained charge, in the course of a little while the dribbling flow from the bottom of the vat commences to increase as a result of the displaced moisture and diluted solution reaching the bottom of the vat as the advance guard of the solution running on top of the charge. This flow is at first of no value, being mainly displaced moisture or solution in which the cyanide has been destroyed by the cyanicides before effecting any dissolution of gold and silver, but gradually accumulates in value. This value may be in a low cyanide strength without precious metals in the case of an ore in which the cyanicides have been well neutralized but the metals are slow to dissolve, or it may be in gold and silver and no cyanide in the case of an ore still containing many cyanicides, but in which the metal dissolves quickly or where an unprecipitated solution is used for the first solution, or the flow may gradually "build up" in both cyanide and the precious metals.

The operator to control the operations of a plant always makes tests of this solution on several charges at the starting up of a new plant, and occasionally thereafter. In making such tests, as soon as the dribbling flow increases substantially, samples of the solution are taken hourly for a short period, then every two to six hours, until the vat is finally drained for discharging. These solutions are tested for their cyanide strength, acidity or protective alkalinity, and gold and silver. The cyanide strength and protective alkalinity of the solutions flowing on top of the charge are also determined. The results are tabulated for comparison and may easily and advantageously be plotted to show the varying cyanide strength and precious-metal content during the different periods. These results indicate when the outflow should be turned from waste to the " weak-gold tank " as containing gold and a small amount of cyanide; when to turn the flow from the " weak-gold tank " into the " strong-gold tank " in the effort to keep the strong solution as high in cyanide as possible at the expense of the weak solution and thereby save in the consumption of cyanide; when to turn back from the strong to the weak-gold tank; and finally the number of washes that it is profitable to give the charges. Correlatively with these solution samples are taken samples from the charge every few hours during the period in which it is supposed the gold and silver are dissolving. These samples are immediately washed to prevent the quick dissolution which takes place when partly-treated ore containing cyanide is exposed to the air, and which would lead to wrong conclusions. The samples are assayed to learn the progress of the dissolution of gold and silver, and the length of time it may be advisable to apply strong solution. The results may be tabulated and plotted in connection with those obtained from testing the outflowing solution. The results from these samples, through the amount of protective alkalinity shown and its variation at different periods, indicate if the proper amount of lime is being used. Also if it is being used in the right way to give a constant alkalinity, or should its alkaline influence be retarded or hastened.

The first outflowing solution from the drain cock should be allowed to flow until its cyanide strength is nearly that of the solution flowing on the charge. The drain cock may then be closed, and the solution allowed to remain in contact with the ore as long as it dissolves gold and silver at an active rate. This can only be judged from the assay of the washed samples of sand taken at different times. With a clean gold ore containing the gold in a fine state of division, the gold may be dissolved by one contact with the strong solution and then in a comparatively short time. With gold ores containing coarse gold or with much sulphide, and with silver ores, a point is reached where the dissolving action of the solution at rest with the ore commences to rapidly fall, owing to the reducing action of the ore and cyanicides having utilized all the oxygen and destroyed most of the cyanide, the dissolution of the coarse gold causing the same effect though it cannot be said to foul the solution. The solution should now be drained from the charge that fresh and active solution may be placed at work. It is preferable to draw the solution off completely, thereby drawing air into the voids or interstices between the grains which were formerly occupied by the solution. If the solution still in contact with the ore is not too enervated but is still strong in cyanide, a rapid dissolution of gold and silver takes place. When the solution is well drained, the drain cock may be closed to prevent escape of the air, and strong solution again flooded over the charge. As the solution sinks in the charge it presses out the air which rises in small bubbles throughout the charge, and undoubtedly has some helpful effect in aerating the solution and charge. After the charge is saturated with solution and the bubbling has ceased, the drain cock may be opened and the charge started leaching with solution constantly running on top of the charge to an amount equal to the outflow. Or the drain cock may be kept closed until the solution again ceases to actively dissolve the gold and silver, when it may be opened for draining the charge, etc.; but it should be remembered that the inflowing solution to some extent presses down and displaces the moisture that still remained with the ore and may collect it in the bottom of the vat, which would require leaching for a short period to insure fresh solution being brought in contact with the bottom of the charge.

Instead of draining the charge completely before adding the second solution, the second solution may be started running as soon as the drain cock is opened, thus causing continuous leaching without aeration. The advisability of continuous leaching or of alternate leaching and aeration varies with the chemical and physical nature of the ore and with the plant. With the metal in a native form, continuous leaching with a freshly-aerated solution will supply enough oxygen to give a high extraction. With a sulphide ore the aeration and oxidation by exposing the charge and drawing in air apparently cracks open the sulphide and better exposes the metal for dissolution, thereby giving a higher extraction, as well as oxidizing the ferrous salts into inert ferric oxide and the alkaline sulphides into thiocyanates or sulphates, reducing the consumption of cyanide and increasing the dissolving rate. Alternate periods of percolation and aeration or intermittent washing require more time than continuous leaching, but the volume of solution required to give a thorough washing is smaller on the same principle whereby washing by decantation is more effective with a precipitate that settles to a smaller percentage of moisture. The thorough drainings also give a better wash on a charge containing slime, or poorly filled, so that in continuous leaching the solution following the line of least resistance largely shunts by portions of the charge with better washings of other parts. Allowing the solution to remain at rest with the ore permits of better diffusion of the rich solution in the slime agglomerates and pores of the grains of ore with the solution filling the larger voids between the grains, and by thoroughly drawing off the solution must better draw out this sheltered solution for subsequent displacement or dilution.

Some information as to the necessity of aerating the charge can be obtained by examining the inflowing and outflowing solutions for their reducing power. Besides the natural drawing in of air by the disappearing and outflowing solution, aeration may also be obtained by using a vacuum pump connected with the leaching vat underneath its filter bottom. This, besides drawing the solution off more thoroughly, draws air through the charge, thus giving a very thorough aeration. The vacuum pump method was introduced at a time when sand charges contained much slime, for the purpose of increasing the leaching rate and giving better draining. However, it was found to pack the charge too tightly for good results and was discarded, except for final draining. The vacuum-pump method for assisting percolation has been used lately with reported good results, and it would appear that it might work well on a charge entirely free from slime. The best method of giving a thorough aeration to a charge without removing it from the vat, consists of pumping air underneath the filter bottom after the charge is drained — a reversal of the vacuum-pump method. The pressure must be low, not exceeding 5 pounds per square inch or it will channel the charge.

It is a problem for the operator of each plant as to whether the alternate or continuous method of leaching should be practiced. The length of time strong or standardized solution should be applied to the ore is indicated by the time it takes to put the gold and silver in solution, but not absolutely. The value in a gold ore may dissolve in 24 hours, but require two additional days for washing out. If 75 per cent of the dissolvable value will go into solution in 12 hours, it may be better to draw the strong solution off or stop its use at that time, and begin washing with a weak solution, relying upon the remaining value to go into solution sufficiently soon during the time of using and washing with weak solution to get the maximum extraction and yet effect a saving in the consumption of cyanide, for the use of a strong solution increases the costs over a weak solution equally effective or effectively used, by its increased activity on the cyanicides and the zinc, etc. Again, a silver ore may require six or eight days to dissolve the metal, while the final washes are completed within a day or two thereafter. In such a case it would not be advisable to stop using the strong solution before practically all the dissolvable metal was in solution. It may be said in passing that there is only one iron-clad rule regarding the amount, time, and strength of solution to be used, and that is that the value shall be dissolved and washed out as thoroughly as is economically practicable, and no two cyanide operators will use the same exact methods, though in principle there is no important difference.

The weak solution follows the strong or standardized solution to wash out the dissolved value and to effect part of the final dissolution. Having determined how many tons of solution will be standardized up to the maximum dissolving strength each day or per charge in the usual practice, this quantity of solution is diverted from that flowing from the vats when at its strongest, the balance of the flow going into the weak solution. With plants crushing in water, the amount of weak solution often increases to a point where it cannot be handled or utilized. In such cases some of the weakest of the solution is run to waste after passing through the zinc boxes. The weak solution is generally applied as a continuous percolation instead of intermittently. A solution is always applied for a long time to the surface of a charge before its characteristics appear in the outflow; following the first application of strong solution the outflow may be weak in cyanide and metal for a long period, gradually building up and continuing high in both long after the change to weak solution, which gives rise to the saying that the strong solution dissolves the value but the weak solution washes it out. This tardiness, besides being affected by the cyanicides and dissolving rapidity of the ore, is increased by the depth of the charge and its slow leaching rate due to slime or the fineness of the ore.

The final water washes are applied after the weak solution has washed nearly all the dissolved metal out of the ore. The amount of water used is just sufficient to keep the volume of solution in a plant constant. A plant treating ore containing slime filled into the vats dry will discharge a tailing residue containing 20 per cent or more of moisture. In such a plant a vat containing 100 tons of ore may be washed with 25 tons of water, and even more to make up for the loss by leakage and evaporation. In a plant treating wet-filled sand, the sand will probably drain to 15 per cent moisture before the cyanide solution is applied, but as the residue is discharged with the same amount of moisture, no wash water can be used, unless owing to the slow dissolving rate of the metal, the first outflowing solution is so barren in cyanide and metal that it may be run to waste. Sand that has been crushed in cyanide solution and wet-filled into the vats will be discharged with probably 15 per cent moisture, enabling the use on a 100-ton charge of ore of nearly 18 tons of wash water, plus the loss by evaporation and by the large leakage incidental to such a plant. In actual practice the slime plant usually requires more than its proportion of wash water, or dilutes the solution in circulation, and when in conjunction with a sand plant where crushing is done in water often requires weak solution to be run to waste, while where crushing is performed in solution, it cuts down the rightful share of wash water of the sand plant.

Where water cannot be used for the final wash, weak solution performs the final washing. When water is used, the wash by weak solution is theoretically to the point where the water will just complete the work. As it is not practical to determine this for every charge by assaying the outflowing solution, the final weak-solution washes usually take the dissolved metals out pretty thoroughly, while the water wash acts as the factor of safety and washes out the final cyanide. To wash a leaching charge efficiently depends mainly upon the amount of solution passed through it and not upon the length of time occupied in washing, consequently the operator determines the number of tons of each solution he wishes to be applied, which is generally reduced to a certain number of washes of so many tons each. Sand charges are generally washed until the final solution assays at least as low as 20 cents per ton or lower, which with well-drained residues showing 15 per cent moisture would indicate a maximum loss of 3½ cents per ton of dry sand, while the actual loss would undoubtedly be much lower. If the final washing had been with a weak KCN solution of 1-pound strength, it would show a loss of .18 pound KCN or more per ton of dry ore. If the final washing was with water and the outflow titrated 1 pound KCN, it would indicate a maximum mechanical loss of much less than .18 pound per ton, for undoubtedly the strong solution is nearly all found at the bottom of the charge, and the same principle holds true with reference to the dissolved gold and silver still remaining.

The tank is sampled for assay after draining. For test purposes a sample of the well-mixed sand may be at once washed by decantation or otherwise to be assayed as a "washed sample." The difference between the washed and the unwashed sample shows the amount of dissolved value lost through poor washing of the charge. The amount of KCN and dissolved gold and silver mechanically lost by being discharged in the residue may also be obtained by taking 300 grams of the residue, adding water sufficient to make a one to one solution, — estimating the moisture or determining it in another part of the sample, — agitating for several minutes and drawing off and testing any aliquot part. Thus if 300 grams of residue containing 20 per cent of moisture are taken, it is equivalent to 240 grams of dry sand and 60 c.c. of water, to which 180 c.c. of water should be added to give a one to one solution. If 120 c.c. of the solution is used for assay, it will represent that metal held in 120 grams dry pulp (practically 4 assay tons), while the titration of 10 c.c. will give the pounds of KCN mechanically lost per ton of dry pulp. Samples should be taken on test charges representing different depths of the sand charge, for it is sometimes found that the metal is not as thoroughly dissolved in the lower part of the charge, due to the lack of aeration and possibly the weakened condition of the strong solution reaching that part of the charge. The bottom of the charge contains more moisture due to its packed condition preventing free draining, and to the tendency of the moisture just above the filter bottom to be retained in the sand by a species of surface tension, while this lower moisture must be much richer than that above.

CHAPTER IX

SLIME TREATMENT AND AGITATION

Definition of Slime. — Solids may be said to exist in two forms, crystalline and amorphous. Substances in a crystalline form have a definite and regular shape; they are compact and substantially solid. Grains, crystals, and solid bodies represent the crystalline structure. The amorphous is the opposite of the crystalline structure; it is irregular and indeterminate in shape, and less compact and substantial than the crystalline. In the cyanide process the crystalline is represented by "sand" and the amorphous by "slime." Sand may be said to be that part of the ore, however fine it may be, which is crystalline, granular, sharp, clean, compact, and under the microscope presents regular structure, sharp edges, and solid faces; which readily settles in still water and does not muddy water, and which can be leached. While slime may be said to be noncrystalline, light, feathery, flaky, noncompact, impalpable material, showing irregular shape and structure; which muddies water and does not readily settle in still water, but remains in suspension disseminated throughout the water, to gradually agglomerate and settle as a flocculent slime to form a plastic clay or mud, very unleachable and impermeable by water. Slime is sometimes spoken of as a "colloid," a term applied to substances suspended in solution in a semisolid state.

While sand partakes of the nature of quartz, slime partakes of the nature of clay. Slime is usually a silicate of aluminum, iron, or alkaline earths. The hydrated aluminum silicate, kaolin ($Al_2O_3.2\,SiO_2.2\,H_2O$), is a most prominent slime or constituent of the slimes, and shows that the term "colloid hydrate" is not a misnomer as a technical term for slime. Slime is found least in hard, crystalline quartz, and most in talcose, clayey, feldspathic, and oxidized ores, or those containing kaolin, alunite, and limonite. The percentage increases with finer crushing as the amount of impalpable powder produced must necessarily increase. The slime produced in crushing a hard quartz probably has certain crystalline qualities to account for its quicker settling and more permeable nature than that arising from a clayey ore, for a slime resulting from a quartzose ore can be settled or filtered much easier than that from a clayey ore. Flocculent slime undoubtedly carries considerable fine sand covered and held in suspension by a coating of slime or colloidal material.

Slime is sometimes defined without reference to the crystalline or amorphous qualities, as that part of the ore which will pass a 200-mesh screen. This interpretation has been given the term because it is generally considered by many that material finer than 200-mesh had best be treated in the slime plant, even though it contains grains that are leachable, also because material ground to 200-mesh is excellent to treat in a slime plant. The use of the term slime in this manner has rendered it necessary in referring to an amorphous slime as discussed in the preceding paragraphs to use the terms " flocculent slime," " clay slime," " colloid slime," or " true slime." It would be well to distinguish a granular slime passing a 200-mesh by some modifying term, as a " sandy slime," reserving the word " slime " for the amorphous and real slime, but using the term " true slime " or other until the present confusion in the use of the word " slime " has passed away. It is highly necessary to distinguish between a sandy slime and a true slime as some processes, agitating machinery, and filters may be wholly or better adapted for treating the one class of material than the other. This is owing to the plasticity and the impermeability by water of a true slime, which as granular material is added and it becomes sandier, becomes less plastic and more permeable and leachable, and in part settles faster. Also because a sandy slime settles to a denser, more compact sludge that is harder to move and disintegrate.

Dehydrating or removing the moisture from a slime by heating or roasting renders it more susceptible to leaching and less adsorptive of moisture which cannot be displaced, a fact that has weight when the roasting of dry-crushed ore is being considered.

Slime Settlement. — The addition of any one of various acids, alkalis, or neutral salts to water containing slime in suspension causes the suspended matter to coagulate and settle much faster and to a smaller bulk than naturally. Lime is the only substance added for settling purposes in the cyanide process. Some idea of the amount of lime required can be obtained by laboratory tests on average samples of the slime pulp placed in graduates, to which known quantities of lime are added, and the subsidence compared at different periods of time. However, the amount used in actual practice is determined by that which gives the quickest settling into the least bulk with an economical amount of lime. Too much lime may retard the settling. The amount of lime used is variable, in some plants a few pounds per ton of dry slime will suffice; in others as much as 10 or 20 pounds of lime has been used per ton of ore.

The settlement of slime by lime or other alkali, an acid, or a neutral salt is on the theory that particles of any kind when suspended in a liquid are electrostatically charged. That these charges while they may be positive or negative for different kinds of suspended matter, are still of the same sign for all particles of the same substance, and consequently repel each other, for two different substances in contact have equal and opposite electrostatic charges at their contact surfaces. The tiny slime particles by the repulsion of their like electrostatic charges together with their relatively large surface in proportion to their weight and their solubility or saturation similar to that of a sponge suspended in water, counterbalance the action of gravity and remain suspended in the water. In short, their density differs so little from that of the surrounding liquid that they remain in suspension or settle at an infinitesimally slow rate. Heating the water lightens its density and lessens its viscosity and confers greater mobility so that the particles may better settle through their higher specific gravity, but on a working scale the cost of heating overbalances the advantage of the quicker settling. The introduction of an acid, alkali, or salt capable of disassociating produces both positively and negatively-charged ions which attract the slime particles having different charges, causing a coagulation of the slime to expose less surface for a given mass and consequently to better settle. Substances used for connecting other substances in this way by their electrostatic charges are termed electrolytes; thus lime is an electrolyte in the settling of slime by its aid.

The settling rate decreases with the density or viscosity of the pulp. With a very dilute pulp the slime at first settles rapidly to leave a clear solution, but the settling rate, the downward movement of the line of demarcation between the slime pulp and the clear solution, gradually grows less as the underlying slime pulp becomes thicker and denser until the settling rate practically becomes *nil.* In this movement the true slime appears to move downward by layers, that slime at the top of the charge when settling was started becoming the top of the settled slime. While the settling rate is thus decreased through the settling of the slime being retarded by the density or increasing density of the medium through which it is settling, in a practical way the depth of the settling column has a most important influence. It is apparent that the slime particles in a charge 4 feet deep in a small-diameter tank will have to settle through practically four times the distance as in the same charge when 1 foot deep in a large-diameter tank of four times the area, and that there is a greater retardation in the 4-foot charge owing to the greater weight of solution or solution and pulp overlying any section of the depth and thus making the density greater. This will explain why shallow tanks of large diameter are necessary in decanting, instead of deep tanks of small diameter.

Classification or Separation of Sand and Slime. — Slime treatment may refer to the treatment of a true slime, of a sandy slime containing the finer sand made in crushing together with the true slime, or of all the ore ground fine — a case of "all-sliming." The methods used in treating these three classes of material do not vary in the main, though the presence or absence of sand is an important detail. Where the decantation system of treatment is used, it is aimed to treat only the true slime, on account of the greater ease with which the sand can be treated in a leaching plant and the higher efficiency of the leaching plant in washing out the dissolved value. With a modern filter plant it is, with most filters, desirable to treat all or a part of the sand with the slime, since they can handle a sandy slime better than a true slime.

For producing a true slime and furnishing a clean sand at the same time, the Dorr classifier is the only machine approaching perfection. Where it is necessary to throw the finer sand into the slime, cone classifiers may be used, or the Dorr classifier modified so as to produce a sandy slime. Cone classifiers never give an absolutely slime-free sand for the leaching plant, consequently their underflow of coarse sand should be reclassified in a Dorr machine.

Pulp Thickening. — The slime pulp runs from the classifiers to pulp thickeners. These are of two kinds: those operating on the settling principle with cone bottoms discharging a thickened pulp, and those using the same settling principle, but discharging the settled slime by some mechanical means. Lime is usually added to the slime flow before entering the thickeners for neutralizing the acidity and to effect quicker settling. The pulp flow is conducted to the center of the tank or cone and introduced into it through a central pipe emptying a few feet below the surface of the water. In this way the slime flow does not disturb the water and settling slime, but emits from the bottom of the pipe and rises and moves at a very slow rate of speed, under conditions favorable for the deposition of the slime material, to overflow the side or rim of the cone or tank as a clear or partly clarified solution or water ready to be reused or run to waste. From the bottom of the tank, the thickened sludge may be drawn off continuously, or intermittently with more or less trouble, with a dilution that is variable but seldom less than one part of solution or water to one part of dry slime.

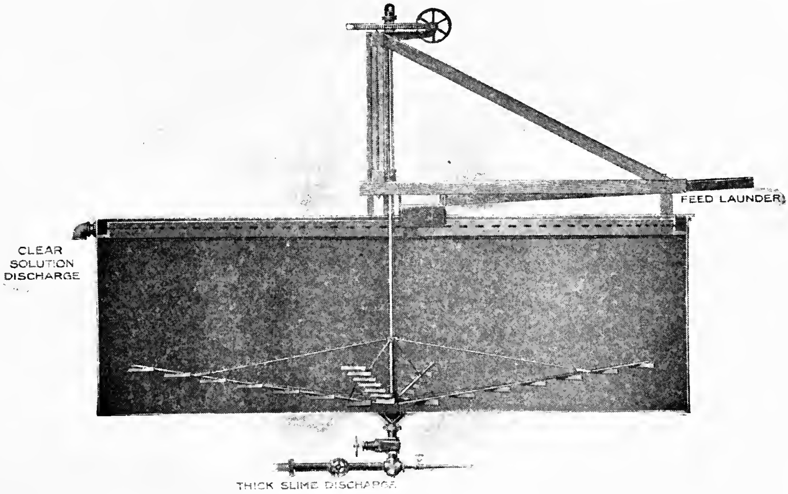


Fig. 7. — The Dorr Slime Thickener.

Charging for Agitation. — The pulp is drawn continuously or intermittently into the agitation tank, or the slime flow may be settled in an agitation or collecting tank in a way similar to that in the pulp thickeners. After the agitator has received its charge, lime may be added as a milk-of-lime, also sufficient cyanide, by being dissolved in a small stream of solution, to bring the lime and cyanide strength up to the desired amount. Lead acetate dissolved in water may also be added for the purpose of precipitating any alkaline sulphides which may form. Any extra solution to bring the charge to the desired proportion of solution and dry pulp is added. If the crushing has been done in solution, the pulp thickeners may only be required to reduce the pulp to the consistence desired for agitation, so that no additional solution need be added. But if crushing is done in water, the pulp thickeners are worked to their highest efficiency or the pulp in the collecting or combined collecting and agitating tank is allowed to settle, and as much water decanted off as is possible before adding cyanide solution and beginning agitation. With a plant crush-

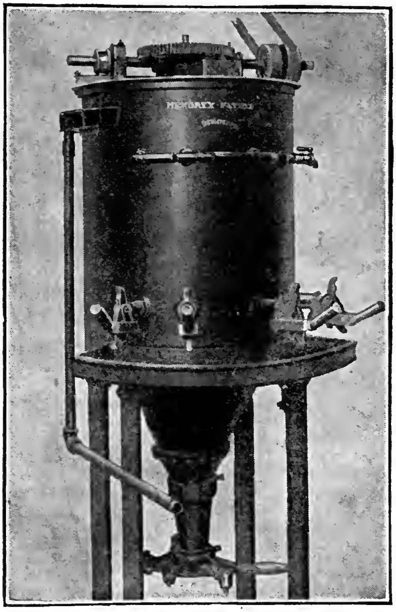


Fig. 8. — The Hendryx Slime Thickener or Tailing Dewaterer.

ing in water, it is necessary to dewater the pulp to the lowest possible percentage of moisture, or the amount of solution in the plant will soon increase to such a quantity that some of it must be run to waste.

Amount of Solution in Agitation. — Slime is agitated with varying amounts of solution. Where decantation is practiced, 1 part of dry slime will be agitated with from 3½ to 6 parts of solution, that a large volume of solution may be decanted off to enable a low tailing to be obtained. Where the slime is filtered without decantation it will be kept much thicker, perhaps as much as 1 part of dry pulp to 1.2 parts of solution (by weight). The density of the pulp or the proportion of solution to dry pulp has an important influence on the dissolution of gold and silver, even though the strength of the solution be the same. The dissolution of the gold and silver will be slower with a thicker pulp. A pulp of 3 parts of solution to 1 of ore against one containing only 1½ parts of solution brings double the amount of cyanide and dissolved oxygen into play, consequently with a thin pulp the strength of solution can be kept lower, while with the same strength the dissolving rate will be faster. In both agitation and percolation, so far as concerns the dissolving of the metal, the effect of a strong solution small in quantity and applied for a short time can be equaled by a weaker solution larger in quantity and applied for a greater length of time. A thin pulp is often agitated and then settled and decanted from to a thicker consistence before being filtered, in the effort to meet the above conditions and to reduce the value per ton of the solution remaining in the pulp to be filtered. With air agitation there is no question but that a good aeration is obtained, but with mechanical agitators the necessity of aeration and the action of reducers in the pulp should be examined. It has been noted in many cases that after a certain length of agitation, no more value would go into solution, but by removing the old solution and applying new, either by decantation or the short contact with wash solution during the filtering process, the remaining dissolvable value goes into solution quickly. The same results on the aeration of a charge and adding newly aerated and precipitated solution has been noticed in the leaching process.

Strength of Solution and Time Required in Agitation. — The strength of solution that it is advisable to use will vary with the nature of the ore and the volume of solution used. On gold ore a strength of .05 per cent (1 pound) to .1 per cent (2 pounds) is generally sufficient, on silver ores up to .4 per cent (8 pounds), and on gold concentrate up to .5 (10 pounds). In some few cases a .025 per cent (½ pound) solution is sufficient on gold ores when using a large volume of solution and a long contact. Some gold ores contain the metal in such a fine state that when crushing in solution and all sliming — crushing all the ore to a sandy slime — nearly all the gold will be in solution by the time the tube mill is passed. But in most cases an agitation of 3 to 18 hours is required with gold ores, up to a few days with silver ores, and up to 10 days with concentrate. The progress of the dissolution of the metals and the consumption of cyanide and lime should be frequently tested by taking samples of the charge, testing and assaying the filtered solution, and assaying the washed pulp. The results may be tabulated and plotted and filed for comparison with others. The solution during agitation may be tested for its reducing power and the alkaline sulphides formed.

Intermittent and Continuous Agitation. — It is not necessary to completely dissolve the gold and silver at the first agitation where decantation is practiced, especially if the dissolving rate becomes slow during the latter part of the period of agitation. The agitations following, for the purpose of mixing the solution with the pulp, can be relied upon to effect final dissolution, together with the long-continued contact between the ore and the solution that takes place in the decantation process. Where the pulp is to be filtered, the value should be dissolved in one agitation unless, on account of the action of reducers and the fouling of the solution toward further dissolving of the precious metals, it is necessary to use fresh solution; though this can probably be met by aeration, at least the cause should be investigated and studied.

Two methods of agitation are in use. The charge, intermittent, or single-agitation system, treating each charge separately and individually, which was formerly used entirely; and the continuous system more recently developed. In the continuous system the charge is delivered continuously to the first of a series or battery of agitation tanks, through which the pulp passes to be delivered from the last agitator in the series to a filter or a stock tank supplying a filter. The method is illustrated in Fig. 9.

The pulp flows from pulp thickeners which settle the slime to the proper dilution or consistence, through the launder *A* to the first of the tanks — which are the Pachuca or Brown air-agitator type. Here it is drawn to the bottom of the tank to rise to the top through the central column and again descend to the bottom, in the process of being circulated up through the central column and down outside of it. The iron pipe *B*, whose inlet end is at least a few feet below the discharge of the central column of the agitator and about midway between the column and the outside of the tank, and which is set on an angle of 60 degrees, discharges into the next tank at a point midway between the column and the outside of the tank. In this way the pulp introduced at *A* is agitated in each tank and flows through the series to be discharged at *C*. The advantage of this system is that the large amount of labor, worry, wear and tear, and loss of time involved in charging and discharging a tank is entirely avoided. With well regulated tanks and equipment, attention need only be given to keeping the machinery in order and a watch over operations. The gain in mill height or the elimination of the costly item of lifting the pulp to a higher level is an important item in this system. In the charge system with the regulation Pachuca tank, the pulp will be discharged at *D* which is 45 feet below the top of the tank. With the continuous system the discharge is at

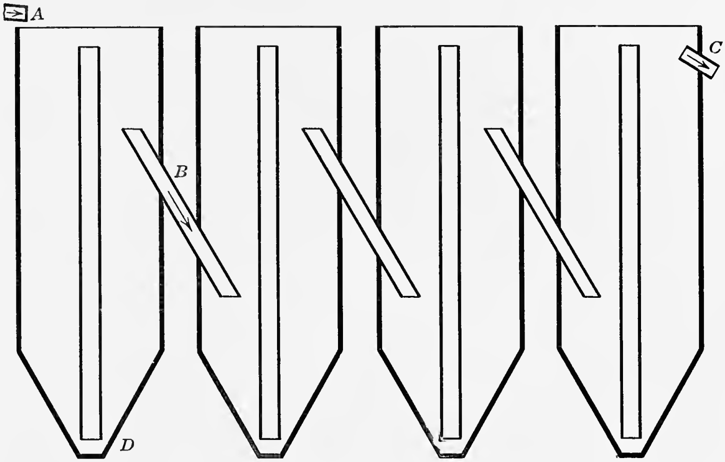


Fig. 9. — Continuous Agitation System.

C, which is only a few feet below the top of the tank. A longer agitation is given the pulp, for with the charge system from $\frac{1}{6}$ to $\frac{1}{3}$ of the time is occupied in filling and discharging the tank, whereas in the continuous system the tank is agitating all the time except when it is desired to work on or repair a tank, which can be done by cutting it out of the series through pipes and valves connecting the tank to tank discharges, the pulp in the tank to be worked upon being drawn off from the bottom of the tank. If the filter or filter-stock tank is placed just underneath the discharge from the last tank, an emergency pump should be provided to lift the pulp from the bottom discharges of the tanks.

The success of the continuous system primarily depends on the pulp being homogeneous in all the tanks; which has been proven to be the case with Pachuca tanks through sizing tests made of samples taken from the different tanks; though in some cases with a pulp containing coarse sand it may require considerable experimenting in the arrangement of the connecting pipes to get the pulp homogeneous in all the tanks. And secondly, that no particle of pulp shall move faster than others through the series and be discharged with less than the proper amount of agitation. That there must be a tendency to do this can easily be seen, but a particle of pulp that is undertreated in one tank, by passing out before receiving its share of agitation, on entering the next tank is placed on an equal footing with all particles entering the tank at that time. Consequently the probability of evil results due to a particle of pulp being shunted across the series of tanks and discharged without the proper amount of agitation becomes less as the number of tanks is increased. By using the same number of agitating tanks in the continuous system as would be required in the charge system, the use of a number of tanks together with the extra agitation due to no loss of time through filling and discharging — which acts as a factor of safety — the same extraction ought to be secured as with the charge system, while actual working results have shown a higher dissolution due to the more prolonged agitation. Even if a lower dissolution was obtained, the saving in labor and other costs by the continuous system would in most cases overbalance the lower extraction.

It has been proposed to employ a single, continuous-treatment tank by using the Just silica-sponge brick bottom. These prepared bricks are used just as a filter cloth in a leaching vat, by being laid and held in a steel frame to act as a false or filtering bottom. The bricks are so porous that by introducing air at a low pressure underneath the false bottom, it will pass through the bricks to emerge in tiny streams that will keep the slime in agitation. Its adaptation to the continuous-agitation system consists in using a rectangular tank somewhat similar to a zinc box with a series of compartments having an upward and downward flow, all compartments being of the same length. The pulp as it flows through this large box would be agitated by the air passing through the brick bottom, while the number of compartments may be many to lessen the tendency for any of the particles of pulp to get less than their proper share of agitation. The successful use of the silica-sponge brick in this manner would allow an agitator to be built and worked at a reasonable cost, to be economical of space, and which could be used as a filter-stock tank or with a variable amount of pulp, for which the Pachuca air-agitating tank is not adapted, since it must have a certain amount of pulp or the air-lift principle will not cause the pulp to circulate and agitate.

The perfection of the continuous system has advanced the cyanide process to a point where the ore may flow in a continuous stream from the ore bin to the tailing dump without any intermittent operations. With the perfection of a continuous filter that with all classes of pulp will cheaply and thoroughly wash out the dissolved metal, even with rich pulp, and that will reduce the cyanide mechanically lost to a relatively small amount, together with a determination of the simplest and most economically efficient agitator for continuous agitation, the day of percolation will be passed and cyanide plants and processes will tend to become as standard in design as the stamp-mill process.

Types of Agitators. — The first style of agitator that was employed, and which has been generally used with the decantation process, is the mechanical or stir agitator. These consist of large, round, flat-bottomed tanks as much as 40 feet in diameter and 20 feet deep. They are equipped with stirring blades attached to a shaft actuated by a gear mounted over the tank or underneath, and passing up through the bottom. Agitation of charges containing only true slime by these agitators is not difficult, especially if the pulp is dilute, but as the amount of sand in the charge increases, the difficulty in agitating becomes greater. This is owing to the tendency of the sand to pack and to resist the movement of the blades, which results in increased power being required, great trouble in starting a settled charge, and severe wear and tear on the machinery. The stirring gear in some types may be started with the arms raised and in the upper and more dilute portion of the charge, to be gradually lowered as the charge responds to the agitation and loosens up. Tanks without facilities for raising or lowering the blades should have them set 2 feet above the bottom of the tank to enable them to be easily started, as the sand then settles below them and yet can be brought into agitation when using a speed of 500 to 800 feet per minute. Short chains and iron cables have sometimes been hung from the arms to assist in stirring the slime below. Baffle plates may be attached to the sides of the tanks to insure better mixing. Two sets of arms, one near the bottom of the tank and the other near the center are excellent. Air may be pumped through perforated pipes in the bottom of the tank or through pipes attached to the arms, to assist in the agitation and to aerate as well. Agitation is often further assisted by centrifugal pumps taking the pulp from the bottom of the tank and returning it to the top of the charge.

The mechanical or stir agitators have been comparatively costly in the horse power and repairs required, and the agitation has been far from perfect. Yet the facts that the tanks, being large in diameter and rather shallow, were well adapted for slime settling and for decantation, and that the arm stirs were excellent for repulping the settled slime with the fresh solution added, is the reason for their extensive use in the past. The thinness of the slime agitated in working with the decantation process — 3½ to 6 parts of solution to 1 of dry slime—has assisted them to do good work. Likewise the treating of all the sand in the leaching plant and all the true slime in the slime plant, so far as the classifying apparatus enabled this, as is always the case with plants treating the slime by decantation. The new plants being built are all equipped with some form of slime filter, consequently mechanical or stir agitating tanks are seldom installed now, except as stock tanks to hold and keep in agitation the pulp to be supplied to the filters. They are well adapted for this, since they do not have to be kept full or nearly full, as is the case with air and some of the other agitators.

Agitation by a centrifugal pump drawing from the bottom of the tank, usually a conical-bottom tank, and pumping to the top of the tank, has been generally discarded owing to the great wear on the pump and pipe by the attrition of the sand. With a true slime the wear is much less than if the charge contains sand. Centrifugal pumps for this purpose are equipped with liners that may be removed when worn, but no liner or stuffing-box arrangement has yet been made that will last for any continued length of time. Aeration with this form of agitation is provided by flowing the pulp over an apron on introducing it to the top of the tank and by means of an air valve between the tank and the pump allowing a small quantity of air to be drawn in and pumped with the pulp. A patented method of using the

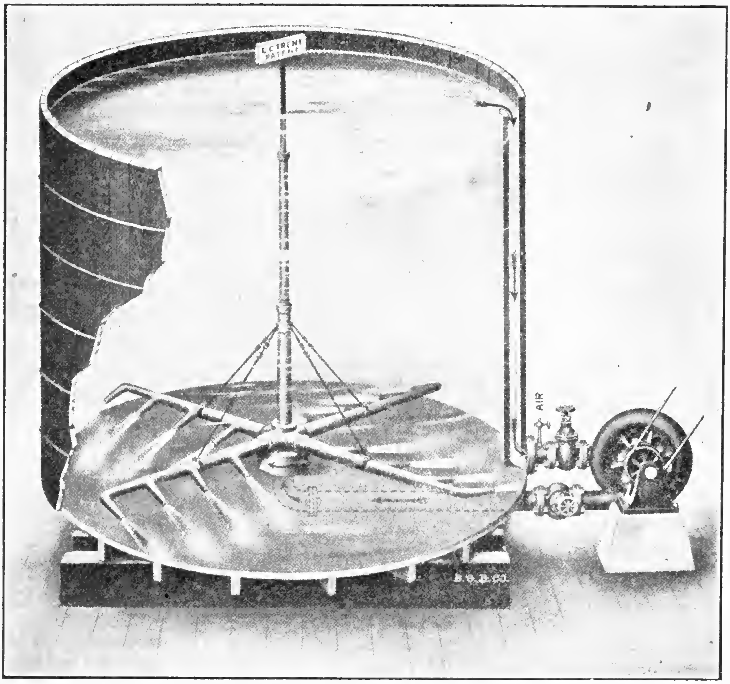


Fig. 10. — The Trent Agitator.

centrifugal pumps for agitating consists in using a deep cone-bottomed tank which has a hollow, central column. The centrifugal pump discharges slimy solution upward into the foot of the column, lifting the pulp up through the column by its force, as the hydraulic elevator does, and in that way starting and keeping up a circulation similar to that with the Pachuca tank. A shield or cap is suspended from the top of the tank around the air lift to form a calm zone of about 6 inches between the shield and the outside of the tank. The slimy solution passing through the centrifugal pump is drawn from the top of this calm zone, and is very dilute and free from sand, consequently it does not cause excessive wear on the pump and piping.

An agitator combining the principle of the mechanical stirrer and the use of centrifugal pumps is the Trent agitator, as shown in Fig. 10. With this agitator the thinner slime taken from the top of the charge is pumped by a centrifugal pump through the bottom of the tank into a revolving four-armed stirrer, from which the slime is emitted by a number of discharges set along the length of each arm, the discharges being at right angles to the arms. The force of the discharge causes the arms to revolve in the same manner as a Butters and Mein sand-pulp distributor. The same principle can be employed by using air under pressure instead of solution to cause the arms to revolve.

The Hendryx agitator, as shown in Fig. 11, is one of the most successful in use. It consists of a cone-bottom tank of ordinary height in which is mounted a central column or tube. In the tube is a shaft driven by a gear overhead the tank. Three propeller blades are distributed along the shaft. The rapid revolution of the shaft causes the blades to lift or force the pulp up the tube to flow over an apron, similar to an umbrella, to the edge of the tank, from which it sinks to the bottom of the cone to be drawn into the central tube again. The tank cannot agitate unless fairly well filled with pulp. The agitation is excellent, the wear little, and the trouble small or practically none.

The Brown or Pachuca tank, Fig. 12, invented by F. C. Brown and first used in Mexico at Pachuca, the air-lift agitator, is considered to be the most successful agitating tank. It consists of a cylindrical tank 45 feet high and 15 feet in diameter, though tanks of greater height and relatively less diameter are in use. For the treatment of slime comparatively shallow tanks will suffice, but for sand a greater height with a smaller diameter will cause less tendency to clog. These tanks end in a cone with 60-degree sides; the steeper the cone the less is the tendency to settle and clog or pack. Within the tank is a hollow column 15 inches in diameter, extending from within 18 inches of the bottom to within 18 inches of the top of the tank. A $1\frac{1}{2}$-inch air pipe discharges upward at and into the bottom of the tube. To operate the tank it is filled with pulp, and air under pressure is turned into the bottom of the column. The air passing upward lightens and lifts the column of pulp within the tube, causing it to overflow and that outside the tube to enter the bottom by its greater hydrostatic head. This results in circulating the pulp up through

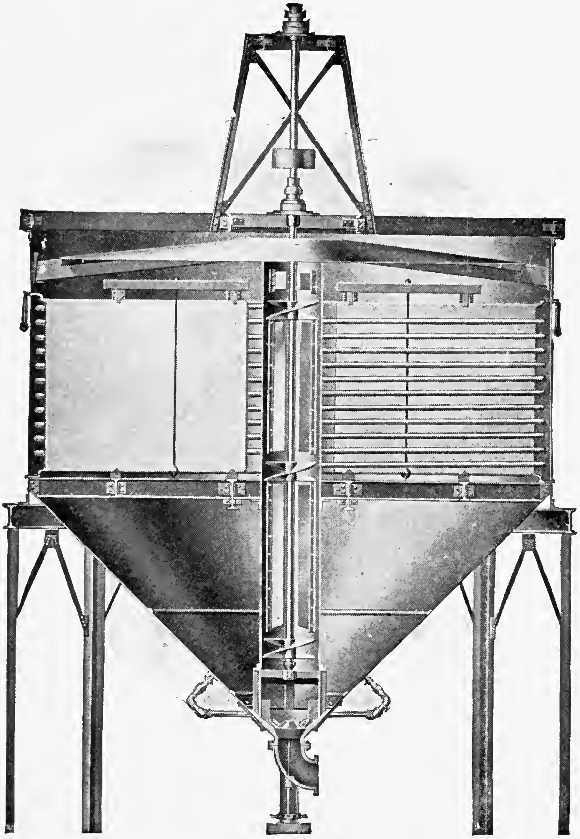


Fig. 11. — The Hendryx Agitator.

the central tube and down the tank outside of the tube. The action is that of the air lift and not similar to the hydraulic elevator. That is, it is not dependent upon the air entering the tube with sufficient force to carry the pulp up, but on the lessening of the specific gravity or density of the pulp through the air mixed and entangled, that the pulp outside may rush into the tube to equalize the hydrostatic head.

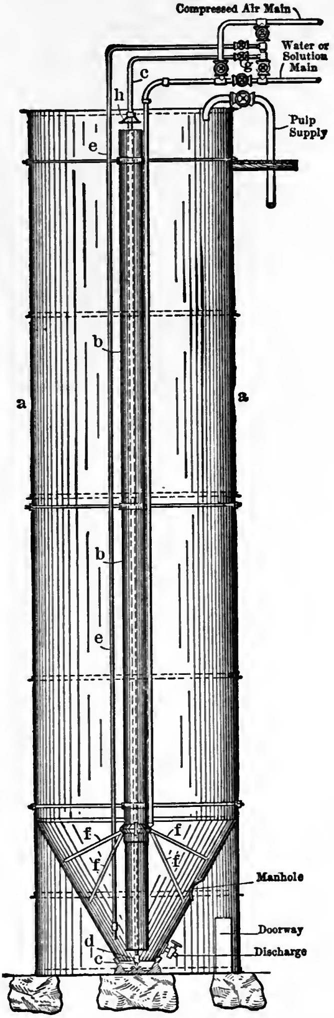


Fig. 12. — The Pachuca Agitator.

The Pachuca tank gives a thorough agitation with undoubtedly a less consumption of power than any other agitator. It can be easily started after once settled, possibly excepting when run on pure sand, in which case the charge should not be allowed to settle. It gives a good aeration, for only a gentle current of air is used, which, according to the theory of the air lift, must become more or less entangled with the pulp. It will do excellent work with a thick pulp, even as dense as 1.2 parts of solution to 1 of dry pulp. It is not adapted for decantation, for as with all tanks of relatively great height and small diameter, the settling rate per ton of clear solution obtained is too low.

The Just process has already been described as a false bottom of porous, silica-sponge bricks which may be placed in any flat-bottom tank. Air under pressure, which may be as low as 5 pounds and thereby within the field of a rotary blower, is introduced underneath the false bottom to pass through the bricks in fine jets and keep the pulp in agitation and suspension. A similar method of agitation by introducing the air through perforated pipes was early tried, but found impracticable on account of the rapid clogging of the pipes and the poor distribution of the air.

Agitation by air is supposed to cause a greater consumption of cyanide through the formation of hydrocyanic acid, though it has not been demonstrated to be a fact. Air agitation causes an increased consumption of lime by the CO_2 of the air uniting with the lime to form a carbonate. The calcium carbonate formed in this way and otherwise gives trouble by coating or clotting the pores of the leaf filters so that they become impermeable and require to be frequently treated with dilute hydrochloric acid to dissolve out the lime.

CHAPTER X

DECANTATION

THE decantation system was the first method devised for treating slime, and was the principal one used until the invention and perfection of the leaf or vacuum filter. A large number of slime plants were working until recently by the decantation process and it now plays a part in many equipped with modern filtering devices. It is used in them to remove part of the dissolved metal that the amount lost by being discharged in the residue from the filter, through the low efficiency of the filter in washing or its use as a dewaterer only and without washing, may be as low as possible. Decantation cannot be entirely eliminated until filtering devices do more efficient work then they are doing under average conditions to-day, for it seems to be a well-accepted rule that pulp containing rich solution should be sent to the filters only after the value of the solution has been reduced.

Theory of the Decantation Process. — The decantation process depends upon the principle that if 1 ton of dry slime having a value of $5.00 per ton in dissolvable gold and silver is agitated with 4 tons of solution until the maximum dissolution has been effected, allowed to settle, and the clear solution syphoned off until the slime contains one part of dry pulp and one part of solution or 50 per cent moisture, 3 tons of solution or three-fourths of the dissolved metal will have been removed, an extraction (referring to the dissolved gold and silver) of 75 per cent. If 3 tons of barren solution are now added for each ton of the dry pulp, the slime agitated for a thorough mixing, left settle, and again decanted down to 50 per cent moisture, there will be an extraction of three-fourths of the remaining value or a total extraction of 93$\frac{3}{4}$ per cent, with 31 cents of dissolved metal per ton of dry pulp yet remaining. A repetition of the cycle of operations will bring the total extraction of the dissolved metal to 98.4 per cent, which would leave 8 cents per ton in the residue. While another cycle would bring the extraction or efficiency of the washing up to 99.6 per cent and give a residue containing 2 cents in dissolved metal.

The use of a larger volume of solution for each wash will reduce the number of washes required or increase the efficiency of the washing, likewise when the slime is settled to a smaller percentage of moisture. This principle is illustrated in the case of a dilution of four parts solution to one part of pulp drawn down to one and one, giving a wash extraction of 75 per cent. Had the dilution been eight to one, the wash extraction would be 87.5 per cent. While had the pulp been drawn down to two of solution to one of dry pulp ($66\frac{2}{3}$ per cent moisture), the wash extraction would only be 50 per cent.

Decantation Process in Practice. — In plant practice it is impossible to obtain such satisfactory results in an economic way or to carry the washings to the extent that may be theorized. The defects of the process are the inordinate amount of the following: the solution to be handled and precipitated, the consumption of lime for settling purposes, the labor in giving the washes, the time consumed in mixing solution and pulp and resettling, the pulp and solution tankage space, and the water, chemicals, and dissolved metal discharged with the residue. Also the inability in many cases to get a thorough intermixing of the settled slime and the wash solution.

When treating 100 tons of dry pulp per 24 hours with a dilution of four of solution to one of dry pulp, it may require four days to give the necessary treatment of dissolving and taking off three solutions. This may be presumed to be equal to 400 tons of dry slime in the plant in the process of being treated, requiring 2000 tons of diluted-pulp tankage space, and the handling of 900 tons of decanted solution per day. If the mill crushes in water it will be almost impossible to wash the slime with any water, or the bulk of the solution in the plant will increase to a point where it must be run to waste. Consequently, a pulp discharged containing 50 per cent of moisture will cause a loss of 1 ton of the final wash solution for each ton of the dry slime discharged. The cyanide in this last wash, often a half pound or more per ton of solution, which is discharged will be lost. If the crushing is done in solution the theoretical amount of water that may be used in washing is equal to that discharged with the residue. Consequently 100 tons of water may be mixed with the 100 tons of dry pulp and the same amount of solution (to which proportion the pulp has settled), and allowed to settle, after which it should be possible to syphon off 100 tons of clear solution. In this way the cyanide mechanically lost per ton of dry slime is equal to one-half the number of pounds in a ton of the last cyanide solution added.

Decantation practice does not proceed along strictly theoretical lines, but according to methods which may be deemed most expedient, one of the principal features of which is that not all of the solution is precipitated. That which comes off in a final or the final decantations, with only a small amount of value, is used as a first or the first washes to dilute and remove the rich solution from a new charge, then precipitated to be again used as a final wash, it being considered more profitable to proceed this way than to precipitate all the solution. The time required for agitating and settling depends upon the nature of the ore and must necessarily vary in each case. The following is an example of decantation practice with an ore crushed in water, settling one to one, treated with a dilution of 4 tons of solution to 1 of dry pulp, and given three washes. The pulp is drawn from the settling or stock tanks to the agitator; to each ton of the dry pulp is added 3 tons of the unprecipitated intermediate or second wash from a previous charge. Cyanide is added to bring the strength up to .05 per cent (1 pound) KCN, no lime being necessary on account of the high protective alkalinity of the solution. Agitation is carried on for 12 hours, by which time the maximum dissolution has been effected. A short time before stopping the agitation 2 pounds of lime per ton of ore are added to be thoroughly mixed for settling the charge. After the agitation has been stopped, the solution is drawn off as fast as it becomes clarified by the settling of the slime, through a hinged pipe left down on the inside of the tank. The solution often contains some suspended matter or, from carelessness in decantation, contains considerable slime, and is run through a sand filter to be clarified before being precipitated. Eighteen hours are required for settling and decanting the first wash. Gold and silver to the amount of $5.00 per ton have been dissolved from the ore; this has been diluted by the four parts of solution to enrich each ton of solution $1.25, to which is added $0.45, for the unprecipitated intermediate wash carried $0.60 per ton and 3 tons of this solution were used to 1 of water in the settled slime. This gives a solution of $1.70 to be decanted and sent to the zinc boxes as the first wash. After the decantation there still remains 1 ton of solution worth $1.70 with each ton of dry slime. To each ton of this solution is added 3 tons of the unprecipitated final or third wash of a previous charge, containing $0.16 per ton. This gives a value to the resulting mixture of $(1.70 \div 4) + (\frac{3}{4}$ of $0.16) = \$0.55$ per ton of solution. An agitation of 3 hours is given to thoroughly mix the charge, followed by settling and decanting for 18 hours to remove the second or intermediate wash. After which 3 tons of a precipitated first wash solution practically barren are added and agitated for 1½ hours and the charge pumped into another tank to insure thorough mixing, requiring 2 hours. Another settling and decanting period of 18 hours is given to draw the solution down to a one and one consistence. The value of this last wash is a little higher than the theoretical amount which should be $(\$0.55 \div 4) = \$0.14$ per ton, consequently the residue which is now discharged contains a little over $0.14 in dissolved metal per ton of dry pulp, and an amount in cyanide equal to that of the last wash given, which was about 0.4 pound KCN. Had all the wash solution been precipitated, the final tailing would have a theoretical value of 8 cents in dissolved metals. If the intermediate wash had been precipitated, but not the final, the theoretical value remaining would be 9 cents. In the first case 6 tons more of solution would have to be precipitated for each ton of dry pulp at a cost of 3 cents per ton of solution or 18 cents per ton of ore. In the second case, 3 tons more of solution would have to be precipitated at a cost of 9 cents per ton of ore to obtain the theoretical difference of 5 cents (.14 − .09).

If the dissolution of the last of the gold and silver is slow, the agitation may be stopped and the first wash removed with a reliance on the solvent activity of fresh solution or of the long, general contact to effect the dissolution still to be made in time to allow it to be removed by the washes. Owing to the inability to get a thorough intermixing with mechanical agitators and flat-bottomed tanks, it is the custom in many plants to transfer the pulp from one tank to another when applying a wash. In this way any sand that has hugged the bottom and corners of the tank together with its adsorbed rich moisture responds to the washing process. If only one transfer can be made, it should be effected after the bulk of the value has been removed by one or two first decantations. The pulp may be transferred after the last agitation to a deeper settling tank than usual and without an agitating device, wherein only the thicker, denser pulp is drawn off intermittently from the bottom of the tank as a residue to go to the slum pond. The drawing off of part of the settled pulp together with a decantation of the clear solution from the top makes room for each charge as added; the entire amount of pulp not being withdrawn at any time. The use of comparatively deep tanks, preferably with cone bottoms, in this way, as a more or less continuous or intermittent method of decanting and discharging, gives a sludge to be discharged having a higher percentage of dry slime, owing to the greater length of time allowed for settling and the pressure weight of the deep column of pulp. Many attempts have been made to make use of these principles in connection with that of cone or other overflow settling devices, especially for a continuous washing and treatment system, but without much practical success except in the case of the last wash.

Mechanical Decantation Processes. — The Adair-Usher process developed in South Africa is dependent somewhat on the above principles. After the metal has been dissolved, and without allowing more than a slight settling of the slime, barren solution is introduced through pipes evenly over the bottom of the slime tank, which should be comparatively deep. This solution enters in sufficient quantity to rise at a rate slightly slower or equal to the settling rate of the slime and to always give a clear overflow of solution from the tank. In theory the barren solution lifts or displaces the rich solution and washes the slime in suspension. When the washing has been carried as far as practicable, the inflow of solution is stopped, the slime left settle, and the clear solution decanted off as usual. This process does not do away with decantation in its entirety, but is only an adjunct or expedient to reduce the number of decantations.

While continuous decantation without mechanical means has not been a practical success, continuous decantation with mechanical appliances has been rendered possible by the introduction of the Dorr pulp thickener. This appliance is now in use dewatering the pulp to a low percentage of moisture, followed by diluting the pulp with wash solution to be again dewatered, using a sufficient number of such machines to thoroughly wash out the dissolved metal. This method seems to be well adapted for some conditions or in a modified form to precede filtration for the purpose of reducing the value of the pulp and solution supplied to the filter, thereby lessening the mechanical loss through the low efficiency of the filter wash.

CHAPTER XI

FILTRATION

Plate and Frame Filter Press. — Owing to the trouble and low efficiency encountered in treating slime by the decantation process, the plate and frame filter press was early made use of for dewatering and washing the dissolved gold and silver out of slime pulp. The filter press shown in Fig. 13 was already in wide use in clay-working plants for dewatering a thin, clay pulp, and in the sugar-making industry for filtering the juices and washing the residuum in the process of making sugar. The plate and frame filter press as used in the cyanide process consists of a number of solid metal plates with a hollow plate or frame alternated between each. These plates are square or rectangular in shape and up to 3 or 4 feet in dimensions, the frames being from 2 to 4 inches thick. The alternate plates and hollow frames are set in an upright position face to face by means of projecting arms or shoulders on each plate and frame resting on two strong bars on each side of the assembled press. This makes a box-like structure with a height and width equal to the dimensions of the plates and frames, and with a length dependent upon their thickness and their number, which may be as high as fifty. Each end of the press is closed with a solid plate, while between each plate and frame is placed a sheet of canvas as a filter cloth. At the ends of the press are screw devices for forcing the plates together and thereby making the press water-tight, or hydraulic or air pressure instead of hand screws is used to open and close the presses. Each plate and frame contains lugs or shoulders bored with holes. When the press is in position these holes form a passageway for the slime pulp and solution. The passageway for the slime pulp is provided with a small opening into each frame or hollow plate only. This allows the pulp under pressure to enter and fill each hollow frame. The solution in the pulp under pressure is forced through the canvas on one of the sides and runs down between the canvas and the solid plate, which is corrugated to better allow the passage of the solution between the canvas and the plate, to a small opening in the corner of the plate where it runs to another passageway or channel of bored holes leading to the solution tank. In some types each plate is provided with small individual cocks emptying into a launder leading to the solution tank. The flow of pulp is stopped as soon as the press is filled with a solid cake of pulp, and wash solution or air followed by wash solution is pumped through a third passageway to enter between the other canvas and the other corrugated side of each plate. Whence it is forced through the canvas into the cake of pulp, washing it or displacing the moisture in it, passing through the opposite filter cloth, and running down between the corrugations to pass out through the hole in the corner of the plate into the passageway leading to the solution tank, just as the solution does which is expressed from the pulp in filling the press. After sufficient washing with barren solution, a water wash, perhaps preceded by an air wash or displacement, is introduced through the same passageway. Finally air under pressure is ad-

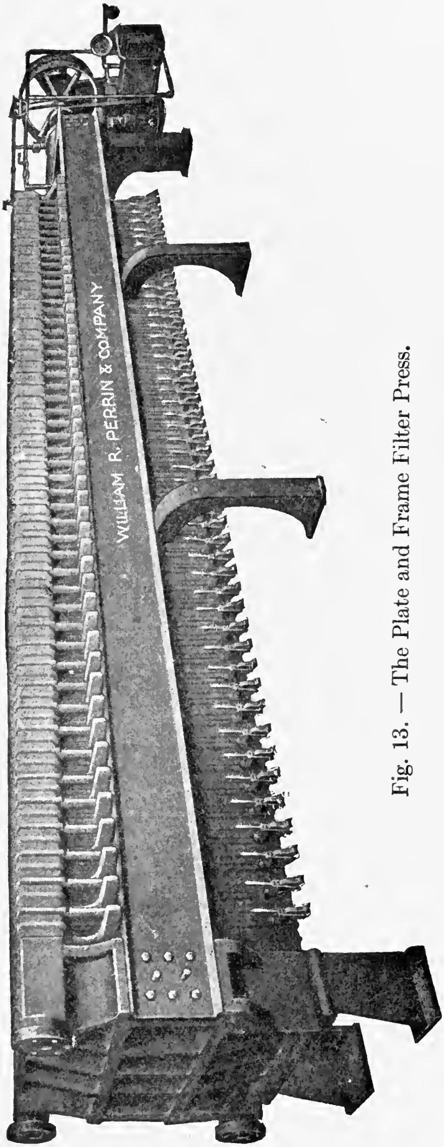


Fig. 13. — The Plate and Frame Filter Press.

mitted, which displaces a large part of the moisture so that a cake containing as low as 15 to 25 per cent of moisture may be obtained. After which the closing screws of the press are opened, allowing the plates and frames to be separated and the washed cake of slime to be dropped into a car or sluice for discharge to the residue dump. The plates, frames, and filter cloths are next brought together, the closing screws are tightened, and a new cycle of operations is started.

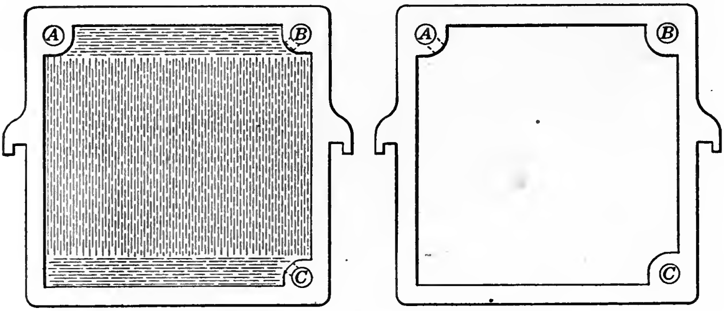


Fig. 14. — Plate and Frame of Filter Press.

A. Passageway for pulp. *B.* Passageway for entering solution. *C.* Passageway for departing solution.

The pulp may be forced into the press by pumps capable of giving a high pressure, by gravity under a high head, or by compressed air and a monteju, though the last method is practically obsolete. The monteju is a large, closed, metal tank or receiver into which the pulp is run as required. To fill the press, the monteju is closed and air under pressure admitted at the top to force or displace the pulp into the press. The principal advantage of the monteju is the lack of wear through the attrition of the pulp.

Filter Press Practice in Australia. — The following details* of treatment with the Dehne press, the best known of the standard filter presses and which has been successfully used in Australia, represents the highest type of work. The slime from the agitators having a consistence of about 1 and 1, is pumped to the presses by a Pern pump having three plungers, 12 by 10 inches, running at 20 revolutions per minute. This is a powerful pump

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* M. W. von Bernewitz in "Slime Treatment at Kalgoorlie," Min. and Sci. Press, Dec. 14, 1907. More Recent Cyanide Practice, pp. 82.

and will fill a press in 10 minutes, lifting in that time about 10 tons of pulp, and charging against a final pressure of 60 pounds per square inch. The time taken in filling may be divided as follows:

Up to 25-lb. pressure	4 min.
Up to 50-lb. pressure	3 min.
Up to 60-lb. pressure	1 min.
Finishing with safety valve blowing off at 60-lb.	2 min.
Total	10 min.

Average screen tests of the ore are:

Held on 40-mesh	Nil per cent
Held on 60-mesh	0.5 per cent
Held on 80-mesh	2.5 per cent
Held on 100-mesh	4.3 per cent
Held on 150-mesh	4.9 per cent
Passed 150-mesh	87.5 per cent

The three Dehne presses each have 50 3-inch frames for the slime. Each press will hold about 4.5 tons dry slime. After a press is filled, the slime is washed for 25 minutes with weak cyanide solution, and a water wash of 5 minutes at 100-pound pressure, during which time each ton of slime is washed with 2 tons of solution. The washing is done by a similar pump to that used in filling, only that it runs at 13 revolutions per minute. The final water wash is dispensed with when there is an excess of mill solution, the press getting 30 minutes with wash solution. The decrease in the assay value of the solution during the washing is:

At start of wash	$12.50
After 5 min.	6.60
After 10 min.	1.50
After 15 min.	1.50
After 20 min.	1.00
After 25 min.	1.00
After 30 min.	0.80

After washing, the content of the press is dried with air for 10 minutes at 80-pound pressure. The press is then opened ready for discharging. Two men empty 11 presses per shift of 8 hours, say 50 tons, onto a traveling horizontal belt-conveyor 18 inches wide. Most of the solution used in washing the presses passes into the mill solution to be again used in the grinding pans, etc. The cyanide solution is made up to .07 per cent (1.4 pounds) and the consumption averages 1 pound per ton treated. An average of three months' costs of slime treatment is:

Agitation and cyaniding	$0.34 per ton
Filter-pressing	0.41 per ton
Precipitation, etc.	0.12 per ton
Disposal of residue	0.04 per ton
Total	0.91 per ton

The time taken in the different press operations is:

Filling press	10 min.
Washing	30 min.
Drying	10 min.
Discharging	30 min.
Screwing-up, etc.	10 min.
Total	1½ hours

The daily capacity is 123 tons of roasted telluride ore and 43 tons of retreated old residue, having an average recovered value of $14.10.

The Merrill Press. — The Merrill press, as developed by C. W. Merrill and shown in Fig. 15, does not differ in principle from the standard filter press. Its dimensions are:

Number of frames	64 to 92.
Size of frames	4 by 6 ft.
Length	Up to 45 ft.
Capacity per charge	Up to 25 tons.
Thickness of cake	3 to 4 ins.

With ores in which the metal goes into solution quickly and that are not too slimy, the dissolution as well as the washing may take place in the press. The pulp is dewatered down to 3 parts of water to 1 of dry slime, or thicker if there is a tendency for the sand to classify in the filling process. It is run into the press by gravity under a pressure of 20 to 30 pounds per square inch (equal to 40 to 70 feet fall). After the press is filled, the water is displaced and the pulp partly dried by air under pressure, when cyanide solution is slowly pumped through the charge followed by air under pressure. The application of the dissolving solution and the aeration is alternated until the precious metals are dissolved, when the usual washing with weak solution and water with final air displacement follows. On the completion of the treatment, which may require 6 hours, the charge is washed from the press by water introduced through a "sluicing bar," which is a pipe extending lengthwise throughout the press. This pipe is provided with nozzles in each chamber and an outside mechanism which causes it to revolve back and forth sufficiently to direct the water discharged from the nozzles into every part of each frame or chamber. In this way the charge is washed out of the press through the balance of the passageway in which the "sluicing bar" lies. From 4 to 8 tons of water per ton of dry slime are required to wash the press out. A large part of the water may be saved by the use of pulp-thickening tanks. Where the metal has been dissolved before entering the press, the operations are similar to those of an ordinary filter press, except in regard to sluicing out the press.

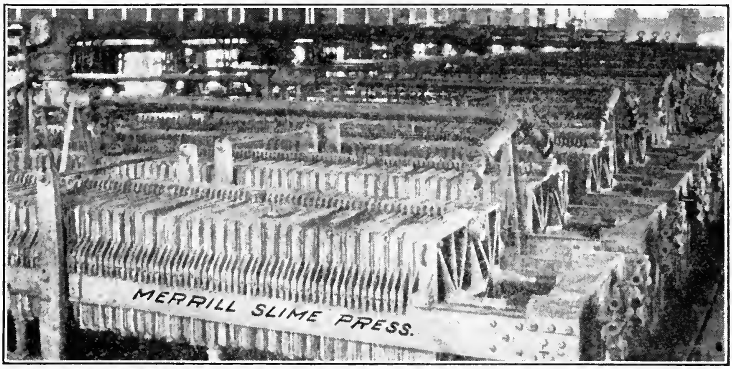

Fig. 15. — The Merrill Filter Press.

It is reported that 50,000 tons of dry slime have been treated per month at the Homestake slime plant at a total cost of 25 cents per ton, of which the filtering cost amounted to $1\frac{2}{3}$ cents per ton. The value of the untreated slime is said to be about 85 cents per ton and the extraction to be 90 per cent. This is a record that has never been approached by any other slime treatment or filtration system, but much of which must be credited to the costly plant and large tonnage available.

The plate and frame filter press is best adapted for the granular slime of quartzose ore, and then requires the addition of considerable fine sand to render the slime more permeable by the solution. The filter press as ordinarily used will not do good washing on an abnormally talcose, clayey slime. Such material is washed in the Merrill press (dissolution of the value having taken place before filling the press) by what is termed "center-washing." In this method the solution expressed from the pulp is allowed to flow through both canvases and the frames are not filled with solid cakes of pulp. An opening of a quarter of an inch or more is allowed in the center of each cake, through which the barren solution immediately following the pulp, without any interlude of aeration, and finally the water wash and air displacement pass to wash the metal-bearing solution out of the slime. The following is the data on a cycle of operations working in this way: Filling with a pulp of 1½ parts of solution to 1 of slime, 30 minutes; washing with 2½ tons wash solution and water per ton dry slime, 35 minutes; discharging, 45 minutes; total, 1 hour and 50 minutes.

Vacuum or Pressure Leaf Filters. — The vacuum or pressure-leaf filter differs entirely from the plate and frame filter press. The principle of these filters is the use of a flat slip or bag of canvas over a suitable thin frame of wood or metal. The inside of the filtering leaf in the suction type is connected to a suction or vacuum pump. On completely immersing the leaf into a homogeneous slime pulp and starting the suction removing the air and solution from the interior of the leaf, the atmospheric pressure causes the slime to collect on the leaf and the solution to pass within and be drawn through the pump or into the vacuum tank through which the pump works. This results in collecting a layer of thickened pulp on the filter leaf, which increases in thickness until the atmospheric pressure is no longer able to force the solution through the cake formed and consequently there is a vacuum within the leaf. The success of the leaf filter has hinged upon the equal permeability of the cake formed, for should any part at the time of forming the cake be more permeable than the rest, more pulp will be drawn to that part and more solution will pass through it, until the resistance to filtration at this point and all others is equal. With a pulp that is homogeneous and a filter cloth of equal permeability, a cake of

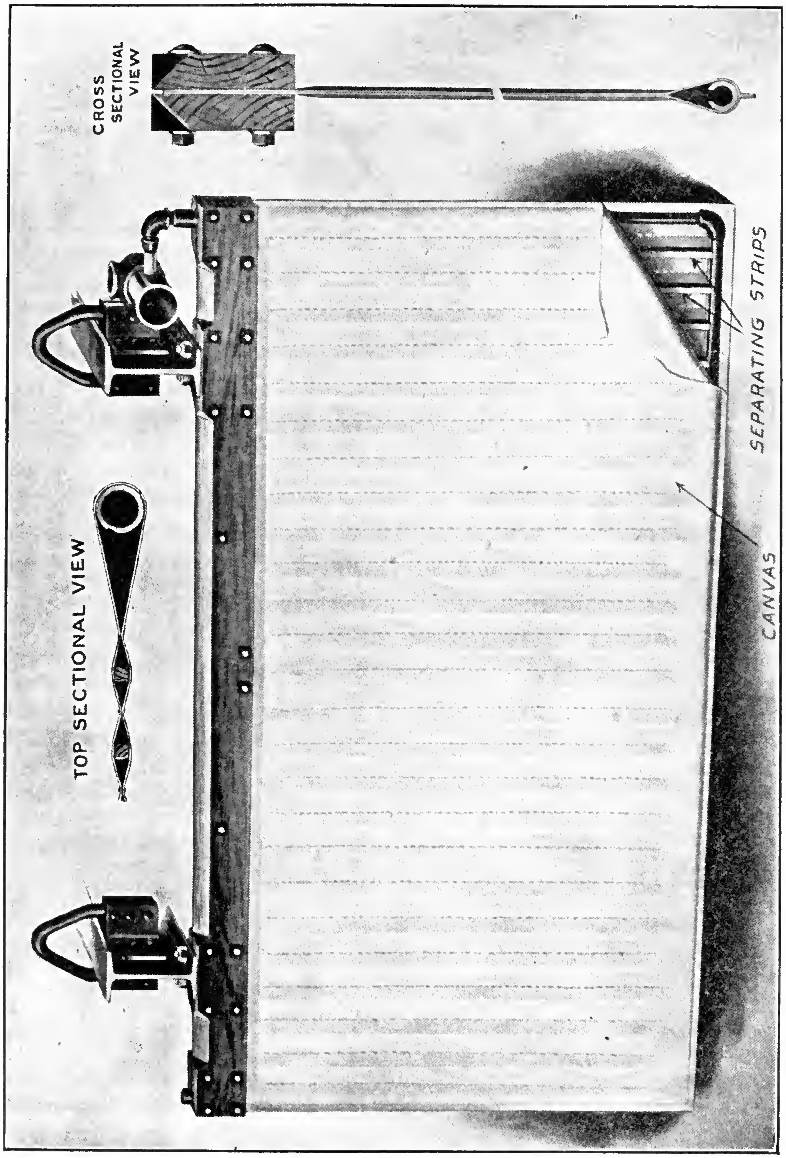

Fig. 16. — Moore Filter Frame.

even thickness will be formed. But should the pulp tend to classify and the sand or pulp in general sink to the bottom, the lower part of the cake will be thick and sandy, while the upper will be thin and slimy. Similarly, a spot on the canvas leaf that is less porous, through a coating of carbonate of lime or other, will have a thinner coating of material, but one of the same composition as that of the surrounding cake.

Having formed a cake upon the leaf as thick as the atmospheric pressure will allow, it may be removed from the pulp without breaking or cracking and immersed in a wash solution, provided the vacuum is continued to an extent just sufficient to hold the cake on the leaf intact. The continuation of the vacuum will cause the atmospheric pressure to force the solution through the cake into the interior of the leaf to be withdrawn by the vacuum pump. This passage of the solution through the charge or cake is the washing process, just as takes place in a leaching vat or in the plate and frame filter press. An equal amount of solution will be drawn through all parts of the leaf, even if the texture and thickness of the cake vary as noted before, for unless the cake cracked or sloughed during the removing process, its resistance will be the same all over its area.

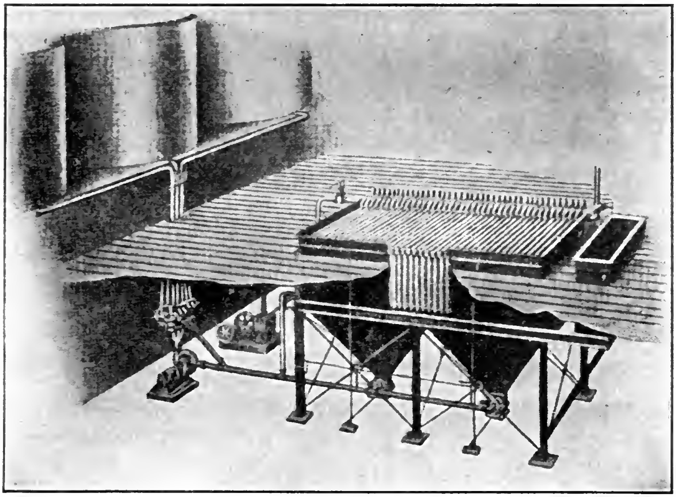

Fig. 17. — The Butters Filter.

After passing barren solution or wash water through the cake until the dissolved metal is displaced and removed from the interior of the leaf, the leaf and its cake are removed and exposed in the air. The vacuum is continued to remove as much of the wash water as practicable, especially should the cake be washed in barren solution and it be desired to keep the mechanical loss of cyanide as low as possible. The next step is to introduce air and water into the interior of the leaf to remove or slough off the treated slime, when a new cycle of operations may be taken up.

Classification of Leaf Filters. — The different leaf filters divide themselves into two classes, the suction or vacuum and the pressure filters. These are further divided into the stationary and movable filters, into the continuous and intermittent, and then into lesser gradations. The Butters is the best known of the stationary, intermittent, vacuum filters. A large number of leaves up to two hundred are arranged in a box with cone bottoms. The pulp is run by gravity or pumped into the bottom of this box to rise and keep the leaves submerged. A vacuum is applied to the interior of the leaves to form a cake an inch or more in thickness. After the cake is formed, the surplus pulp is removed by gravity or pumping to the stock tank supplying the filter, and wash solution or water introduced into the box and drawn through the cakes. Having removed the wash solution, the box may be filled with water to assist in carrying out and removing the slime, or the slime may be dropped into the bottom of the box to receive a little sluicing or water to make it slide out and run to the slum pond. In either case the cake is dropped by reversing the force that has formed and held it in place, by turning air or water or both into the interior of the leaf.

The Moore is the best known of the movable, intermittent vacuum filters. The series or "basket" of leaves is fastened together in such a way that it may be easily and quickly lifted and transferred from one tank to another by means of a traveling crane or a lifting and revolving device. The basket of leaves is lowered into and kept submerged in the stock-pulp tank until the cake is formed. It is then raised and transferred by means of the crane to an adjoining wash-solution tank where it is washed. After which it is lifted and transferred to a tank or hopper into which the cake or charge is dropped in a way similar to with the Butters filter.

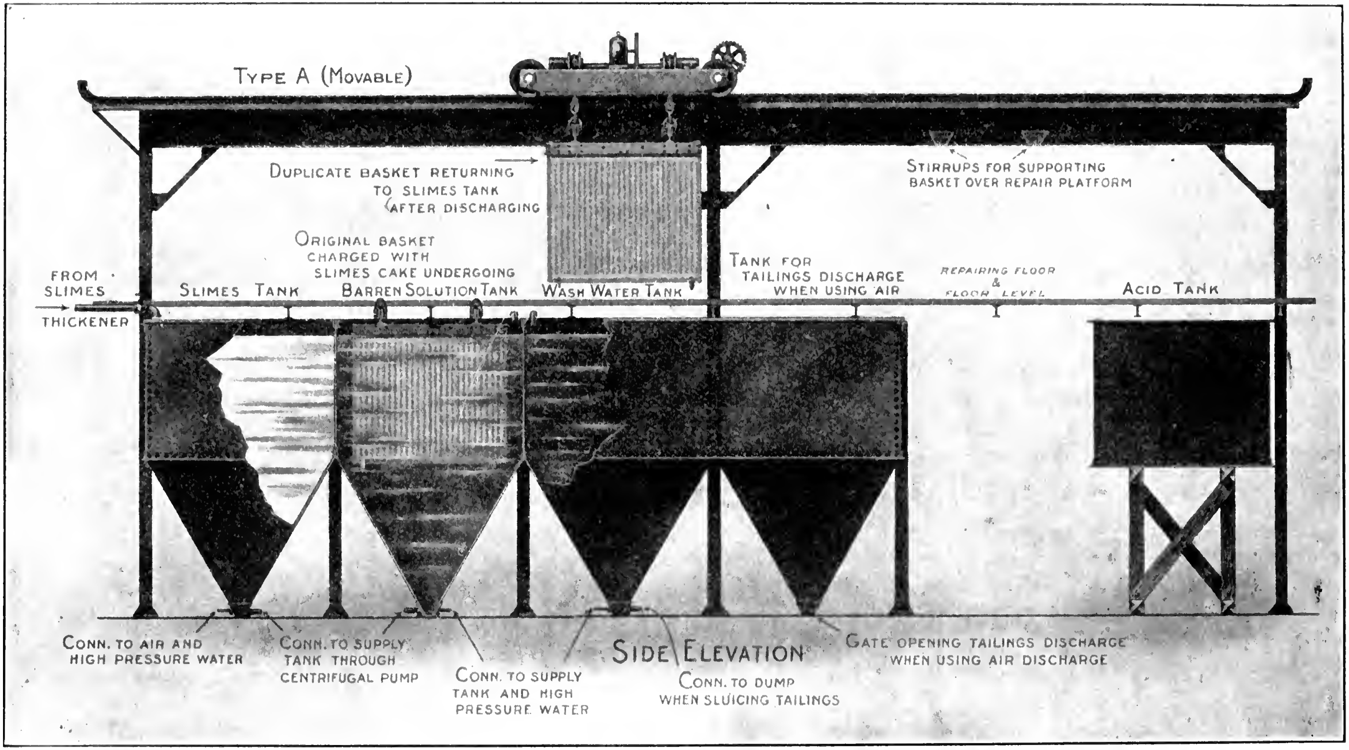

Fig. 18. — The Moore Filter.

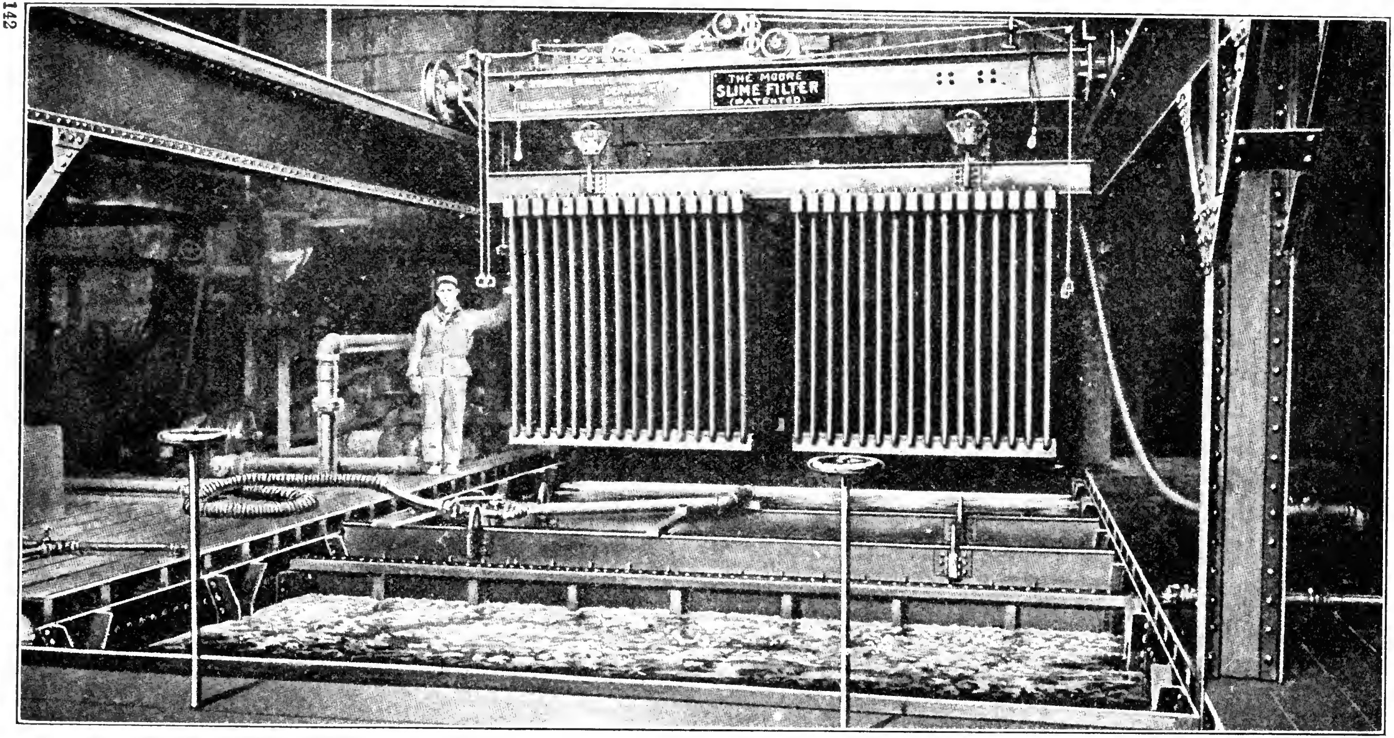

Fig. 19. — A Moore Filter, as used at Deadwood, South Dakota.

The Oliver shown in Fig. 20 is a vertically-revolving, continuous vacuum filter. It consists of a revolving drum which may be as large as 12 feet in diameter and 18 feet broad. The surface or face of the drum or wheel is prepared as a leaf-filtering surface and divided into a number of compartments, connected on the inside with a vacuum or suction pipe and a pipe for admitting compressed air. The drum is partly immersed in a tank or box of thick pulp and revolves at a slow rate of speed. The mech-

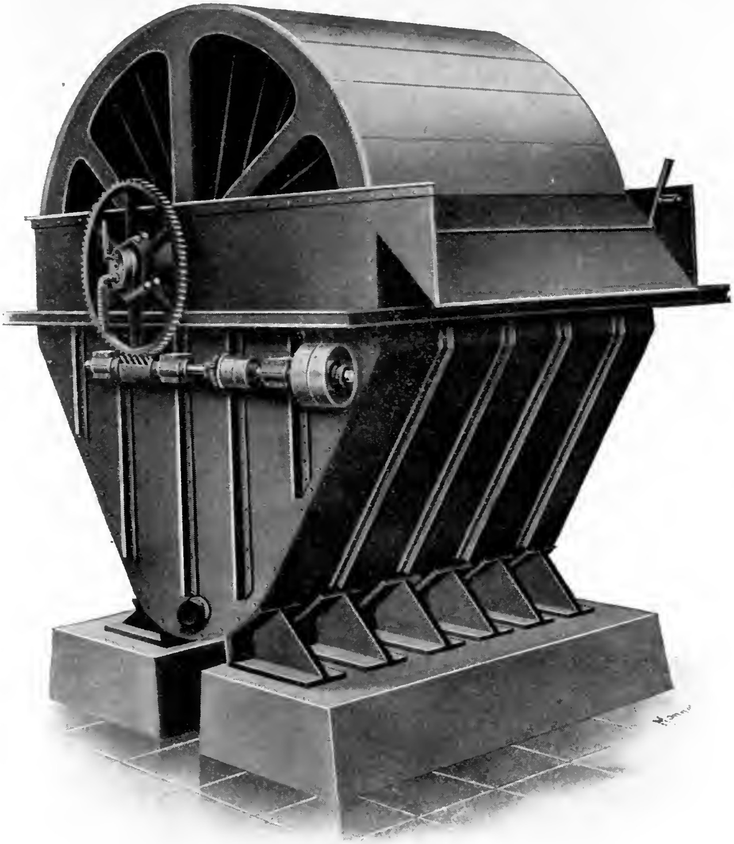

Fig. 20. — The Oliver Continuous Filter.

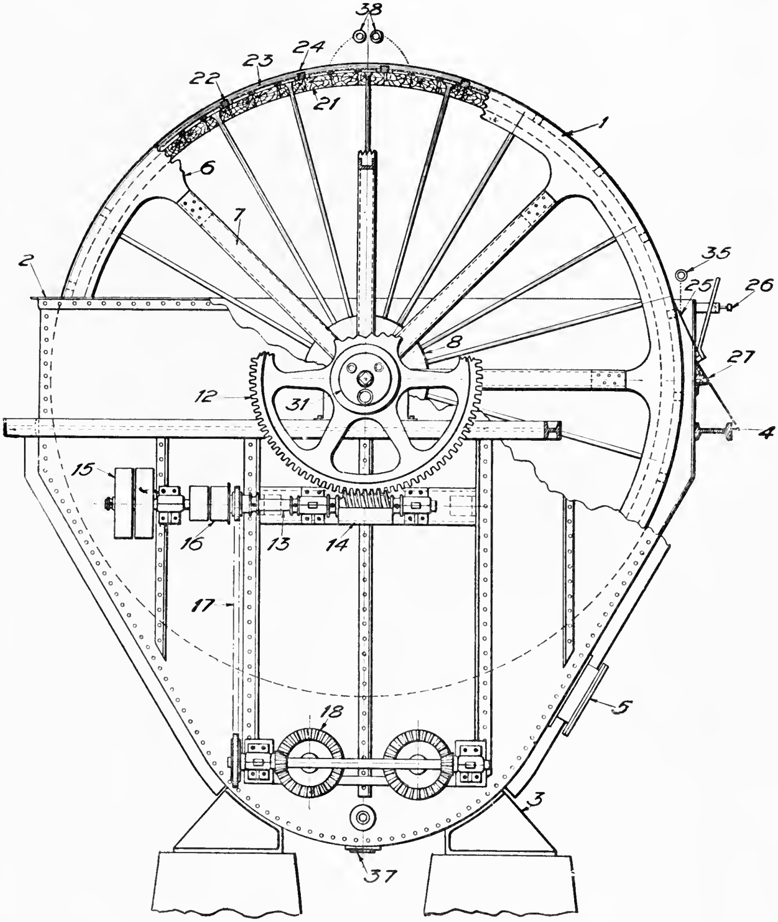

Fig. 21. — The Oliver Continuous Filter (End View).

List of Parts.

1. Filter Drum.
2. Steel Filter Tank.
3. Cast Iron Pedestals.
4. Steel I Beam Frame.
5. Manhole.
6. Cast Iron Spider Rim.
7. Channel Steel Arms.
8. Hollow Trunnion.
9. Steel Shaft.
10. Main Bearings.
11. Stuffing Boxes.
12. Worm Drive Gear.
13. Worm Shaft.
14. Oil Well for Worm.
15. Filter Drive Pulleys.
16. Pulleys for Agitator and Wiring.
17. Chain Drive for Agitator.
18. Bevel Gears on Agitator Shafts.
19. Agitator Shafts.

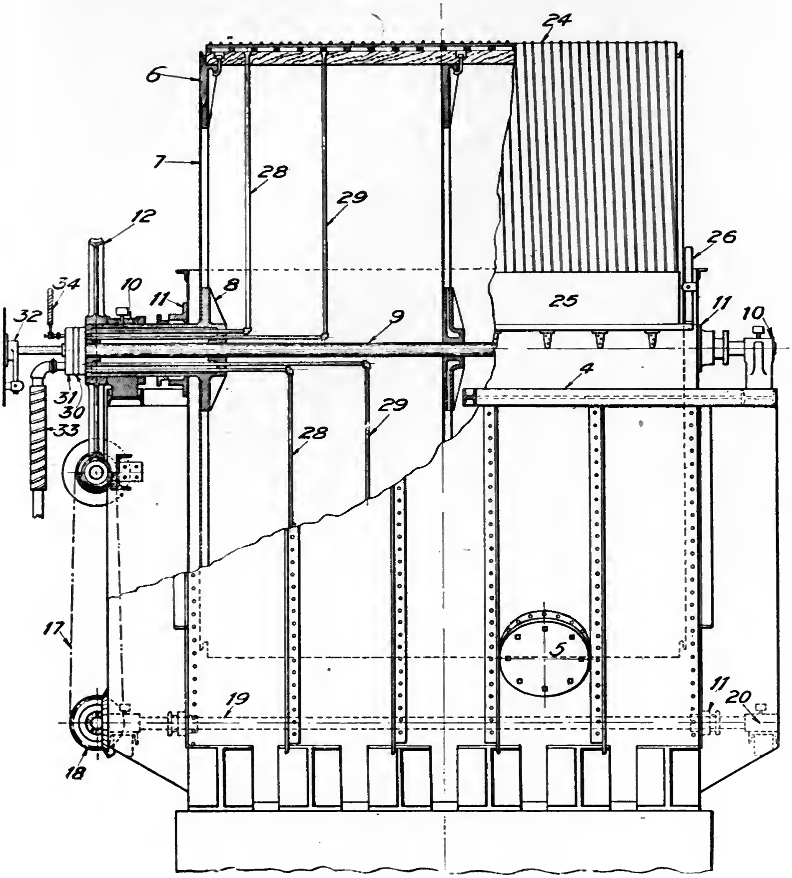

Fig. 22. — The Oliver Continuous Filter (Side View).

List of Parts.

20. Agitator Shaft Bearings.
21. Wood Staves for Drum.
22. Section Division Strips.
23. Filter Medium.
24. Wire Winding.
25. Steel Scraper.
26. Scraper Adjustment.
27. Tailing Apron.
28. Vacuum Pipes.
29. Compressed Air Pipes.
30. Regrinding Valve Seat.
31. Automatic Valve.
32. Adjusting Lever for Valve.
33. Vacuum Hose Connection.
34. Compressed Air Connection.
35. Discharge Spray Pipe.
36. Emergency Agitator Pipe.
37. Drain Flange.
38. Wash Water Pipes.

anism acts automatically to cause a vacuum which makes a cake of $\frac{1}{4}$ to $\frac{1}{2}$-inch thickness as the drum passes through the pulp. As the cake emerges from the pulp the atmospheric pressure displaces a large part of the solution adsorbed by the slime, after which a line of wash water or solution across the width of the drum applies the wash. Air is finally drawn through to displace as much of this wash as possible. Just before each section of the drum with its washed and air-dried part of the cake reënters the pulp, the vacuum is automatically shut off and air under a light pressure introduced to cause the cake to drop off, assisted by a scraper.

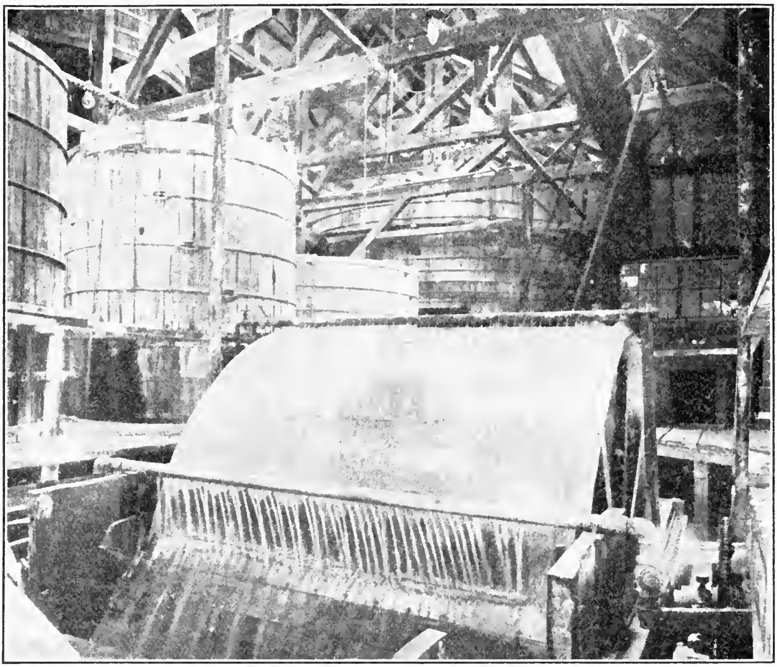

Fig. 23. — A 50-ton Oliver Continuous Filter, at North Star Mines Co., Grass Valley, California.

The Hunt is a horizontally-revolving, continuous vacuum filter as shown in Fig. 24. It consists of a horizontal, annular filter bed underneath which a vacuum only is applied. A carriage mounted inside of the filter ring and supported on a track outside of it revolves continuously. The pulp is roughly classified into sand and slime and each is delivered to a hopper at the middle of the filter.

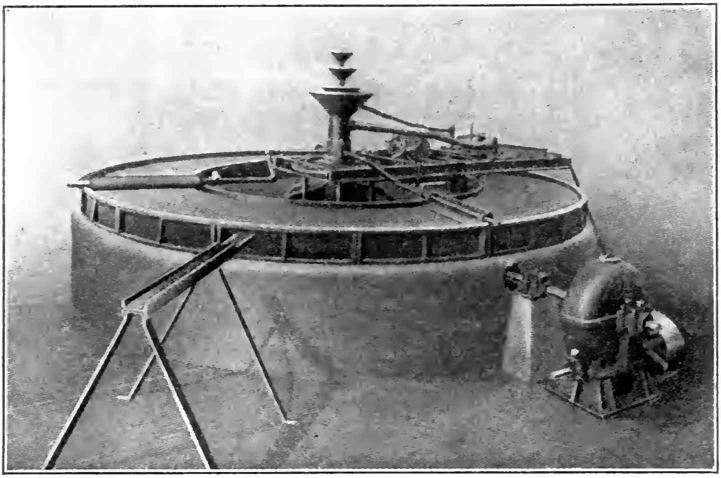

Fig. 24. — The Hunt Continuous Filter.

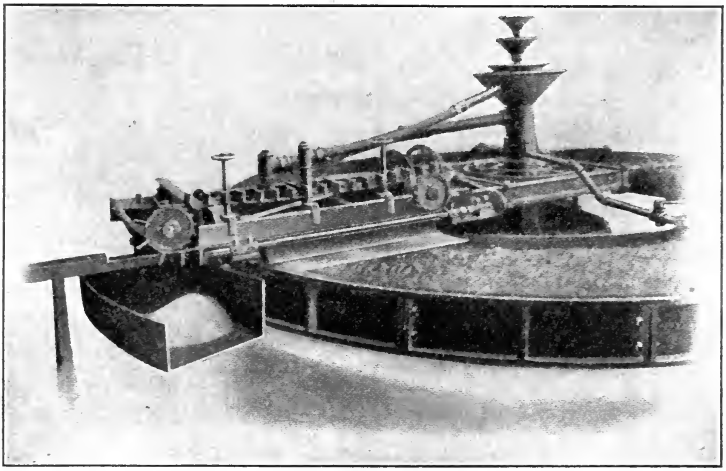

Fig. 25. — Carriage of the Hunt Continuous Filter.

From the hoppers the pulp runs out through arms to be delivered evenly across the width of ,the filter ring. The sand is first deposited to form a good filtering medium, and is immediately followed by a layer of slime delivered over it. The vacuum operates to withdraw part of the moisture from the bed of deposited pulp, while a pipe delivering a spray of wash solution or wash water follows at a suitable interval; the vacuum finally drying the pulp which is scraped off to fall over the outer edge of the filter ring by a scraper placed in front of the arm delivering the sand. A novel feature is the use of a filter bed consisting of triangular, wooden slats filled with coarse sand and dispensing with filter cloths. A similar device is being used in South Africa to dewater the sand before adding cyanide solution and transferring to the leaching tanks.

The Ridgeway Filter, as shown in Fig. 26, is a horizontally-revolving, continuous vacuum filter with an intermittent action. It consists of an annular ring made up of a pulp, a wash-water, and a discharge tank. A revolving carriage with suitable mechanism carries 14 trays of over 3 square feet area each. The under sides of these trays or plates are prepared as leaf filters with vacuum and compressed-air attachments. As the carriage revolves the trays or leaves are first immersed in the pulp, through which they pass and from which they are mechanically lifted to emerge with a cake of pulp and then to be lowered into the wash solution through which they pass. Each tray is finally lifted out and brought over the discharge hopper, where the vacuum is automatically cut off and compressed air admitted to detach the cake, when the tray again passes into the slime pulp.

The Kelly, as shown in Fig. 27, is a movable, intermittent pressure filter. It consists of a long boiler-like tank set on a small incline. The lower head of this pressure tank is fitted with a quick-acting closing device. Upon opening the clamp, the frame carrying the head together with the set of vertical filter leaves running the length of the tank may be run out of the tank chamber, running on suitable tracks within and without the tank. After dropping in the usual way the cake adhering to the leaves, the carriage now lightened by the removal of the load of slime can easily be drawn into the pressure tank and the head locked. Pulp is then pumped into the tank under suitable pressure which may be as high as 80 pounds per square inch. As soon as the air has been displaced and the tank is consequently full of the pulp, the cake commences to form. The pressure of the pump acts just as atmospheric pressure would, except that on account of the increased pressure the cake is made and washed in a comparatively short time. The pressure of the pump in forming the cake causes the solution expressed and filtered out of the pulp to pass into the interior of the leaves, there to run through suitable pipes out of the press into the solution tank. The cake having formed, which is indicated by the decrease in the solution flowing from the press, pumping pulp into the tank under pressure is stopped. The surplus pulp is then allowed

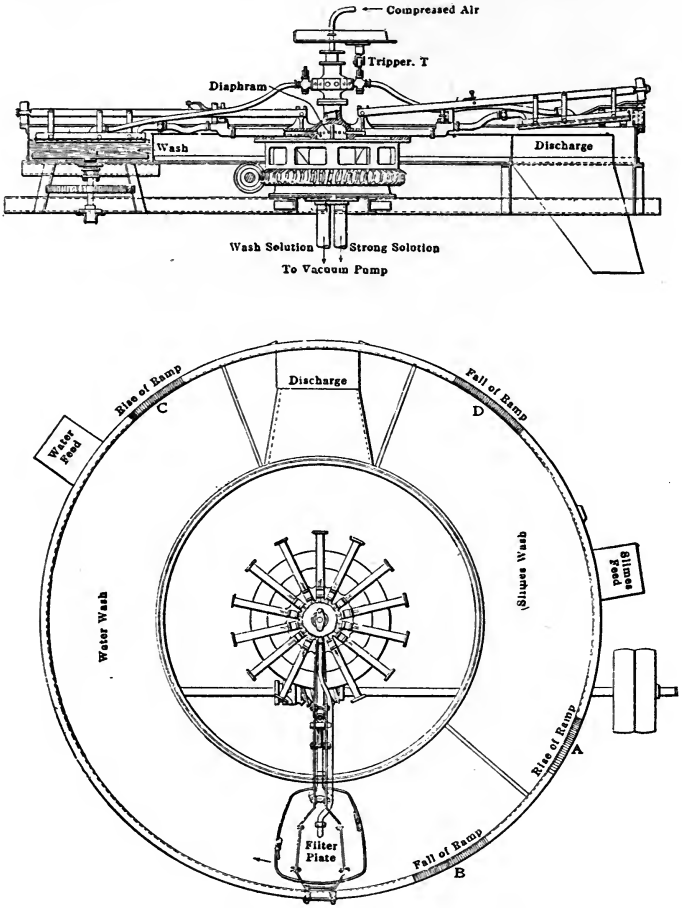

Fig. 26. — Section and Plan of Ridgeway Filter.

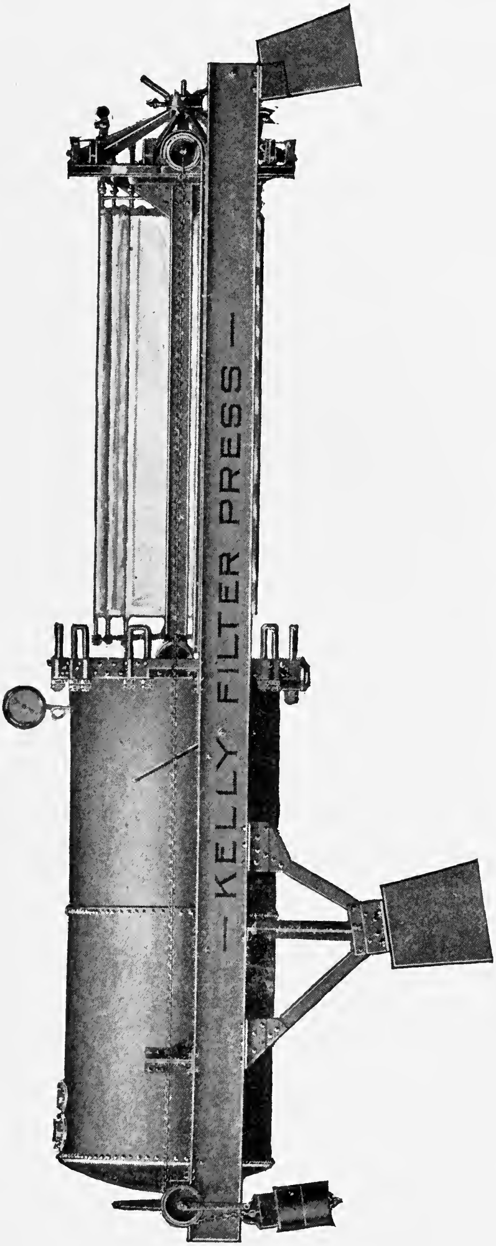

Fig. 27.—The Kelly Filter Press.

to run from the filter back into the stock tank, being displaced by air under a low pressure to hold the cakes in place. After the surplus pulp has been removed, wash water or solution is pumped into the tank and continued under high pressure for as many minutes as experiments have indicated are required to give a good wash or to pass a certain volume of solution through, when the surplus solution is removed in the same manner as the surplus pulp was. The remaining moisture in the cake is displaced by the air under pressure until no more solution runs from the press. The final step is to cut off the air pressure, and unclamp the head to run out the carriage and remove the load of slime. The details of the press have been perfected to such an extent that a battery of four presses has been handled by one set of levers and valves, and two batteries of four presses each have been tended by one operator and a helper.

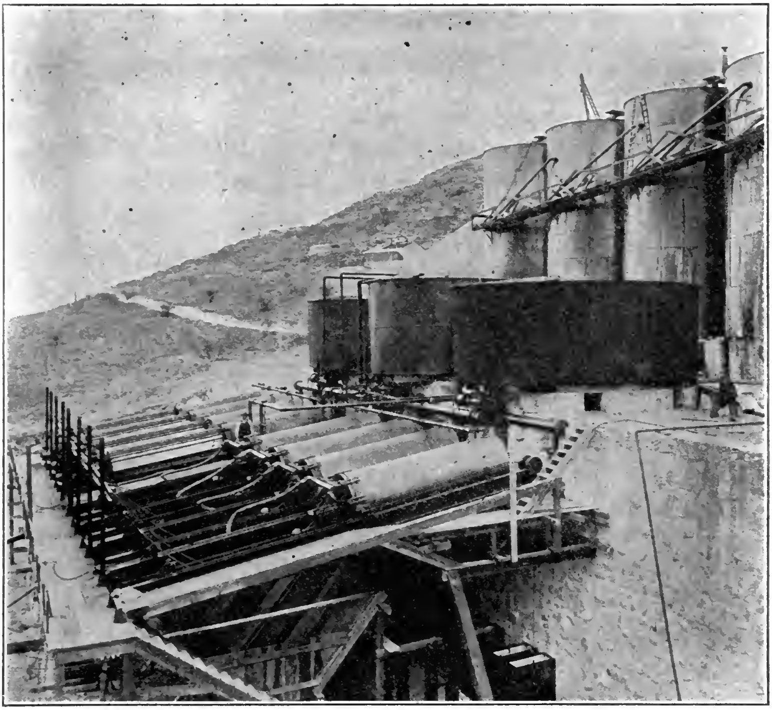

Fig. 28. — Kelley Filter Presses, and Continuous Agitation.

The Burt rapid filter is a stationary, intermittent pressure filter. It is somewhat similar to the Kelly, except that the filter leaves are suspended vertically at right angles to the length of the tank, which, being set at a considerable incline that the surplus pulp may easily run to the outlet, makes the leaves in the shape of an elongated circle. These leaves are only removed for repairs, consequently the slime cake is discharged by introducing air and water into the interior of the leaf and letting the pulp slide out through a discharge opening.

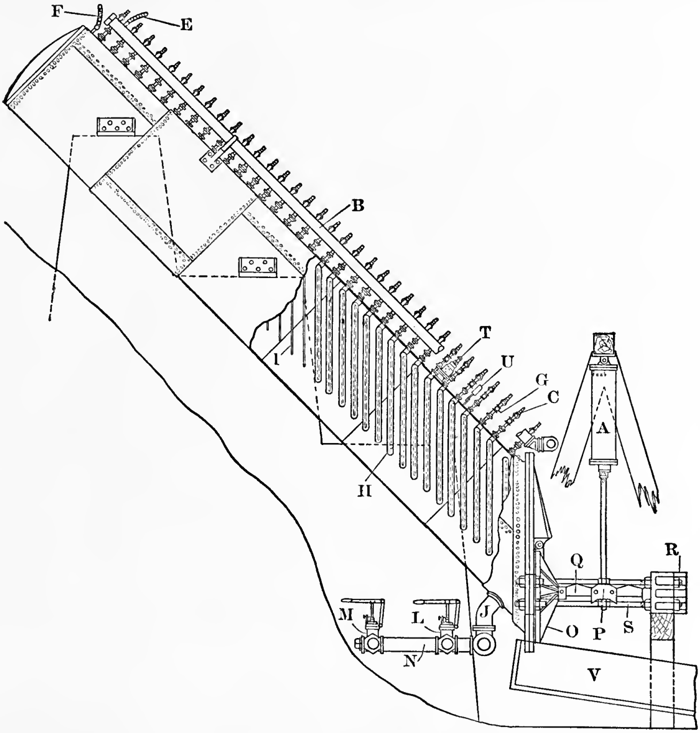

Fig. 29. — The Burt Rapid Filter.

The Burt revolving filter is a revolving, intermittent pressure filter. It consists of a long revolving shell very similar to a tube mill or revolving drier. It has a length of about 40 feet and a diameter of about 42 inches, and revolves at a speed of 15 revolutions per minute. The interior shell of the cylinder

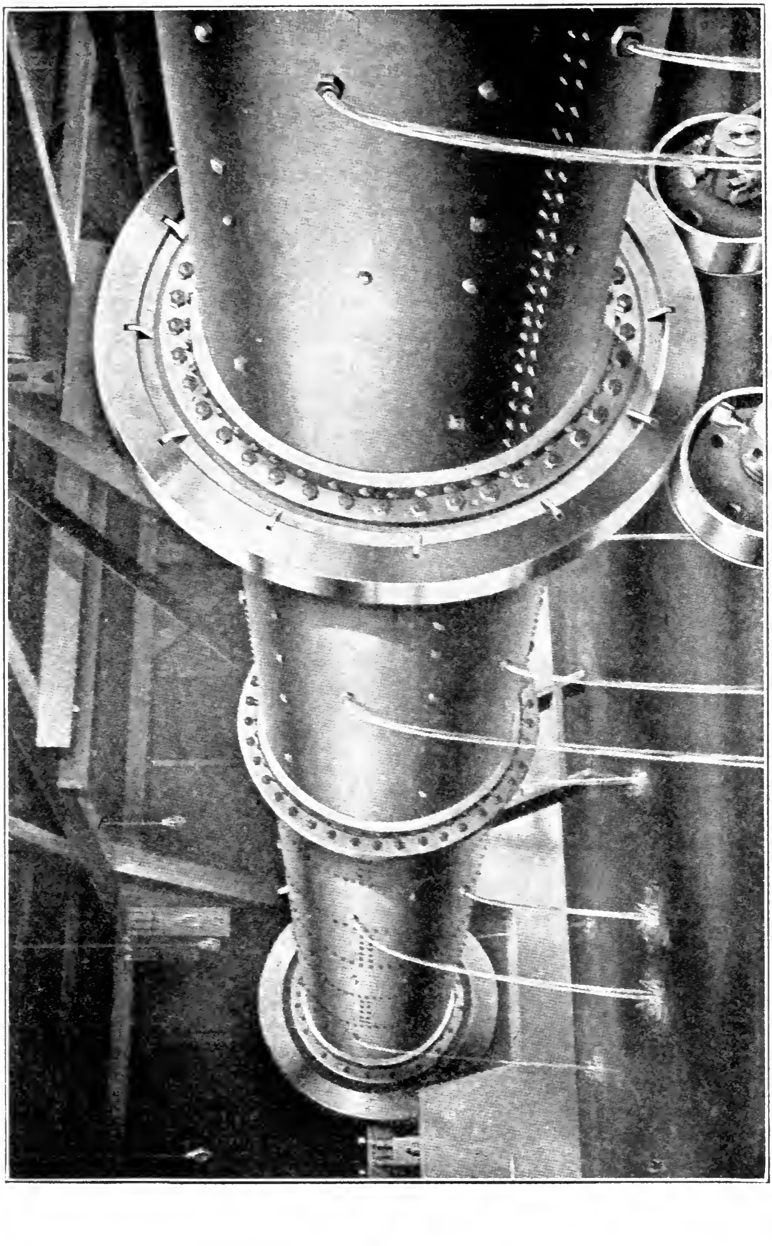

Fig. 30. — The Burt Revolving Filter.

is prepared as a leaf-filtering surface. The required amount of slime pulp for a charge is delivered to the interior of the filter through a valve at the point where the feed is delivered to a tube mill. After the charge of slime has been admitted, air under a pressure of 25 to 45 pounds is turned in. The liquid slime pulp remains constantly in the bottom of the filter throughout its length as it revolves. The air pressure causes the solution to pass through the filter cloth and out through holes in the shell to a sump over which the filter revolves. The pulp gradually collects as a shell on the filter surface. When the cake has been made, as is indicated by air coming out of the solution discharges, the wash solution is admitted and kept under air pressure. As the tube is constantly revolving this results in a very good washing or displacement of the original gold solution in the pulp. After final air displacement the entire end of the filter is opened by a quick-acting device. The removal of the air pressure causes the cake to fall to the bottom of the cylinder, and the addition of a little water together with the revolving of the filter causes the washed pulp to slide out the discharge end; when the end gate is closed and a new charge started. The advantages claimed are that it will handle very sandy pulp, requires no excess pulp or solution to be returned, gives a very efficient wash through its method of making and washing a cake, and requires but little repairs on the filtering medium.

To tabulate these illustrations of the different types of filters:

Stationary charge, intermittent, vacuum. Butters.
Movable charge, intermittent, vacuum. Moore.
Movable charge, vertically revolving, continuous, vacuum. Oliver.
Stationary charge, horizontally revolving, continuous, vacuum. Hunt.
Movable charge, horizontally revolving, continuous-intermittent, vacuum. Ridgeway.
Movable charge, intermittent, pressure. Kelly.
Stationary charge, intermittent, pressure. Burt rapid filter.
Movable charge, vertically revolving, intermittent, pressure. Burt revolving filter.

The leaf filter in many cases is more efficient than the plate and frame filter press, and is much cheaper to operate, excepting the Merrill press, which when used with the " center-washing " system may be considered as another type of the leaf filter. Leaf filters are especially efficacious in handling a clayey slime that the plate and frame press cannot wash or only with difficulty, and by the " center-washing " process. This is due mainly to the inability to wash such a slime cake 2 or 3 inches thick, whether made in the plate and frame press or in the leaf filter. However, the leaf filters are far from being perfect and able to handle all classes of material, consequently while they are in most cases the best device available, the particular one to be used should be selected with great care, bearing in mind its limitations and the conditions with which it must cope.

One of the first troubles encountered is the necessity of having a slime that contains a considerable amount of granular material to give porosity to the cake and enable a thick and easily-washed charge or cake to be made. This is a condition which is not hard to meet, but the necessity of this granular material being extremely fine may work a hardship where all of the ore is being treated as a slime, in requiring the ore to be crushed far beyond the economic point that will allow a high dissolution of the precious metals. As the amount of sand or granular material increases, the slime becomes less plastic and more permeable by the wash solution, consequently with a true slime a cake of only ¼ to ½ inch may be possible, but as the amount of sand increases, a cake up to 2 inches and even more may be made. Besides the increase in capacity that the sandier charge gives, there is invariably a quicker and better washing. But an increase in the amount of sand or its coarseness increases the inability to make a good cake and give all the pulp a good washing, for as the pulp supplied to the filters becomes more dilute and the sand becomes larger in quantity and coarser, the pulp classifies more in the making of a cake through the settling of the sand. This introduces difficulties into the making and washing of a cake and removing the pulp, even though the principle of the cakes being built up with an even permeability works admirably. That coarse sand which settles in the corners and bottoms of the filter tanks especially gives trouble. It has been attempted, without entire success, to stop this classifying in the stationary vacuum type by pumping the thicker sand settling to the bottom of the filter to the top as the cake forms. With the movable vacuum type the pulp is kept in agitation usually by air, causing the formation of considerable carbonate of lime which closes the pores of the filter cloths. In both cases there is trouble with pulp containing a large amount of coarse sand. The pressure filter is able to handle this material much better than the vacuum or atmospheric-pressure filter, for by reason of the higher pressure used (up to 80 pounds) as against the atmospheric pressure (up to 15 pounds) a cake can be made in the pressure filter in 10 minutes, including filling and emptying, that will require from 40 minutes to 1½ hours to make with the vacuum filter. The higher the specific gravity of the pulp, the more viscous and dense it is, and the larger the amount of dry pulp in it, the better it can be worked, for the larger the amount of dry pulp and true slime the better it will hold the slime and sand in suspension and prevent classifying. When containing but little sand and that very fine, a dilute pulp may be used, but as the sand becomes greater in quantity and coarser the pulp must be thickened. The advantage of the pressure filter over the vacuum type in making quick charges on this class of material — pulp containing much sand — is pronounced.

The pressure filters have the disadvantage of small capacity per charge which they overcome to some extent by the rapidity with which the charges are made, but this requires that much additional attention. The intermittent-vacuum filters can be built with enormous capacities, which reduce the amount of labor required per ton treated considerably over the pressure type. The movable-pressure filter has the advantage over the stationary type, of each charge being exposed to view after the washing, so that it is possible to know just how the operations are proceeding and take steps to overcome any acute difficulties, as well as allow repairs to be easily made. It probably gives the most efficient washing of all the types of filters, but requires more manipulation than the others. Intermittent-pressure filters can be installed in small plants cheaper than the intermittent-vacuum filters, but are not so suitable in first cost and operating expenses for large plants.

The Ridgeway, in which the charge revolving horizontally is immersed in the wash solution, appears to give the most efficient wash of the continuous filters, but has the disadvantage of low capacity and extreme delicateness. Those filters, whether continuous or otherwise, that wash by means of a spray cannot be considered as such efficient washers as those in which the cake is washed by submersion, but have the advantage of simplicity, requiring little attention, and simplifying the operations generally. The Oliver, the revolving-drum filter, appears to be well adapted and working largely at present on filtering a true slime, a material that other filters have not yet satisfactorily handled. The Hunt and similar filters, as horizontally-revolving mechanisms treating sand or slime, would appear to be well adapted to treating a pulp containing a large amount of coarse granular material. Apparently coarse sand is best treated by a mechanism in which gravity assists in holding the sand at the point where it is attached to the filter, as on the bed of a horizontal filter. Continuous filters have the advantage of requiring little attention, whereas all intermittent ones require continuous attendance. Like pressure filters their small capacity well adapts them for small plants, but not so well for large plants. Another advantage of the continuous filters is that the costly item of returning the surplus pulp from the stationary-vacuum filters, moving the filter basket in the movable-vacuum type with its complicated machinery and wear and chance for breakage, or the pumping against pressure in the pressure type, is avoided in the continuous filter which takes by gravity the feed of pulp at a constant rate. A great advantage of the continuous over the intermittent system, in which the surplus wash solution is returned to its stock tank, is that there is no " building up " in the value of the solution. When the surplus wash solution is returned from a filter tank, it is richer than when it entered, through its contact with the rich solution of the unwashed pulp as found in the pipes, the bottom and sides of the filter tank, and in the cakes themselves. Likewise in wash-solution tanks after the movable leaves have been introduced and removed, though undoubtedly not to the extent where the unwashed pulp and the wash solution enter and are withdrawn from the same box or tank. The value in the wash solution builds up rapidly in this way and would soon approach that of the solution in the slime cakes as first formed, were it not that this solution is sent to the zinc boxes as soon as its value mounts to a certain figure. This is one of the principal weaknesses of the leaf-filter process and is a source of great mechanical loss of the dissolved precious metals. Some plants keep down this loss by constantly washing with barren solution, returning the surplus to the crushing department or elsewhere, but this involves the precipitation of a large amount of solution and generally results in a high mechanical and other loss of cyanide, through the impossibility of segregating a low-strength solution for filter washing and for crushing.

The filter cloths of all filters using the leaf filter or "center-washing" system are subjected to being encrusted, coated, or clotted by a carbonate of lime, much of which results from the action of air used in agitation upon the lime in solution, and in some silver plants has caused the protective alkalinity to be kept lower than otherwise desired. This coating, in spots and generally, reduces the permeability of the cloth or increases its resistance to the flow of solution through it, so that a thinner cake is formed than usual and the capacity is reduced. The deposit of lime is removed by immersing the leaves in a ½ to 2 per cent solution of hydrochloric acid, which removes the lime as a chloride. The item for the removal of the lime by treating the leaves is quite large with some types of filters. With the pressure and plate and frame types the acid wash may be pumped through the press, but with the other types the leaves must be removed or other means employed, involving considerable labor.

The type of filter to be installed is a matter of personal opinion. All have their good and bad features, each is well adapted for certain conditions and poorly for others. That none of them is perfect or is the acme of what is to be desired, is a matter of universal knowledge, and is indicated by the number of prominent plants that use their modern filters as dewaterers, without attempting to give the pulp any wash.

CHAPTER XII

PRECIPITATION

The cyanide solution carrying gold and silver, in which condition it is often called "pregnant" or "gold" solution, having been removed from the ore by draining the percolation charge if a sand or leaching plant, by decanting the clear supernatant solution from the settled slime if a decantation plant, or by forcing the solution from the pulp through a filter cloth or other medium if the slime pulp is finally filtered, is conveyed to the gold tanks to be supplied to the precipitating department as needed. This solution after precipitation is called "barren" solution. Outside of a few plants where electrical precipitation is used on silver ores, and which appears to be assisted by zinc boxes where a close precipitation of the gold is required, zinc is universally used as a precipitant, either in the form of threads or shavings or as a fume or dust.

Reactions in Zinc Precipitation and Formation of White Precipitate. — The precipitation of the precious metals may be said to be due to the replacement of the gold and silver in the double cyanide by the zinc and to electric currents set up by the chemical reactions which electrochemically deposit the precious metals from the solution. Other theories have been advanced and undoubtedly have some weight, but the above is both more acceptable and more illustrative of the principles and practice of precipitation.

The precipitation of gold and similarly of silver in the presence of free cyanide may be expressed in the equation:

$$KAu(CN)_2 + 2\,KCN + Zn + H_2O = K_2Zn(CN)_4 + Au + H + KOH.$$

In the absence of free cyanide as:

$$KAu(CN)_2 + Zn + H_2O = Zn(CN)_2 + Au + H + KOH.$$

The following reaction may take place in the absence of any metal to be precipitated:

$$Zn + 4\,KCN + 2\,H_2O = K_2Zn(CN)_4 + 2\,H + 2\,KOH.$$

Alkalis act upon zinc to form an alkaline zincate, as:

$$Zn + 2\,KOH = Zn(OK)_2 + 2\,H.$$

While the alkaline zincate may be dissolved in free cyanide, as:

$$Zn(OK)_2 + 4\,KCN + 2\,H_2O = K_2Zn(CN)_4 + 4\,KOH.$$

Zinc oxide (ZnO) formed through exposure of the zinc to the atmosphere, especially when the zinc is moist, may be changed to zinc hydroxide ($Zn(OH)_2$), or the zinc hydroxide may be formed directly, as:

$$\begin{cases} ZnO + H_2O = Zn(OH)_2. \\ Zn + 2\,H_2O = Zn(OH)_2 + H_2. \end{cases}$$

Zinc oxide or hydroxide may be formed in a similar way by the oxidizing effect of a solution without free cyanide or alkali to combine with the zinc. The oxide or hydroxide may be changed to the zincate by an alkali, as:

$$\begin{cases} ZnO + 2\,KOH = Zn(OK)_2 + H_2O. \\ Zn(OH)_2 + 2\,KOH = Zn(OK)_2 + 2\,H_2O. \end{cases}$$

In the presence of free cyanide the oxide or hydroxide may be changed to the potassium zinc cyanide, as:

$$(ZnO + H_2O) + 4\,KCN = K_2Zn(CN)_4 + 2\,KOH.$$

The law of mass action undoubtedly prevails to make some of the above reactions reversible.

The tendency is strong for the double cyanide of zinc, the zinc potassium cyanide ($K_2Zn(CN)_4$), to disassociate in a weak cyanide solution into the simple cyanides, as:

$$K_2Zn(CN)_4 = 2\,KCN + Zn(CN)_2.$$

Zinc cyanide ($Zn(CN)_2$) is insoluble in water, hence in very dilute cyanide solutions it precipitates to form the white precipitate of the zinc boxes. Zinc cyanide is dissolved and reacted upon by alkali, as:

$$\begin{cases} Zn(CN)_2 + 4\,KOH = 2\,KCN + Zn(OK)_2 + 2\,H_2O. \\ 2\,Zn(CN)_2 + 4\,KOH = K_2Zn(CN)_4 + Zn(OK)_2 + 2\,H_2O. \end{cases}$$

As both the zinc potassium cyanide and the potassium zincate ($Zn(OK)_2$) are soluble in water and still more so in alkaline solutions, the white precipitate appears to a slight extent only in strongly alkaline solutions, even though weak in cyanide.

The dissolved zinc or the zinc oxide eventually forming zinc hydroxide ($Zn(OH)_2$) is acted upon by free cyanide, as:

$$Zn(OH)_2 + 4\,KCN = K_2Zn(CN)_4 + 2\,KOH.$$

Thus the zinc oxide or hydroxide which is insoluble in water and, in consequence, in a very weak cyanide solution will precipitate to form the white precipitate of the zinc boxes; in a solution strong in cyanide will form the soluble double potassium or other alkaline zinc cyanide; or in a solution strong in alkali will form the soluble potassium or other alkaline zincate.

Clarifying the Solution. — If there is a tendency for the solution to leave the sand vats or agitation tanks carrying considerable suspended matter or slime which would interfere with precipitation, the solution may be filtered before or after the gold tanks by means of sand filters. These consist of boxes or small tanks with filter bottoms similar to those of leaching vats, except that the filter cloth is very porous, usually coarse burlap. This is covered with about twelve inches of coarse sand. The solution filtering through leaves its slime covering the sand, where it may be periodically scraped off. Or the solution may be introduced into the bottom of the gold tank by means of a pipe or baffle board, and syphoned off the top for precipitation in a much clearer and more settled state. Plate and frame filter presses have also been used for clarifying purposes.

Zinc Boxes. — Zinc boxes for holding the zinc shavings are arranged that the solution may in all cases flow upward through the shavings. This gives better results than a downward flow for several reasons: It permits an easier, gentler, and better-distributed movement of the solution as it rises upward through the zinc. It allows the partly consumed and better precipitating zinc in the bottom of each compartment to come in contact with the solution before the newer zinc, giving more effective precipitation, greater economy in zinc, and less trouble. It causes less disturbance in dressing the boxes of the fine gold and silver slime adherent or fallen from the zinc. The upward movement assists the hydrogen bubbles formed to naturally rise and become liberated instead of adhering to the zinc, coating and fouling it against precipitation.

In the bottom of each compartment is placed a screen or false bottom from three to six inches above the floor or bottom of the zinc box. This screen may vary from 4 to 12-mesh. It serves to hold the zinc some distance from the bottom of the box that the solution may easily reach the entire lower area of the zinc shavings to rise evenly throughout the mass, and that the space below the screen may act as a retainer for the fine gold and silver slime falling off the zinc and passing through the screen. The first compartment of the box is often used as a settler to assist in clarifying the solution by having no zinc placed in it; while the last compartment may be used in a similar way to prevent fine gold-silver slime from being carried away when the box is disturbed or the flow is too great. Often a filter of coarse sand, sawdust, oakum, fiber packing, or coarse filter cloth is used in these end compartments. The filtering medium in the last compartment is eventually added to the zinc slime melted to get any gold-silver slime which it may have caught.

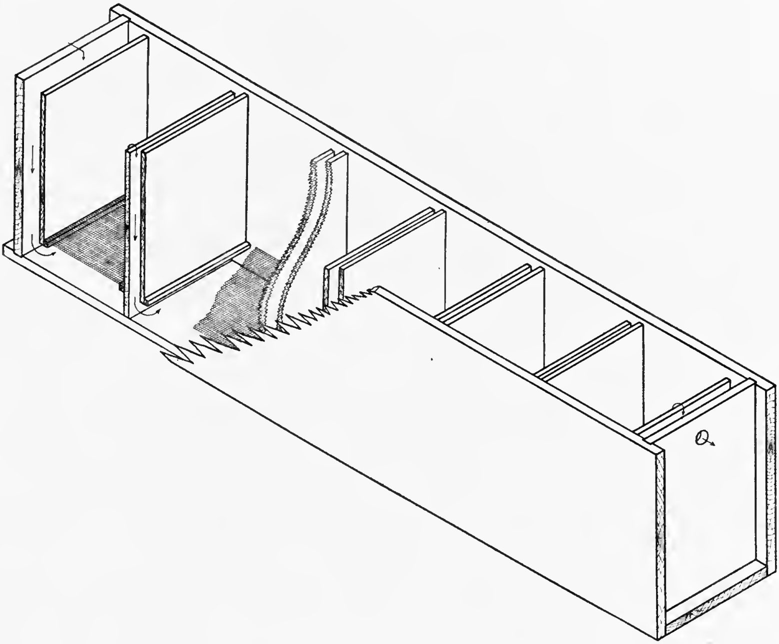

Fig. 31. — Zinc Box.

Zinc boxes vary considerably in size. The larger sizes should always be used to secure economy of space and care. A good size for a large box would be to consist of eight compartments for upward flow, each 21 inches long (in direction of flow), 27 inches wide (across box), and 33 inches deep above the screens. Each compartment of this box will hold approximately 10 cubic feet of zinc shaving. The six compartments, allowing the end ones for settling purposes, will contain a total of 60 cubic feet. The majority of zinc boxes are built with their compartments in the form of a perfect cube or nearly so.

Size of Shavings. — The important point in zinc precipitation is the necessity of exposing a large area of zinc to the solution. Consequently to get the highest efficiency from the zinc, it is prepared as a fine dust or in shaving or threads $\frac{1}{32}$ to $\frac{1}{8}$ inch wide and from $\frac{1}{400}$ to $\frac{1}{1800}$ inch in thickness. A pound of shavings cut with a thickness of $\frac{1}{1500}$ inch will expose about 80 square feet of zinc surface, and when cut with a thickness of $\frac{1}{750}$ inch will give about 40 square feet of zinc surface. This will indicate why equal weights of the finer-cut shaving will precipitate better than those cut coarser. Some idea of how zinc becomes effective through fine division can be seen when it is said that a pound of zinc equals 3.854 cubic inches, and when cut into shaving and placed in the boxes at the rate of 6 to 8½ pounds per cubic foot, the solid metal amounts to only 1.3 to 1.9 per cent of the space actually occupied by the zinc shavings. How fine a shaving should be used must be determined by actual practice. The coarsest shaving that will give a satisfactory precipitation should be employed, as the coarser the shaving the slower it will be to break up into short zinc which entails a higher mechanical loss when gathered into the clean-up and greater trouble in operating and dressing the boxes. A thickness of $\frac{1}{1500}$ inch is generally used in gold plants, and from $\frac{1}{750}$ to $\frac{1}{1500}$ inch in silver plants.

Weight of Shaving and Amount Required. — The weight of a cubic foot of finely-cut zinc shaving when packed in the boxes, in the customary manner in gold plants, will vary from 6 to 8½ pounds, depending upon their thickness and how snugly and tightly they are packed in. With the coarser-cut shaving a greater weight of zinc may be packed in a cubic foot, both because the greater thickness of the threads gives a greater weight of zinc in comparison to the voids, and because the thicker and stronger threads or shaving may be more tightly packed in the boxes without being easily broken and channeled by the solution. Silver plants using coarser shaving pack 12 to 13 pounds of zinc per cubic foot. It is reported that as high as 18 to 23 pounds of zinc per cubic foot have been packed in the boxes in an experimental way.

In gold plants it is customary to allow 1 cubic foot of zinc-box space for each ton of solution to be precipitated in 24 hours. The rate of flow in the average gold plant is probably a little higher than this; in some it reaches over 2 tons per 24 hours for each cubic foot of zinc shaving. In silver plants the rate of flow will vary from 1 to 6 tons of solution per 24 hours for each cubic foot of zinc shaving; probably $1\frac{1}{2}$ to 3 tons would represent the average practice. The higher rate of flow in silver plants is due to the stronger solution used causing more effective precipitation, and to the fact that the efficiency of the precipitation, as referring to the weight of precious metal still remaining unprecipitated in the solution, need not be as great as in gold precipitation. A flow of 1 ton per cubic foot of zinc per 24 hours gives a contact between the solution and the shaving of about 45 minutes.

The number of compartments to be filled with zinc will depend upon the tonnage put through the boxes and the effectiveness of the precipitation. No more zinc should be kept in the boxes than is necessary to secure good precipitation, or zinc will be consumed unnecessarily. It hardly appears necessary to have more than one or possibly two lower compartments, in which the new zinc is added, containing bright zinc. If the zinc in too many of the lower compartments remains bright and assays of solution taken from the different compartments show that the precious metals are all precipitated in the upper compartments, either some of the lower compartments should be left empty or the flow should be increased to the point where the filled compartments are working at a proper efficiency. In short, under normal conditions it is the rate of flow per cubic foot of zinc that must be kept constant. If the flow through the boxes is increased, the number of cubic feet of zinc must be increased. If the flow is decreased, the number of cubic feet of zinc may be lessened. It is also necessary in studying conditions or making comparisons to remember that under equal conditions the effectiveness of the precipitation depends upon the area of zinc exposed per ton or unit of solution, and that the thickness of the shaving and the quantity or weight per cubic foot as packed are just as important factors in giving the zinc area as the number of cubic feet of packed shaving used.

Packing and Dressing the Boxes. — It is necessary in packing the boxes with zinc shaving to take the greatest care to prevent channeling and an uneven flow through the zinc. The use of a number of compartments, through each of which the solution must flow, reduces the danger of poor precipitation from this cause. Taking the case of a clean-up of the gold-silver slime being made to illustrate the method of caring for the boxes, after the compartment has been washed clean of all slime through a hole in the bottom emptying into a launder, the discharge is closed and the screen put in place. New zinc is now taken, preferably that cut a little coarser than as ordinarily used. This is fluffed by gently pulling apart and untangling the bunches, especially the more compact ones, and is placed evenly over the screen to the depth of a few inches. On top of this is spread an even layer of the short zinc being returned to the boxes, then a layer of the long zinc being returned, to be followed by a layer of the short zinc, and so on until the compartment is filled. The purpose of the coarse new zinc is to act as a screen as long as it is able to withstand the dissolving action, to diffuse the solution passing up through it, and to utilize the short zinc even after it is well eaten up by catching and holding it, instead of allowing it to drop and pass through or clog the screen. The compartments being used are packed in this manner; the short zinc and that already much acted upon are concentrated in the first two or three compartments. After the short zinc is utilized, the already-acted-upon long zinc is used, and finally whatever new zinc is needed is placed in the lower compartments. The fluffing of the zinc and its arrangement together with the placing of the layers of short zinc must be studied with a view to preventing channeling and an uneven flow.

The boxes should be dressed as frequently as needed, which is usually every second or third day. The necessity of dressing and how often it should be done can be studied from three points: the increase in the assays of the solution leaving the box, the amount of zinc consumed and extent of channeling, and the gradual creeping down of the blackening and discoloration of the bright new zinc in the lower compartment or compartments by the gold and silver precipitated. Small lots of new shaving may be placed in the lower compartments to experimentally observe the progress of discoloration.

In dressing the boxes between clean-ups, the flow is first turned off and the operator begins work, using a pair of rubber gloves or with hands and arms greased with vaseline or other harmless grease compound to keep the skin from becoming rough, irritated, and sore by contact with the solution. He either moves the zinc from the lower compartments to the upper ones, where practically all of the consumption has taken place, adding the new zinc to the lower compartments, or adds new zinc to whatever compartments are in need without moving any of the old zinc. It is preferable to move the zinc toward the head compartments and add the new zinc at the foot of the box, as zinc that has already been acted upon is a more active precipitant than new zinc, and by moving it toward the head of the boxes it causes a greater precipitation in these head compartments and collects that occurring in the balance of the box into them. Moving the zinc up gives a better precipitation, a higher-grade slime, and a concentration of the gold and silver into a smaller area of the box, making the clean-up less bulky and leaving less gold and silver in the box. In moving the zinc up, the well-acted-upon and rotten zinc should be disturbed as little as possible. Most of the precipitation can be effected and collected in the first two or three compartments. These should be disturbed as little as possible, the operator confining himself to closing up any channels or open edges or corners, and placing the necessary zinc taken from the third or fourth and lower compartments. No zinc should be removed from any compartment in which such an amount of precipitation takes place that the shaving becomes rotten and tends to break up in handling, as it causes a greater mechanical loss and disturbs the gold-silver slime, rendering it more liable to be carried out of the box. The zinc should be spread out and laid down in layers, which, being at right angles to the movement of the solution, conduce to allow an even flow and contact. The zinc should be well packed into the edges and corners of the compartments, as the greatest channeling occurs at these places. A little experience and study will soon indicate the best procedure. The practice of beating the zinc tightly into the compartments with a stick is inadvisable, since it may pack the zinc too tightly for a uniform flow. Yet the zinc should be firmly and carefully and not loosely packed in place. In some cases the zinc when being cut has been folded into skeins or hanks the length of the compartments. The skeins are packed into the compartments in layers at right angles to each other, new zinc being added as needed without disturbing the old zinc. This method tends to give a more uniform flow with less channeling.

General Care of Precipitation. — The boxes may be allowed to stand for 15 minutes after dressing before starting the flow through, for the purpose of settling any disturbed zinc slime. The highest rate of flow can only be determined by assaying the solution as it leaves the box and observing the progress of discoloration of the bright zinc in the lower compartments, but must not be so rapid as to carry the fine zinc slime out. It must be remembered in determining this that the best precipitation takes place after the boxes have been newly dressed and more especially after a clean-up, when the zinc is clean, active, and well-arranged; while it reaches its poorest just before dressing and cleaning-up. The maximum efficiency of precipitation takes place when an equal volume of solution passes by each part of the zinc, which is when the zinc is newly arranged. This efficiency rapidly lessens, for channeling quickly forms whereby the solution is not brought evenly in contact with all the zinc, which must be met by rearranging the zinc through a new dressing of the box. Solution strong in cyanide keeps the zinc clean and active and gives a good precipitation, consequently it can be run through the boxes much faster than weak solution.

One of the evidences of good precipitation is the rising to the surface of the bubbles of hydrogen which are generated during the precipitation, as is shown in the equation representing the process of precipitation. Too many bubbles given off are undesirable, for it indicates a large consumption of zinc. Strong cyanide solutions cause a large evolution of hydrogen through the excessive dissolution of the zinc. Too high a protective alkalinity will act similarly, and besides consuming an unnecessary amount of zinc may give rise to excessive hydrogen which may cling to and lift the zinc out of the boxes or which may polarize it against effective precipitation. Where the zinc has been fouled in this way, a low strength of solution may be run through rather fast, followed by shaking and rearranging the zinc. An excess of lime or alkali may cause a deposit of lime salts or other compounds as a species of white precipitate. Where slime from the ore has been carried into the boxes, or lime salts and other compounds have formed loose deposits or incrustations, — such deposits as cannot be dissolved and washed out by solutions strong in cyanide and alkalinity, — and good precipitation cannot be secured, the boxes should be cleaned up. In some cases the incrustations have been removed by dipping the shavings in dilute acid to dissolve the deposit, followed by rinsing the shavings in water.

Trouble is often experienced in precipitating from weak solutions. It is impossible to predict how weak a cyanide solution can be successfully precipitated, for mill solutions showing no free cyanide have been precipitated to a trace of gold and silver. Solutions weak in cyanide precipitate better when carrying considerable protective alkalinity and the cyanogen compounds that accumulate in a solution when working a clean ore. But such solutions precipitate poorly when without or low in protective alkali or when fouled through the large accumulation of the compounds that enter a solution when treating an acid or base ore. The cause and remedying of poor precipitation from weak solutions, and without increasing the strength of solution, is in some cases a difficult problem to solve. The effectiveness of the precipitation is mainly a matter of keeping the zinc clean and free from everything except metallic zinc and the metals precipitated. More especially in keeping the zinc free from the zinc oxide, hydroxide, and cyanide insoluble in water and poorly soluble in weak solution, and which tend to form by contact of zinc and water or weak solution. An alkali will be efficient in doing this, just as cyanide, by promoting a vigorous chemical action which dissolves and removes the fouling compounds, which dissolves the zinc for replacing the precious metals, and which sets up the electric currents to deposit or further assist in the deposition of the gold and silver.

White Precipitate. — One cause of poor precipitation has been the formation of a white precipitate in the zinc boxes. In a few cases this precipitate has been lime salts, due to the excessive use of lime or other alkaline neutralizer, but it is generally zinc as a hydrate ($Zn(OH)_2$), a cyanide ($Zn(CN)_2$), or a zinc potassium ferrocyanide ($K_2Zn_3(Fe(CN)_6)_2$), with some salts of the bases. This precipitate is an inert, grayish-white, and somewhat granular substance which usually forms in the weak-solution zinc boxes or when very dilute solution is being run through a box. It also forms more rapidly when treating pyritic ores or those containing considerable acidity, especially when there is no protective alkalinity in the solution. It forms in the upper compartments of the boxes and greatly hinders precipitation by coating the zinc and matting and caking it together. A consideration of the equations shown before in this chapter and what has been said under Alkalinity and Lime regarding the reactions in the zinc boxes show that the white precipitate is due to too low a strength in cyanide and alkalinity of the solution passing through the box. When solutions are low in cyanide and alkalinity the zinc potassium cyanide, which may be considered to normally exist, tends to disassociate with the formation of zinc cyanide, while considerable zinc hydroxide forms from the action of the weak solution on the zinc. Zinc cyanide and zinc hydroxide are insoluble in water and naturally precipitate in weak solutions, but are soluble in solutions of some strength in cyanide and alkali. Consequently the solution should be kept up to a sufficient strength in cyanide and alkali to prevent the white precipitate from forming, or if formed a strong cyanide solution which may have considerable alkalinity should be run through the boxes to dissolve and carry out the precipitate, which will take some time. In some cases the zinc has been taken out of the boxes and washed in strong caustic soda solution, but this course is inadvisable, as increasing the cyanide strength and alkalinity will generally remedy the trouble, except where the zinc and zinc slime are caked with a large amount of the white precipitate, which may necessitate disintegrating the cakes and lumps by a species of clean-up. The zinc potassium ferrocyanide, which probably forms to a greater extent in the working of decomposed pyritic ores, is less soluble in alkaline solutions than the zinc hydrate and zinc cyanide, and when once formed may require a clean-up to remove. For the purpose of keeping the boxes clean and active and the quantity of white precipitate small, the solution flows entering the strong and weak-solution boxes may be made interchangeable, the flows to be alternated as may be necessary; or no division is made in the solution to create one of high and another of low strength, but all is kept at a strength that will give good precipitation. The strength of weak solution is sometimes increased by a drip of strong solution into the head of the box, or occasionally adding a chunk of solid cyanide there, or by standardizing or adding cyanide or lime or caustic soda to the contents of the gold tank.

Zinc-Lead Couple. — The zinc-lead couple has been used to a large extent to secure satisfactory precipitation from weak solutions. It is prepared by dipping the new shavings into a 5 to 10 per cent solution of lead acetate ($Pb(C_2H_3O_2)_2.3\ H_2O$) for a few minutes until they turn black from the lead deposited, when they are placed in the compartments as usual. In some cases a 1 per cent solution of lead acetate has been allowed to drip into the head of the boxes, though dipping appears to be the better. The zinc and lead together form a galvanic couple which greatly assists precipitation.

Copper in Solution. — Copper in solution tends to precipitate when the solution is low in cyanide, and to remain in solution when the cyanide strength is high and interfere with the dissolution and precipitation of the precious metals. Small quantities of copper in weak solutions are precipitated upon the bright new zinc in the lower compartments. When the amount of copper is small, this does not interfere with the precipitation, in fact it usually assists through the formation of a galvanic couple, but large quantities may interfere by giving the zinc too thorough a coating of copper or, when remaining in solution, may prevent precipitation of the gold and silver. The principal means of removing the copper have been the use of lead shavings in the head compartments or the zinc shavings dipped in lead acetate and placed in the lower compartments, lead or the zinc-lead couple being a good precipitant of copper. Attempts have been made to keep the amount of copper dissolved from the ore low by the use of weak dissolving solutions, and then to keep the copper in solution by raising the strength of the solution before entering the zinc boxes, but the best method appears to be to precipitate the copper from the solution in some convenient way and get it out of the system.

Mercury in Solution. — Mercury found in old tailing from amalgamation mills is usually changed to an oxide, salt, or other compound which is readily attacked by cyanide solution. Mercury in its metallic state is slowly dissolved by cyanide solution, the dissolved metal in any case being precipitated in the zinc boxes if not before. A small amount of mercury dissolved in this manner is not harmful but very beneficial, for it is the best substance for removing alkaline sulphides by forming an insoluble mercuric sulphide (HgS) with the sulphur. Compounds of mercury, principally mercuric chloride, have sometimes been used for this purpose with excellent results. Small quantities of mercury on the shavings may increase precipitation through the formation of a mercury-zinc galvanic couple, but large quantities are detrimental through causing the shavings to break up and slime. Where the zinc slime contains a large amount of mercury, it may be retorted to secure the mercury before being treated for its gold and silver.

Cutting of Zinc Shavings. — Care must be exercised in cutting zinc shavings or in purchasing shavings already cut, as zinc oxidizes easily when heated. With careless cutting the zinc heats rapidly and is often further assisted to oxidize through being cooled by cold water poured over it. Such partly-oxidized zinc is not highly efficient in precipitating and unduly breaks into short zinc. Zinc exposed to the atmosphere slowly oxidizes, consequently fresh-cut zinc is the best. Zinc containing a small amount of lead is an excellent precipitant as it is a zinc-lead galvanic couple without further treatment; in fact, most zinc shavings contain a small amount of impurities which are beneficial, and zinc dust containing a small percentage of lead is sometimes ordered.

Mechanical and Chemical Consumption of Zinc. — Zinc is consumed in the precipitating process in two ways — chemically and mechanically. The zinc chemically consumed by the reactions in the zinc boxes goes out of the plant in the solution discharged with the tailing residue and lost by leakage, by being precipitated in the zinc boxes as spoken of in connection with the white precipitate, and mainly by being precipitated in the ore. The nature of the precipitation occurring in the ore is not well known, but is supposed to be with alkaline sulphides to form the insoluble zinc sulphide, to be precipitated as a zinc carbonate, or the precipitation may take place in the ore under conditions similar to those by which the white precipitate is formed in the zinc boxes. It is an interesting fact that the amount of zinc in a plant solution remains fairly constant though the solution be used for years. The zinc in solution is generally considered to exist as a zinc potassium cyanide ($K_2Zn(CN)_4$), more especially when the difference between the free and total cyanide indicates enough cyanogen to be combined with the zinc, which by the formula would be 1 pound cyanogen in terms of potassium cyanide combined with .251 pound of zinc. It has been observed that as the zinc potassium cyanide is apparently regenerated into free cyanide, by increasing the protective alkalinity or as the free and total cyanide approach each other, the amount of zinc in solution falls, but exactly how it is removed under these conditions is unknown.

The zinc consumed mechanically in the precipitating process is that removed from the boxes with the gold-silver slime to be treated and melted into bullion. The amount of zinc consumed in this way is high in a gold plant, since much short zinc is removed from the boxes to be acid treated, etc., in the effort to get all the bullion possible. In a silver plant the amount is much less since a close clean-up, as referring to the comparative weight of precious metal left in the boxes, is not made. The stronger the solution used, the greater will be the chemical consumption. The weaker the solution, the greater will be the mechanical loss, for the use of weak solutions is attended with the production of much short zinc and other zinc-box troubles, especially when the solution is also low in gold and silver. The precipitation of rich solutions is attended with a much less consumption of zinc mechanically and often chemically than that of low-grade solutions.

The consumption of zinc is usually reported on the basis of the tons of ore treated. For technical purposes it is also desirable to report it on the basis of tons of solution precipitated and ounces of bullion produced, for ores require widely varying amounts of solution for their treatment; likewise the value of ores and the richness of their solutions vary. The amount of zinc shavings used will vary from $\frac{1}{8}$ to $\frac{1}{2}$ pound per ton of ore treated in a gold plant, and in a silver plant from $\frac{3}{4}$ to $1\frac{1}{2}$ pounds and upward. The consumption will vary from $\frac{1}{8}$ to $\frac{1}{3}$ pound per ton of solution precipitated. On a gold plant the consumption will be from 4 to 20 parts of zinc to 1 part of bullion produced, while in a silver plant $1\frac{1}{3}$ parts and upward will produce 1 part of bullion. Theoretically, in the replacement by zinc of the gold and silver in the potassium gold or silver cyanide ($KAu(CN)_2$ or $KAg(CN)_2$) formed in the dissolving process, 1 part of zinc should precipitate 3 parts of gold or 1.65 parts of silver.

Regeneration of Cyanide and Alkalinity. — There is often an apparent regeneration of free cyanide taking place in the zinc box as shown by titrating the solution entering and leaving the box. This may amount to as much as $\frac{1}{4}$ or $\frac{1}{3}$ pound per ton of solution. The exact cause of the regeneration is indeterminate, but is due to the complex reactions occurring in the zinc box. There is also a slight increase in the alkalinity; this can be more clearly understood than the regeneration of cyanide, by considering the equation representing the precipitating reaction which indicates the formation of caustic potash or a similar alkali.

Zinc-Dust Precipitation. — Zinc dust is a highly satisfactory precipitant of gold and silver, due to its fine state of division, its agitation with the solution, and the forcing of the solution through that already in use. As at first developed the solution to be precipitated was run into tanks holding from 15 tons upward of solution. After a tank was filled with solution it was agitated, usually by air pipes in the bottom of the tank. Fresh zinc dust amounting to about $\frac{1}{4}$ to $\frac{1}{3}$ pound per ton of solution was sprinkled over the charge. This zinc dust together with that already in the tank from previous charges and stirred up by the agitation was sufficient to precipitate the metal with 15 minutes or more agitation, after which the solution was drained through plate and frame filter presses to the barren sump tanks. In the filter press the solution passed through the already-acted-upon zinc which gradually accumulated by being carried in from the agitation tanks.

As at present used the zinc dust is fed continuously to the solution by being spread upon a long, slow-moving belt or by other feeding device — due to the small quantity of dust constantly required and its tendency to agglomerate, it has been hard to secure a satisfactory automatic feeding mechanism. The dust is usually fed together with the solution into a mixing and agitating device wherein the precious metal is largely precipitated. The solution then runs to or is pumped through the precipitation plate and frame filter press. Or the zinc dust is fed to the solution at the intake of the pipe leading to the press. The press is located as far from the intake or mixing device as possible, to secure the better precipitating effect of moving solution and zinc dust. No air is allowed to reach the press as would be the case were it or the intake drained, or the zinc will become much oxidized. The precipitated metal or zinc-gold slime is removed by opening the press and separating the frames, allowing the slime to fall into pans underneath. The filter cloths are scraped clean and returned, or are occasionally burned and added to the meltings. An increased quantity of zinc dust is used when starting anew after a clean-up — at which time any zinc slime in the agitation tanks or mixing device is removed — until a quantity has accumulated in the press. The amount of dust used is increased or decreased as the tailing solution increases or decreases in the amount of gold and silver still unprecipitated. Zinc dust is especially efficacious in precipitating from weak and low-grade solutions. In these cases it is often assisted by the zinc-lead couple formed through

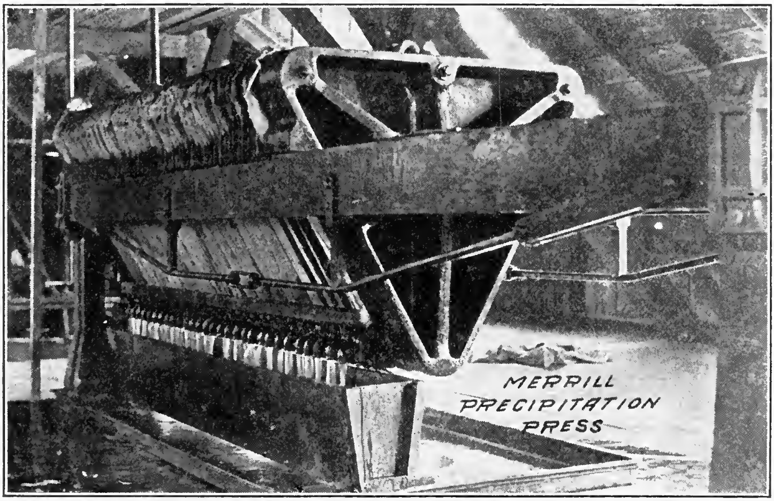

Fig. 32. — Merrill Precipitation Press.

allowing a solution of lead acetate to drip into the mixing device or using dust containing a little lead. The amount of zinc dust used is about equal to the shavings that would be required; in some cases more is necessary, but with careful manipulation less can be employed. The cost of dust is about one-third less than of shavings. The fineness of the precipitate in gold and silver, or the proportion of the precious metals to the base metals or zinc, is about that of slime from zinc shavings washed through a 30-mesh screen, and is subject to being increased by careful manipulation, principally through the cyanide dissolving more of the zinc collecting in the press and thus reducing the zinc content of the precipitate. One of the advantages of using zinc dust is that the entire metal precipitated is obtained each clean-up, and none is left remaining as in the case of using zinc shavings. This indicates that the zinc-dust process is more applicable for gold than for silver plants, since in a gold plant much of the short zinc is collected into the clean-up and considerable value left in the box, whereas in a silver plant only the fine, slimy precipitate is taken, making the cost of refining and the mechanical consumption of zinc less, while less value is left in the box. The zinc-dust process is more adapted to large than to small plants, since the installation cost is high and it requires more continuous attention than the zinc-shaving method. The installation and labor costs are not in proportion to the tonnage, but fall rapidly per ton treated as the plant is increased in size. The installation and working out of a successful zinc-dust precipitation method requires higher ability and closer study than with shaving precipitation, but is capable of being developed to a higher degree of efficiency and economy.

CHAPTER XIII

CLEANING-UP

The precipitated gold and silver are removed from the zinc boxes or presses usually weekly or semi-monthly in a silver plant, and semi-monthly or monthly in a gold plant It is customary to run a strong cyanide solution — .75 per cent (15 pounds) to 1 per cent (20 pounds) — through the boxes for a few hours before the clean-up is started, to loosen the gold-silver slime deposited on the zinc; it also cleans some of the dissolvable white precipitate out of the boxes. The cyanide solution is displaced by running water through the boxes for an hour or longer, that the cyanide may not injure the hands, tend to redissolve the gold, or appear in the cleaned-up precipitate. The operator lifts the zinc from the first compartment into a tub, preferably filled with water to prevent the zinc from oxidizing through exposure to the air. All the zinc is removed from the compartment, also the screen in the bottom. A screen is now suspended in the compartment and the removed zinc washed on it as free from slime as possible, by being disentangled, gently teased, rinsed, and finally being rinsed in a clearer water. Care is taken to break the zinc up as little as possible into short zinc. Three products are made in the process of washing: the slime passing through the screen, the washed " shorts " or " metallics " not passing through the screen and up to two or three inches in length, and the washed long zinc.

After all the zinc taken from the first compartment is washed, the drain plug at the bottom of the compartment is opened to allow the water and the gold-silver slime, including that just washed free from the zinc and that which had previously fallen through the screen, to run through a launder or hose into a sludge or precipitate tank. The compartment is washed out with a little clean water, and the plug and screen replaced. If there is no bottom discharge and drain launder or hose, the contents of the compartment may be allowed to settle, the water syphoned off, and the slime dipped out and carried in pails to the sludge or clean-up tank. Having replaced the screen, a two or three-inch layer of new zinc, preferably cut coarser than regularly, is placed evenly over the screen; on this is spread a two-inch layer of the washed short zinc, then a layer of the washed long zinc, followed by short zinc, and so on until full, as described in connection with the care of zinc boxes. The zinc is kept under water as much as possible at all times to prevent oxidation. The second and succeeding compartments are treated similarly, though in most cases where the zinc has been moved toward the head of the box, not more than the first three compartments need be cleaned out, for in the others an appreciable amount of slime cannot be secured. In some cases the zinc is transferred to a clean-up tank, there to be washed and returned to the compartments.

The slime flowing from the compartments is washed through a screen into the sludge or clean-up tank. The mesh of this screen and that used in washing the zinc in the boxes will vary with how close a clean-up is desired, or how much short zinc is to be included in the clean-up. In a gold plant all zinc not washed through a 10 or 20-mesh screen is usually returned to the boxes, in some cases still coarser zinc is put into the clean-up, though the amount of precious metal in the short zinc is comparatively very small. In a silver plant, owing to the lesser value of the same weight of silver bullion as of gold bullion, only the slime passing through a 30 to 60-mesh screen is generally taken, all the short zinc being returned to the box. This results in a precipitate high in bullion, usually 60 to 75 per cent, and low in zinc. When the quantity of short zinc is too large to be advantageously placed with the long zinc, it may be put in trays suspended in the head compartment, or provisions made for agitating it with rich solution to precipitate the gold and silver in a way similar to zinc dust, being used in this manner until cut or dissolved to a slime. It may be advisable to screen the precipitate to be refined into two classes, that held on a 30 to 60-mesh screen, high in zinc and low in bullion, and that passing the screen, which would be high in bullion and low in zinc, so that separate treatment may be given each.

If the precipitate is not to be acid treated it is pumped into a small plate and frame filter press, where it may be dried by blowing air through before the frame is opened for the removal of the precipitate. Or the precipitate may be run into a small tank with a false bottom similar to that of a leaching vat, the moisture being drawn off and the precipitate partially dried by producing a vacuum underneath the filter cloth. In either case the water drawn from the precipitate is pumped to the head of the zinc boxes to catch any fine precipitate, or run into a tank where it is settled and later used as plant solution, the settlings going into a clean-up.

CHAPTER XIV

ROASTING AND ACID TREATMENT

The precipitate may be refined into bullion in four ways:

Melting.
Roasting and melting.
Acid treatment and melting, with or without roasting.
Smelting with litharge and cupellation.

In melting the precipitate without further treatment, it may be completely dried in a pan set in an oven or over a fire or in a steam-jacketed pan. Or it may be dried as well as possible while in the plate and frame precipitation or clean-up press or in the clean-up vacuum tank, by blowing or drawing air through, followed by mixing with flux and melting without further drying. Melting without further treatment is advisable with a precipitate high in gold and silver and low in zinc, such precipitate as would pass a 40 to 60-mesh screen. As the quantity of metallic zinc in the precipitate increases, the fineness of the bullion will decrease and more zinc will go into the bar. Also more gold and silver will be carried off mechanically in a finely-divided state by the dense fumes arising from the volatilization of the zinc.

Roasting. — Roasting the precipitate before melting converts the zinc into an oxide, so that it can much more easily enter into the slag as a silicate of zinc or dissolved metallic oxide, instead of into the bullion as much of it does when it has not been converted into the oxide. Other base metals and substances are more or less oxidized, decomposed, or volatilized to render the subsequent melting and the slagging off of the foreign substances easier and the grade of the bullion higher. Roasting is especially advisable with the coarser precipitate which must contain much zinc, as a high-grade bullion with but little zinc and bases can be secured in this way from such material. The precipitate to be roasted is placed in heavy cast-iron pans which are put into ovens or roasting furnaces; they are even put over fireplaces and a fire built directly under them. It is hard to say to what extent the roasting should be carried. The better the roasting, the better the zinc and bases will be oxidized to pass into the slag. The roasting is often carried to the point where the zinc takes a dull fire. This is not harmful if the roasting has been carried on slowly so that the fumes and ebullition do not carry off gold and silver, so that the zinc oxidizes instead of volatilizes. Niter, potassium nitrate (KNO_3), to the amount of 3 to 10 per cent is sometimes well mixed with the precipitate before roasting, either as a powder or as a solution saturating the precipitate. This causes a rapid and complete oxidation by converting the zinc into zinc oxide, which being nonvolatile, does not carry finely-divided gold and silver away in the fumes. The farther the roasting is carried, the higher the loss will be, though the use of condensing chambers has shown that the loss is small. During the roasting process the slime should be stirred as little as possible or not at all to avoid loss by dusting.

Acid Treatment. — In the acid treatment the precipitate is settled and dewatered by decantation in the sludge tank into which it runs from the zinc boxes, and acid added to dissolve the zinc. The acids used are sulphuric (H_2SO_4), sulphurous (H_2SO_3), hydrochloric (HCl), and bisulphate of sodium ($NaHSO_4$). Nitric acid (HNO_3) has also been used, but its use is inadvisable as it dissolves more of the precious metals than the other acids. Hydrochloric acid has been used to a slight extent. It has a greater cost and a higher dissolving effect on the precious metals, but has the advantage of forming soluble chlorides with the lime and lead that may be removed by washing. Acid treatment is generally not attempted with silver precipitate, since the short zinc is usually returned to the boxes leaving a slime high in bullion, while the acid tends to dissolve silver and thereby cause a loss. Gold precipitate is usually acid treated, especially if it contains much short zinc, either treating all the precipitate or only that going into the clean-up which does not pass a 40 to 60-mesh. It is seldom profitable to treat the gold slime passing a 40-mesh by acid, except when it is desired to treat coarser material and the facilities do not allow of a segregation of the two classes of material.

Sulphuric Acid Treatment. — Sulphuric acid (H_2SO_4) is generally used for dissolving out the zinc and to some extent the other bases before melting, forming zinc sulphate ($ZnSO_4$) and other sulphates. After the water has been decanted off the settled slime in the sludge or clean-up tank, which, if acid treatment is to be carried out, should be of wood or lead lined, the sulphuric acid is slowly and carefully added to avoid the charge boiling over, in an amount making from a 10 to 20 per cent solution of sulphuric acid. Theoretically one part of zinc requires 1½ parts of sulphuric acid to be converted into zinc sulphate, while a 16 per cent solution of sulphuric acid appears to act to the best advantage on zinc. In actual practice from ¾ to 1¾ pounds of cheap commercial sulphuric acid is used for each pound of dry precipitate, the amount of dry precipitate and its moisture being estimated and the sulphuric acid and any additional water being first added by this estimation, sulphuric acid being finally added according to the continuance of the reaction. On the addition of the acid, there is an active liberation and forcible ebullition of hydrogen as vile-smelling fumes, also some hydrocyanic acid from the decomposition of cyanogen compounds in the precipitate, for which reason the refining tank is covered with a hood and uptake to carry off the fumes. As soon as it is seen that the boiling has subsided to a point where the sludge may be stirred without danger of boiling over, stirring is carried on mechanically or by hand. A sudden tendency to boil over is stopped by the addition of cold water. The heat developed by mixing the acid and water and the chemical reactions is sufficient to cause the active formation of zinc and other sulphates. Sulphuric acid is added from time to time with thorough stirrings until there is no more reaction, showing that the zinc has been dissolved and that the acid is in excess of that required. Stirring should be continued for half an hour after all action has ceased. The "cutting down" of the precipitate by acid may be accomplished within a few hours, but usually an entire shift is allotted to it. The operator must use care not to be overcome by the fumes. Breathing the fumes of ammonia affords relief. Where the ore contains arsenic, fumes of arseniureted hydrogen may be given off which are highly poisonous; several deaths have occurred from this. Preliminary treatment with nitric acid or a general treatment with one part nitric and two parts sulphuric acid to change the arsenic into a nonvolatile arsenic acid has been recommended or treatment made with bisulphate of sodium, though it would appear better to dispense with acid treatment on such precipitate.

The tank is filled with water and thoroughly stirred after the zinc has been cut down by the acid, after which the precipitate is allowed to settle and the acid and sulphate solution decanted off. Three or four or more washes by decantation may be given in this way with water, when the sludge is allowed to run or is pumped into a small plate and frame filter press or a vacuum filter tank as used in the clean-up process, where it is washed free from all sulphates and soluble matter by pumping or drawing water through. The washing by decantation and in the press or tank is usually with cold water, though there is some advantage in using hot water as is often the practice, since one part of water at 1° C. will dissolve .42 part zinc sulphate, at 20 degrees will dissolve .53 part, at 50 degrees will dissolve .67 part, and at 75 degrees will dissolve .80 part. Lead sulphate ($PbSO_4$) is practically insoluble, while one part of calcium sulphate ($CaSO_4$) is soluble in 500 parts of water. Whereas the solubility of calcium chloride is one part in $1\frac{1}{3}$ of water, and of lead chloride ($PbCl$) is one part in 93 parts of water. Which indicates the advantage of using hydrochloric acid when the precipitate contains large quantities of lime or lead. The lead and calcium sulphates besides entering the acid-treated slime, may coat the zinc so that it is not acted upon. A thick, solid deposit of gold and silver, such as sometimes takes place from an extremely rich solution, may prevent the zinc from being acted upon.

The acid washes should be collected in a tank and allowed to settle until the next clean-up, when they are syphoned to waste. Or they may be agitated with scrap zinc before settling, or run to waste through such zinc.

The partly-dried slime is removed from the clean-up press or vacuum tank and may, if containing a small amount of moisture, be fluxed and melted without further drying; it may be dried before fluxing and melting, or it may be roasted before fluxing and melting, the roasting being very similar to that given precipitate that has not been acid treated and may include adding a small amount of niter. Acid treatment by sulphuric acid followed by a thorough roasting is the method usually employed in America on gold slime, while silver slime with or without roasting is melted without acid treatment.

Sulphurous Acid Treatment. — Refining by sulphurous acid (H_2SO_3) does not differ materially from that by sulphuric acid. Metallic or solid sulphur is burned in a simple generator or air-tight stove supplied with air under a pressure of a few pounds, sufficient to give the necessary oxygen to form sulphur dioxide (SO_2), and force it into a clean-up tank to be absorbed by the water therein with the formation of sulphurous acid, as:

$$SO_2 + H_2O = H_2SO_3.$$

The precipitate may be added before or after the water has absorbed sufficient acid. The zinc is dissolved as:

$$Zn + H_2SO_3 = ZnSO_3 + 2\,H.$$

The sulphites of lead and lime formed appear to be more soluble in an excess of the acid than the sulphates formed similarly in the sulphuric-acid treatment, but are not soluble in water. The method seems to be as efficient as the sulphuric-acid treatment with a remarkably low cost, while the solid sulphur is much easier transported than the liquid and dangerous sulphuric acid. The method has been successfully used in America.

Bisulphate of Sodium Treatment. — In refining or cutting down the zinc by a solution of bisulphate of sodium ($NaHSO_4$), — which occurs as a solid substance, — the chemical is dissolved in a stock tank to dilute the acid solution used in treating the zinc to equal about 10 per cent H_2SO_4. The dissolving takes place as in the ordinary sulphuric-acid method, but it is claimed with less danger from gassing or from poisoning by arseniureted hydrogen. The reaction being:

$$2\,NaHSO_4 + Zn = Na_2SO_4 + ZnSO_4 + 2\,H.$$

The principal advantage of using bisulphate of sodium is that, occurring as a solid chemical, it may be easier transported than sulphuric acid, and may be a cheaper method of refining. It will be noticed that not all of the sulphur is made available to unite with the zinc. The method is being used to a limited extent in South Africa.

CHAPTER XV

FLUXING AND MELTING

Constituents of Zinc Slime to be Melted. — The nature of the precipitate to be smelted varies widely, due to the different conditions under which the precipitation takes place, and more so as to the methods of cleaning-up and preliminary refining. This relates to the amount in the precipitate of gold and silver, of zinc, of other base metals and substances, what these bases are, and in what form they exist, whether metallic or as oxides, sulphates, etc. The constituents of a precipitate may be divided into four classes:

Gold and silver.
Metals and bases as oxides and sulphates.
Metals in their metallic form and other reducers.
Silica or lime.

The larger the amount of gold and silver, the less will be the amount of flux required, for the bases to be slagged off will be less. The percentage of precious metals in the precipitate is controlled by the manner in which the precipitation is conducted and the fineness of the screen through which the clean-up is made. The percentage in the precipitate is further increased by the roasting or acid treatment which removes part of the bases. The bases changed to sulphates by the acid treatment are removed by washing, except those sulphates which are insoluble in water and sulphuric acid, mainly, calcium sulphate and lead sulphate. Imperfect washing may cause some of the soluble sulphates to remain in the precipitate. The oxides are formed in the roasting process, whether without or following acid treatment. The base metals in their metallic form are those that have not been acted upon by the roasting or acid treatment, or are to be found when no such treatment has been given the precipitate. They are principally zinc and also lead where the zinc-lead couple has been used. Other reducers are insoluble cyanogen compounds that have been precipitated in the zinc boxes. Silica is due to turbid and slimy solutions passing through the boxes, or to a deposition of dissolved silica or alumina on the zinc. Lime and other alkaline substances may be deposited in the boxes when the protective alkalinity is high, and not be removed by roasting or acid treatment.

Purpose of Fluxing and Smelting. — The purpose of fluxing and smelting is to form a slag containing the base metals and substances, mainly as silicates, borates, and dissolved oxides, and to form a bar of fine bullion containing the gold and silver. To form these two,—the slag and the bullion,—it is necessary to add flux to the precipitate that will unite with the bases in the proper proportion to give a slag that is fluid, so that the small shots and particles of gold and silver may easily settle down into the bar of bullion, instead of remaining suspended to form a rich slag: one that will form and be liquid at a low heat; that will be neutral and not so basic as to destroy the crucible through abstracting the clay in its demands for acid or siliceous flux to slag with the bases of the precipitate; that will not be so acid as to utilize more flux than can be economically gotten along with and require a high heat to make fluid — a neutral slag is usually more liquid; and that will be of small bulk to fuse the largest amount of precipitate and take up the least melting space.

Sodium and Potassium Carbonates as Fluxes. — As fluxes used in melting zinc slime are sodium carbonate (Na_2CO_3), sodium bicarbonate ($NaHCO_3$), potassium carbonate (K_2CO_3), or potassium bicarbonate ($KHCO_3$), preferably the carbonates of sodium, and of these the sodium carbonate which is $1\frac{1}{2}$ times stronger than the sodium bicarbonate. The sodium and potassium carbonates and bicarbonates are a basic flux and unite with acid fluxes or substances, especially with silica, to produce a sodium silicate, as:

$$Na_2CO_3 + SiO_2 = Na_2SiO_3 + CO_2.$$

The carbonates fuse between 800° and 900° C.

Borax and Borax Glass as Fluxes. — Borax ($Na_2B_4O_7 . 10\ H_2O$) and borax glass ($Na_2B_4O_7$), the anhydrous borax from which the water of crystallization has been driven off, are acid fluxes and unite with basic fluxes and substances to form a borate or to dissolve the metallic oxides and hold them in the slag solution.

The formula of a borate may be written $Na_2O \cdot 2\ B_2O_3$, in which the boracic acid (B_2O_3) acts similarly to silica (SiO_2), or in this way:

$$ZnO + Na_2O \cdot 2\ B_2O_3 = Na_2O \cdot ZnO \cdot 2\ B_2O_3.$$

Silicates and borates dissolve together to form what may be termed silicate-borates, which will lower the fusing or slag-forming points and promote fusion in general. Borax melts at 560° C. It helps to give a quick fusion and a liquid slag. Too much or too little borax in the fusion will cause a thick slag. Borax glass has nearly twice the strength or available $Na_2B_4O_7$ of the hydrous borax.

Silica as a Flux. — Silica (SiO_2) is an acid flux which combines with metallic oxides and bases to form silicates containing varying quantities or proportions of silica. Generally the higher the proportion of silica in the silicate, the less fusible or fluid it is. Silica itself can only be melted at a tremendous heat, but melts easily with a basic flux.

Fluor Spar as a Flux. — Fluor spar (CaF_2) is a neutral flux that fuses at a high temperature. It is but little used. Its chief value is to give fluidity to the slag.

Niter as a Flux. — Niter, potassium nitrate (KNO_3), is a basic flux fusing at 339° C. It is used to oxidize the base metals that they may more readily pass into the slag as a silicate, borate, or dissolved metallic oxide, instead of into the bullion in a metallic state. It is especially valuable for oxidizing zinc when the same has not been effected by roasting, or the zinc removed by acid treatment. It does not so readily oxidize lead and the other metals, as it gives off its oxygen at too low a heat. Niter in the process of oxidizing gives off its acid portion leaving the base, potassium, which actively combines with silica to form a potassium silicate and will abstract the siliceous matter of the crucible if silica is not otherwise available, and is assisted in this by the oxidizing influence of the acid portion on the carbon of the crucible.

Manganese Dioxide as a Flux. — Manganese dioxide (MnO_2) is a basic flux and oxidizing substance, as niter, but is not as destructive to the crucible. It is a better oxidizer of lead than niter, though with silver bullion it causes more silver to enter the slag.

Determining the Flux to be Used. — The flux for a precipitate cannot be calculated in a practical way. A method that has been used is to prepare test charges of precipitate and flux, smelt them in assay crucibles in the assay furnace, and observe the resulting slag and button, probably assaying or panning the slag to find how low in value it is. This method is rather unsatisfactory, for the fusion in an assay furnace is a quick one at a high heat, as against the slow and lower heat when melting on a working scale. Also the extent to which the charge will abstract silica from the graphite crucible cannot be well determined; in fact, a higher silicate containing a larger proportion of silica will generally be made in the assay furnace fusion. In preparing the flux for a precipitate at a new plant, past experience is relied upon in connection with a careful observation of the conditions under which the precipitate has been prepared, to indicate the quantity and proportions of the flux to be used. The melting is watched and studied, the effect of the flux on the crucible is noted, the slag is examined for its character and later assayed, and any necessary flux added. In this way a formula for the flux is worked out.

The flux must be varied according to what is to be slagged off, and it would appear that the proper way to discuss the proportions would be to take a type of flux and vary it to meet the different requirements, for an examination of the formulæ given by various authorities gives little information, except as the proportions are varied to meet variations in the precipitate. The following is given as the normal extremes of a well proportioned flux that has been found very satisfactory:

	Low	High
Precipitate	100 parts.	100 parts.
Borax glass	12 parts.	30 parts.
Sodium carbonate	6 parts.	15 parts.
Silica	3 parts.	8 parts.

The borax glass (or equivalent of borax) as an acid flux unites with the bases and oxides of the base metals to form borates, it also dissolves the metallic oxides that they may remain suspended in the slag. The sodium carbonate (or sodium bicarbonate or the potassium carbonates) as a basic flux unites with the acid constituents which are mainly if not entirely silica, forming a sodium (or potassium) silicate which acts as a flux on bases and metallic oxides for which sodium or potassium alone is not a flux. The silicates acting with the borates as silicate-borates lower the melting point of the charge and assist in the fusion; their increased complexity more readily dissolving and holding suspended the slag constituents of the charge. The silica unites with the soda and the base oxides to form various silicates, so that the slag is partly a complex solution of various silicates, a thing which assists the fusion. It would appear at first that sodium carbonate being a base should not be added to the precipitate which is basic itself, that only borax and silica should be added, but it is found that the formation of a certain amount of sodium silicates by the addition of soda is desirable to get a rapid and satisfactory fusion at low heat. For, as mentioned before, sodium silicates are a flux for the bases and metallic oxides, as in the formula $Na_2O \cdot ZnO \cdot SiO_2$ of a sodium zinc silicate. Soda is a desulphurizer and, consequently, may be useful for that purpose with a precipitate containing sulphur as a sulphide. Fine quartz tailing or other sand high in silica is generally used to supply the silica. Ground glass, assay slag, and less preferably the slag from previous meltings is sometimes used. These are already complex silicates and easily fused, but had best be dispensed with in favor of silica and sodium carbonate, as giving more desirable silicates and a greater effect.

A fluxed charge may be considered to be divided into two components, acid oxides and basic metallic oxides, which are to be fused to a liquid neutral slag. The acid oxides consist of the silica (SiO_2) and the boracic acid or boron oxide (B_2O_3) of borax, which are supplied as a flux. The basic metallic oxides are the constituents of the precipitate outside of the gold and silver, to which is added sodium carbonate (Na_2CO_3) that its sodium oxide (Na_2O) may form the desirable sodium silicates with the silica to act as an acid flux and carrier for the bases.

Variations Due to Zinc and the Use of Oxidizers. — The low amount of flux in the formula given before will give a high-grade bullion on a precipitate containing 60 to 80 per cent of bullion and not roasted or acid treated, such as that screened through a 40 or 60-mesh screen. As the amount of zinc increases, the amount of the flux must be increased or more zinc will enter the bullion. The low amount of flux will give a slag containing considerable of the precious metals and quite often some matte. So that it is more suitable and economical for a silver than for a gold plant, which would use more flux, perhaps to the extent of the high amount and more when the percentage of bullion obtained is low and the base metals are in a metallic form.

Where the amount of zinc is high and it has not been oxidized through roasting but tends to enter the bullion, from 3 to 10 per cent of niter may be added to the precipitate, — even 20 per cent has been used when the slime contained lead, — for the purpose of oxidizing the base metals. Manganese dioxide has been used in quantity up to 40 per cent and is especially recommended for oxidizing lead which usually enters the precipitate through the zinc-lead couple, and if not oxidized and slagged off or combined with sulphur as a matte — a sulphide of a base metal — will enter the bullion. The manganese base of the manganese dioxide does not appear to so actively combine with silica as the potassium base of niter does, and consequently does not attack the crucible to such an extent. Graphite crucibles are usually made of one part of fire clay and two of graphite, the clay acting as a binder to give form and plasticity. Clay is practically a silicate of alumina and the potassium base set free by the niter in its oxidizing action will act upon the clay, unless plenty of silica be otherwise present, thereby corroding and destroying the crucible, which is assisted by the oxidizing of the carbon by the niter or manganese dioxide. The use of niter or manganese dioxide in the flux is generally avoided by thoroughly roasting or acid treating the precipitate. Their use involves considerable care so that the experienced melter prefers to dispense with them or use clay liners in the graphite crucible when using a large amount of the oxidizers. When used, an amount of sodium equal to the potassium or manganese should be omitted.

A large amount of zinc is also taken care of through the use of plenty of silica and soda to form a zinc silicate or sodium zinc silicate; the amount or proportion of the silica should be kept high or it will be abstracted from the crucible, for zinc silicate has a corrosive action upon crucibles. The soda is also valuable in this way by having an oxidizing influence on the zinc and giving a liquid slag. But as borax also fluxes the zinc by the formation of a borate, it should be used together with the soda and silica and to jointly assist by the formation of silicate-borates, more especially when the zinc is in the form of an oxide as borax has a high dissolving effect on the oxides. When metallic zinc is present, it bubbles and boils off at a high temperature with dense fumes of zinc oxide, which may cause a loss by carrying off the precious metals.

General Variations and Fluxing Procedure. — The presence in metallic form of other metals than zinc, and other reducers, call for similar treatment as zinc. If the precipitate contains much sand, the silica in the flux is lessened or dispensed with. The presence of a large amount of lime would call for an increase in the silica or borax, or the use of less soda. The addition of soda will increase the fluidity of the charge, but an excess of soda or its addition without sufficient silica in the charge must be guarded against on account of its corrosive action on the crucible. It should always be borne in mind that soda is hard on a crucible. Borax increases the fluidity under normal conditions, but too large an excess makes the slag thick Borax should be used to thin the charge and give fluidity when the silica is not in excess — when the slag is not stringy — its use for that purpose is preferable to soda as it is not destructive to the crucible. Silica in the quantity giving a neutral slag gives a fluid charge, which increases in pastiness as the amount of silica is increased. The higher the silica, the less corrosion of the crucible. Where the charge is too thick from excess of silica, it should be thinned down by the addition of soda. An acid or siliceous slag is stringy, can be pulled into long strings when cooling, and is glassy and brittle when cold. A basic slag is " short," cannot be pulled into strings when melted or cooling, and is stony and dull looking when cold. Raising the temperature liquefies the pastiness due to a high percentage of silica, and makes the slag more fluid in general. A noncorrosive slag at a red heat may attack the crucible at a white heat, because the higher the heat the higher will be the silicate formed, but this corrosion increases with the time, for the higher silicates are slowly formed in this way. The appearance of graphite in the slag indicates that the crucible is being attacked and that more silica should be added to the charge. The addition of glass, assay slag, or fluor spar should thin down the charge without materially varying the acid or basic qualities of the slag, at least not to the unsafe side, and may be preferable to the practice of adding lime with a corresponding amount of borax and silica to confer fluidity and complexity.

The fusion must be carried on for some time after the charge has subsided and settled in a quiet fusion, to insure all the constituents being decomposed or in a homogeneous slag, and the precious metals fused and settled into the bullion in the bottom of the crucible. Shots of precious metal throughout the slag or settled on top of the bar indicate too thick a slag, owing to an insufficiency or wrong proportion of flux or to too low a heat. Bullion of a high degree of fineness, a rich slag, and a high cost of treatment, fluxing, and melting go together. Dehydrated or anhydrous fluxes, those free from moisture and water of crystallization, have been generally recommended that loss by ebullition and boiling may be minimized, but the success being attained in melting partly-dried precipitate will cause less attention to be paid to this. Oxidizers especially cause boiling and ebullition. Clay liners set inside of graphite crucibles have been used to lessen the corrosive effect of the slag. Their use causes a long and slow fusion with a large quantity of flux, so that they are but little used. They are valuable when using an oxidizing agent, as niter or manganese dioxide.

Matte Formation. — A matte is a combination of sulphur with a base metal as a sulphide. It may be formed artificially by adding sulphur if the base metals are present, as they usually are, or by adding iron if the sulphur is present. With both sulphur and base metals in the precipitate, it forms naturally. The sulphur in the precipitate may be principally as zinc sulphate due to poor washing of the precipitate after acid treatment, to lead and calcium sulphates formed by acid treatment and insoluble, and to insoluble sulphur compounds deposited in the zinc boxes. The removal of the bases by acid treatment, the oxidation of the bases and sulphates by roasting, or the use of an oxidizer, as niter or manganese dioxide, reduces the amount of matte or eliminates its formation. The use of an excess of soda will reduce the amount of matte formed, for soda is a desulphurizer, while the basic slag formed will dissolve the matte and hold it in suspension. A matte increases the fineness of the bullion by taking into itself base metals that would otherwise enter the bullion. It lowers the value of the slag by collecting gold and silver into the matte that would otherwise be found in the slag, but it also prevents some gold and silver from entering the bullion.

Annealing of Graphite Crucibles. — The graphite crucibles must be thoroughly annealed, by being placed in a warm or hot place for perhaps a week, and finally by being slowly brought to a high heat, to drive off all the absorbed moisture. This is done by placing the crucible on a boiler, stove, or furnace, and finally in the fire box, before using it for the first time. If this is not done, the sudden heating of the crucible will in all cases crack off part of it by an explosion resulting from the steam formed by the absorbed moisture. There is less tendency for the crucibles to break after they have once been annealed, but between melts they should be kept in a warm, dry place.

Melting Furnaces. — Two types of melting furnaces are used: the stationary, in which the crucible must be removed by a pair of tongs in pouring the bullion, and the tilting furnace, in which the entire furnace with the contained pot is tilted for pouring off the slag and bullion. The application of the tilting furnace to the melting of cyanide precipitate is of comparatively recent origin and has been very successful, though trouble has been encountered in some cases in learning the best method of handling it. Both hard fuel, such as coal, coke, and charcoal, and soft fuel, as oil, distillate, and gasoline, are used. Gasoline or distillate is the most advisable fuel where only a small amount of gold bullion is to be melted at a time, as the furnace and accessories are obtained and installed at small cost. For plants of some size, oil or cheap distillate is the most satisfactory and economical, except where the local conditions make the cost of oil or distillate inordinately high above hard fuel. Liquid fuel gives a higher and quicker heat with less labor and dirt than hard fuel, but is more severe on the crucibles.

Preparation of Precipitate and Flux. — The precipitate may be only partly dried before putting into the crucible. This will save the labor and loss involved in drying and in the dusting when mixing with flux, but care must be used in the melting to add the precipitate before the last has fused down, or loss by spitting may result. Or the precipitate may be thoroughly dried before placing in the crucible, the flux being added before or after the drying. In some cases the precipitate is partly, but not thoroughly dried, then mixed with the flux and made into briquettes by a briquetting machine, that the briquettes may be handled without loss by dusting. When the precipitate is not thoroughly dried but is melted moist, the necessity of briquetting is small.

The amount of the precipitate is weighed or estimated for adding the necessary amount of flux. The flux may be added by charging it and the well-dried precipitate into a closed revolving barrel, by spreading it over the precipitate and shoveling to mix, or by spreading the precipitate and flux in alternate layers which receive some further mixing when being transferred to the crucibles. A thorough mixing of the flux and precipitate is good, but is not absolutely essential. If it tends to cause a loss by dusting, it need not be so thoroughly done.

Melting Procedure. — The annealed crucible is generally loaded with the fluxed precipitate nearly to the top before starting the fire. After the crucible becomes heated the charge subsides through the melting of the precipitate in the bottom, and more precipitate is added as space is made. Care is taken to add the precipitate before that in the pot has fused down, that there may be little dusting of the newly-added fluxed precipitate. Often a little borax glass is spread over the charge to melt quickly and prevent loss by dusting and fumes. When no more precipitate can be added to the pot, the top of the charge is allowed to fuse, and after having subsided for some time into a quiet fusion is stirred with an iron rod previously made red hot to prevent the slag and metal from adhering to it. The slag and fusion are critically inspected to note if additional flux or longer heat is required.

The fusion being brought to the proper condition for pouring, by the application of sufficient heat for a period long enough to bring the mass into quiet fusion, and by the addition of any required flux to vary or thin the slag, it is well stirred with the heated-iron rod to settle any shots of metal before pouring. The fusion may be poured into a conical mold or into a regular bullion mold, allowing the slag to overflow or run through a slot into the slag mold, while the precious metals sink through the slag, to be held in the bottom of the bullion mold. The slag only may be poured into the conical or slag mold; or the slag may be dipped off by a heated assay crucible held in a pair of assay tongs. The slag may be granulated for easy sampling and shipping by being slowly poured into water. The buttons of bullion from the bottom of the conical molds or the slabs from the bullion molds are collected and melted together to form the bar of bullion for shipment, or if the bullion is retained in the pot it is poured after the final fusion. After the slag or slag and bullion have been poured, fresh precipitate is charged into the crucible and the melting continued. In some cases before pouring the bar for shipment, the slag is dipped off and some attempt made to refine the bullion, it being finally cast without any slag, though casting a bar without a covering of slag appears to be of no advantage, in fact may cause trouble by the metal sloughing off the top of the bar. The mold should be painted with a lime emulsion or a carbon, as a mixture of lampblack and oil, soot from burning waste, etc., before the pouring to prevent the bullion and slag from sticking to the mold. The mold must also be well warmed that the cold mold may not be cracked by the sudden introduction of the hot metal, and that the slag and metal first introduced will not be chilled, so that a good bar or button of bullion cannot be secured.

Treatment of Slag and Crucibles. — The slag obtained and the old crucibles all contain considerable metal. The method of securing this varies. It is invariably in a metallic condition, as shots. The material may be run through the crushing mill and perhaps most of the metal caught by amalgamation. It has been run through a separate stamp battery, when the quantity was large, to be concentrated, the concentrate being melted in the melting furnace and the residue cyanided or shipped to the smelters. It has been charged into an amalgamating barrel and amalgamated, the residue being cyanided. Cyaniding the residue is not usually very efficient. It has been sacked up and shipped to the smelters. A lesson has been taken from smelter practice by pouring the slag into a conical mold with a clay-stoppered hole a few inches above the bottom or apex of the mold. The slag is tapped by removing the clay plug after a shell ½ inch thick has formed, allowing the core to be granulated for milling or shipment to the smelters by running into water, while the richer shells, into which much of the shot and prills of metal have settled, are treated separately or used as flux in melting precipitate.

Treatment of Matte. — Matte forms on top of the bar of bullion as a tough, brittle film of base metal and sulphur. It is usually undesirable, though it has been artificially produced for the purpose of increasing the grade of the bullion. As it contains considerable gold and silver, it should be saved to be melted into a large bar and shipped, or to be refined at the plant. It has been fused with borax and soda, and the addition of niter to oxidize the metals, to give a button of gold and silver and a matte and slag of very low value. A. E. Drucker * gives the following method of obtaining an extraction of 85 to 94 per cent of the value in the matte. Alternate layers of borax, matte, and cyanide, all finely crushed, are put into a graphite crucible. The crucible is heated at a white heat for two or three hours until the charge subsides and action ceases, when the thick slag is skimmed off and the contents of the crucible poured.

Smelting with Litharge and Cupellation. — In the lead smelting of zinc slime, the precipitate with or without acid treatment is dried to a small per cent of moisture and mixed with litharge, borax, silica, and powdered coke. The fluxed material is briquetted to enable easier handling and less dusting. The briquettes are melted in a cupel furnace, the resulting slag being drawn off. The lead that has been reduced with the gold and silver is cupeled off as litharge by means of a current of air blown across the molten metal, oxidizing the lead to litharge, which is drawn off to be ground and reused in the next melting. After the lead has been removed in this way, the fine gold and silver is allowed to cool, when it is removed, cut up, and melted in the usual graphite crucible into a bar of high fineness for shipment. The slag, cupel bottoms, sweepings, and by-products are smelted in a small lead blast furnace, the lead produced being cupeled later. The method, like the zinc-dust process of precipitation, is well adapted for large plants producing a considerable quantity of precipitate. It is apparently a cheaper and more efficient method of turning the precipitate into fine bullion than the usual practice.

Assay of Zinc Precipitate.† — Zinc precipitate may be assayed by three methods: by scorification, crucible fusion, or a combination method involving preliminary refining by acid. In

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* *Mining and Scientific Press,* May 18, 1907. Recent Cyanide Practice, p. 260.

† See C. H. Fulton and C. H. Crawford in Bull. No. 5, South Dakota School of Mines.

the assay by scorification, $\frac{1}{20}$ to $\frac{1}{10}$ assay ton of precipitate is taken to 70 grams or more of test lead and a cover of a small amount of borax glass. The crucible fusion may be made with the following charge:

$\frac{1}{10}$ assay ton precipitate.
70 grams litharge.
5 grams sodium carbonate.
1 gram flour (or other reducer).
5 grams silica.
2 grams borax glass.

By the combination method, $\frac{1}{10}$ assay ton or more of precipitate is boiled for a continued length of time with 20 c.c. sulphuric acid and 60 c.c. water, finally filtered, washed, dried, incinerated at low heat, and residue fluxed and fused in the usual crucible fusion.

In all cases the slag and cupel of the first fusion should be ground up, fluxed, and fused in the same crucible or scorifier, and the results added to the first fusion.

CHAPTER XVI

CYANIDATION OF CONCENTRATE

The cyanidation of concentrate or the separated sulphide constituent of an ore involves no departure in principle from standard cyanide practice, but simply stress upon certain parts of the manipulation to meet the abnormal conditions connected with the sulphide and its treatment. A clean gold ore with the precious metal finely divided and upon the breaking planes or faces of the crystals of the ore, and a sulphide with the metal in a coarse state and interbedded with and in the pyritic crystals are the two extremes, of which the base or pyritic ores being cyanided to-day are an intermediate. The methods of cyaniding concentrate include roasting, leaching, agitation, filtration, decantation, oxidation, and fine-grinding as with ordinary ores. The prominent characteristics to be considered, are: Precious metal, especially with gold ores, is usually in a comparatively coarse metallic state, susceptible of being amalgamated to a certain extent, and requiring considerable time for dissolution. The holding of the precious metals to a large extent within the pyritic crystals, requiring fine-grinding, oxidation, drying, or roasting to liberate the value. The presence of iron, copper, lead, arsenic, antimony, etc., either metallic or as compounds, and the resultant high consumption of cyanide and the tendency of the solution to foul. The necessity of meeting the high acidity generated. The action of cyanide and alkalinity upon the sulphide to form soluble or alkaline sulphides and the consequential necessity of supplying oxygen. The quick settling of the concentrate and its tendency to pack and become impermeable. Its comparatively high value and that of the solutions resulting from its treatment.

Treatment by Percolation. — The treatment of concentrate by percolation usually requires from 10 to 30 days to obtain a good extraction. In some cases the concentrate as obtained is stored under water to prevent the formation of ferrous sulphate and sulphuric acid through the decomposition of the pyrite. Though where the concentrate is not to be finely ground it is better to spread it out to dry, as this causes the grains of pyrite to fall apart, decompose, and allow solution to enter them, that the metal may be better dissolved. If the concentrate, as charged into the leaching vat, contains much soluble acidity, lime may not be added to it, but the charge water-washed until drainings indicate no acidity, to be followed, to remove the latent acidity, by a wash of water saturated with lime — a saturated solution of lime water will contain about $2\frac{1}{2}$ pounds CaO per ton — until the drainings show some alkalinity. The latter method is not advisable unless the cyanide solution can be used with a protective alkalinity high enough to meet the acidity as it may be generated in the charge, which would be indicated by the outflowing solution always showing a protective alkalinity. When lime is added to the concentrate it is impossible to say, without studying each case in detail, what quantity should be used and how fine it should be crushed. It should be added in quantity and crushed to such a mesh that its alkalinity will be dissolved and given off at the same rate that the acidity is generated. This, of course, cannot be satisfactorily accomplished. The results of laboratory tests and experience with previous charges must be relied upon. With a gold sulphide, from 3 to 10 pounds of lime will usually be sufficient; this had best be added unslacked and crushed to a 10-mesh, the larger part being coarse granules. After the concentrate and lime have been charged into the leaching vat, they should be water-washed until the drainings show alkalinity. Any lack of alkalinity through the slow dissolution or insufficiency of the lime should be met by alkalinity in the water wash or solution. It is inadvisable to dispense with water-washing and at once run on weak cyanide solution, as a better neutralization of the acidity is made and all the soluble compounds are removed instead of entering the cyanide solution, which they may foul and make more viscous, and finally be precipitated in the zinc boxes.

The charge is drained after the drainings show alkalinity, when the first cyanide solution is run on, preferably one low in cyanide and strong in alkalinity. The weak solution is run on once, or for a short time, until it is apparent that the active cyanicides have been met and the strong cyanide solution will not be too quickly destroyed, after which strong standardized solution is used until the dissolution is accomplished, with final washes of weak solution and water. The strength of the strong solution will vary from .2 per cent (4 pounds) to .75 per cent (15 pounds), seldom higher; a strength above .5 per cent (10 pounds) is generally undesirable. Strong solutions in the presence of sufficient oxygen are more active than weaker ones in dissolving the precious metals, but they also act more upon the base metals and compounds, causing them to enter the solution more and a greater consumption of cyanide. Fresh strong solution should be constantly supplied, perhaps by continuous leaching, to replace about each particle of gold that which has been utilized and weakened by dissolving coarse gold or neutralized by the strong cyanicides. Each strong solution should be well drawn off that air may be drawn into the charge, both to assist in the dissolution of the precious metals and, by oxidation, to decompose and break open the pyrite for better contact between the gold and silver and the solution, though this is bound to develop considerable acidity. The iron and other metallic salts from the decomposition of the sulphide abstract oxygen from the charge in effecting their formation, and by supplying plenty of oxygen the salts are finally oxidized into harmless oxides or less active cyanicides, as the oxidation of ferrous salts into the ferric oxide or hydrate. The first solution should be allowed but short contact with the charge if rich solutions are undesirable, as where there is considerable leakage or they go directly and undiluted to a zinc box, the shavings of which they would coat with solid metallic gold to cause considerable metallic zinc to enter the melting. Fresh solutions are necessary to supply oxygen to get quicker dissolution, consequently continuous leaching with periodical complete drainings to aerate the charge is best. It is not only desirable but usually essential to oxidize artificially, not by chemical oxidizers, but by pumping air through the charge at a pressure of 3 to 5 pounds below the filter bottom when the charge is drained. It is not advisable to draw air through by means of a vacuum pump applied beneath the filter bottom, on account of packing the charge. Leaching charges of concentrate should be shallow, say 4 or 5 feet, to allow of easy aeration. The solutions should be well aerated, which may be, when excessive aeration is desired, by means of an air cock between the pump and the solution tank whereby a small quantity of air is drawn in and pumped with the solution, or by allowing a little air under pressure to escape into the solution tank. The solution should be tested for its reducing power and for alkaline sulphides. It may be advisable when about half the treatment period has passed, to shovel the charge over, placing the bottom on the top. This is an excellent method of aerating, especially when it places the bottom where the least dissolution has taken place owing to the absence of oxidation — which is often noticed in treating sulphide — on top where the greatest dissolution is effected. However, when air is occasionally pumped through the charge, shoveling over may be more beneficial on account of the packing and peculiar cementing or caking of the sulphide. To lessen the tendency to pack and cake, coarse tailing or ore sand may be mixed with the concentrate.

The protective alkalinity of the inflowing solution should be sufficient to give a slight protective alkalinity in the outflowing solution. Though a high protective alkalinity is necessary to protect the cyanide from decomposition, it will form some soluble or alkaline sulphides, for many sulphides are acted upon in this way by alkaline solutions, while cyanide decomposes these and other sulphides to form the alkaline sulphides. The alkaline sulphides abstract the oxygen necessary for dissolving purposes and in weak solutions reprecipitate silver, and perhaps gold or at least retard its dissolution, so that they should be prevented from forming or should be gotten out of the solution when once formed. These are removed as insoluble sulphides by the zinc in solution or by the addition of lead acetate. The effect of adding lead acetate occasionally to the solution to the extent of a total of one-half to one pound per ton of concentrate should be studied, even if no alkaline sulphides are ever found in the solution. Another way of accounting for the reduced extraction, that has often been noted when using a high protective alkalinity on sulphide gold ore, is that the alkali acts upon the pyrite and partly-decomposed pyrite, causing a gradual oxidation into ferric oxide (Fe_2O_3), in which process is consumed a large amount of oxygen, this being taken from the solution causes it to lose its dissolving power. Whether the cause is the reducing action of alkaline sulphides or of ferrous salts or some unknown process, it is apparent that aeration is a most important thing. The effect of a high and a low protective alkalinity on both the rate and extent of extraction and the consumption of cyanide should be studied.

Treatment by Agitation and Fine-Grinding. — Treatment by agitation will usually give a little higher extraction than leaching in about one-sixth to one-third the time, and with a less consumption of cyanide, but is generally carried out in connection with fine-grinding. The fine-grinding is best accomplished in a tube mill or grinding pan. The pulp may be caused to flow over amalgamating plates, for in some cases over 50 per cent of the gold may be secured as amalgam, reducing the cost and losses in the subsequent cyaniding, and probably reducing the time of dissolution by removing the coarse gold. Unless the concentrate contains cyanicides that it is desirable to remove before applying the cyanide, it may be advantageously ground in a medium cyanide solution — about .1 per cent (2 pounds). Mechanical agitators, unless of a special type that can be started while raised free from the charge, are not suitable owing to the high specific gravity of the concentrate and its tendency to pack. Some form of air agitator is preferable, and in all cases air should be supplied during the agitation. It should be learned how the dissolution progresses, for with base ores it has been noted that after the passage of some time, the rate of dissolution rapidly falls until the old solution is removed and aeration effected, when new solution again causes a rapid dissolution. If supplying air to the charge does not cause the gold to go into solution with the maximum rapidity, the charge should be allowed to settle, the clear solution syphoned off, and new solution added for further agitation. Treatment of slimed sulphide by decantation is easy as it rapidly settles to a small bulk, and while it gives satisfactory results, the up-to-date plants shorten the time of treatment by filtering the pulp after the bulk of the value has been washed out by decantation. Most of the leaf filters are unsuitable for handling this class of material, since on account of the high specific gravity of the sulphide, the cake must be formed within a few minutes, or the sulphide will settle out of the solution, and owing to the richness of the solution a thorough and highly-efficient wash must be given. The Kelly filter press making a cake under pressure in 2½ to 5 minutes is now in satisfactory use in such plants.

General Considerations. — Sliming the sulphide should be carried as far as possible, as giving a higher and quicker extraction. It is often the case that the finer concentrate is higher in value before regrinding and treatment, and lower in value after treatment than the coarser concentrate. In some cases the finely-ground sulphide may be mixed with coarse sand and successfully leached, though agitation is better, for it gives a higher extraction than leaching, a thing that is not noticeable with gold ores but holds to some extent with silver ores. Treatment costs by agitation are generally slightly less than by percolation, for the less consumption of chemicals, due to lessened oxidation and formation of cyanicides, overbalances the cost of agitation. A plant for leaching concentrate is comparatively inexpensive, consequently that process is the one often employed by small operators, whereas with a large amount of concentrate to be treated the high installation cost of a sliming and agitation plant is soon met by the increased extraction. The concentrate after treatment often contains sufficient value to warrant its shipment to the smelter, if the original heads were extremely high, or to be exposed on a dump for retreatment later, after it has oxidized, or by some new process yet to be devised to treat this class of material. In case of placing on a dump, a thorough washing should be given, that no soluble cyanogen may be left remaining to effect a dissolution that will later be washed out by rains and lost. Many plants that are now shipping their sulphides would find it more profitable to first treat them in a simple leaching plant, or more expensive fine-grinding and agitating plant, before shipping the residue; the cost of treatment being offset by a return of nearly 100 per cent of the amount extracted by cyanide, instead of on a basis of 95 per cent as paid by the smelters, and the lower freight rate and smelter charge on the lower grade of material shipped. To which may be added the shrinkage of the actual value and quantity, which is made by the smelters, as a factor of safety for their own protection and profit, and the quicker realization of the value in the concentrate when produced at an isolated mine. In some cases where a large amount of concentrate is produced, it might be advantageous to run it continuously to a grinding mill and amalgamating plates and ship the residue. Treatment of sulphide involves a problem as to whether it should be cyanided with the ore or removed by concentration and treated separately. When treated with the ore, much of the dissolvable value in the sulphide is not obtained, on account of the short treatment given the ore, but this may be more than offset by the cost of a concentrating plant and its operation. As the sulphide is small in quantity even if high grade, and large in quantity but low in grade, treatment of it in the ore without concentration becomes more advisable. Fine grinding and agitation of the ore and a quick dissolution of the larger part of the value in the sulphide make for treating the sulphide in the ore, especially where means are provided for grinding the sulphide finer than the ore in general. This may be performed by a slow-speed Chilian mill discharging the lighter particles by overflow, or by returning to the tube mill the coarse sand and sulphide separated out by a cone classifier, or a "roughing" or concentrating table following the tube mill and making a closed circuit of the heavier material. A determination of the extraction from the sulphide with varying periods of treatment, both when contained in the ore and as concentrated out, will give enlightenment on this subject.

Gold concentrate is nearly always amenable to cyanide treatment. Silver concentrate is less amenable, but important advancement in the treatment of this class of material may be expected. Iron pyrite, zinc blende, and galena present little interference or it is easily met. Arsenopyrite or mispickel, the sulphide of arsenic, may usually be treated satisfactorily in large quantities, though it has a high reducing action. Stibnite, the sulphide of antimony, often causes trouble or may prevent successful treatment. It is an active reducer, in which it is similar to mispickel but much more pronounced in its action, and by removing the oxygen, through the formation of alkaline sulphides, hinders or prevents the dissolution of the precious metals. It also holds to some extent the precious metals in a mechanical combination which the cyanide cannot break. Copper in unoxidized pyrite or in a hard state is but little acted upon by cyanide, and a considerable quantity is not a barrier to successful cyanidation, but when in a soft oxidized state readily dissolved by cyanide, a small quantity of copper may render the consumption of cyanide too high and require special methods of treatment and precipitation.

Where the sulphide contains a large amount of cyanicides which are not removable otherwise, especially iron and copper, a preliminary treatment may be given with a very dilute sulphuric or hydrochloric acid solution to remove the cyanicides. After the acid has performed its work, the soluble salts and any excess of acid remaining are removed by washing, the excess of acid to be used on the next charge. The acid treatment appears to effect a decomposition of the sulphide to give a higher extraction, in addition to greatly reducing the consumption of cyanide. It would appear that the acid solution for dissolving or altering the cyanicides could be prepared very cheaply, by passing the sulphur dioxide (SO_2) given off by burning sulphur into water to form sulphurous acid, as is done in making sulphurous acid for the acid treatment of zinc slime. The copper in the acid solution washed out of the ore can be recovered by running the solution over scrap iron.

Roasting of the sulphide will usually allow a high extraction to be obtained in a short length of time with a low consumption of cyanide. It has been generally abandoned in favor of fine-grinding and agitation, except with telluride ores where the roasting is used to separate the tellurium from the gold, since gold in combination with tellurium is not amenable to cyanide treatment outside of the bromocyanide process or analogous chemical processes. In the bromocyanide process, the addition of bromine to cyanide enables the cyanide to dissolve gold from telluride and arsenical ores, which it could not otherwise do, and increases the dissolving efficiency generally. It has been successful in treating sulphide, though but little interest has been taken in its use as the advantages of its employment are usually not warranted by its increased cost.

CHAPTER XVII

ROASTING ORE FOR CYANIDATION

Ores are roasted preliminary to cyanidation in two distinct ways: a dehydrating roast to remove the moisture and an oxidizing roast to remove the sulphur or tellurium and render the cyanicides innocuous. The dehydrating roast is a misnomer, for the process is simply a drying one. When ore to be dry-crushed is of a wet, clayey, talcose nature it is necessary to remove the moisture by passing the ore through driers, or it will clog the rolls and screens and be poorly sized. This drying process dehydrates or drives the moisture out of the ore, destroying to a large extent the adsorptive and flocculent qualities of the clayey matter, making it less plastic and more granular and leachable. It also opens the capillaries and parting planes of the more crystalline ore so that it is more easily fractured and crushed, and that the cyanide solution may better penetrate it. The influence of drying, and more especially of roasting, is very marked on some ores, which, due to the large amount of clayey matter and its adsorbent and plastic qualities, will adsorb the cyanide solution and refuse to allow it to be displaced even when mixed with much coarse material. In drying, if the ore contains much sulphide, a high heat or a real roasting tendency cannot be allowed — unless carried to a "dead" or complete roast — owing to the formation of ferrous salts and other cyanicides. In such a case the soluble salts or free acidity should be water-washed out of the ore, and the insoluble acidity neutralized by an alkaline wash. The drying of ore followed by dry-crushing was formerly much in vogue, but the perfection of fine-grinding, agitating, and filtering machinery has caused a decline in the practice, especially where drying and fine-crushing are necessary. The leaf filter is well adapted for handling the clayey slime, which formerly gave unsatisfactory results until dehydrated and rendered more granular and leachable by roasting.

The oxidizing roast, while rendering the ore more leachable and easily crushed, is for the purpose of driving off the sulphur as sulphur dioxide (SO_2), thereby converting the base metals into inert oxides, that are not reducers or cyanicides, and liberating the gold and silver mechanically held, or for driving off the tellurium chemically combined with the precious metals in a telluride ore. Due to the high insolubility in a cyanide solution of gold in combination with tellurium, such ore must be roasted before being cyanided, though bromine is used in connection with cyanide as the bromocyanide process in Australia, the bromine giving a higher solvent effect to the cyanide solution.

The changes occurring in the roasting of an iron pyrite (FeS_2) may be given as:

$$FeS_2 = FeS + S.$$
$$3\ FeS + 11\ O = Fe_2O_3 + FeSO_4 + 2\ SO_2.$$
$$2\ FeSO_4 = Fe_2O_3 + SO_3 + SO_2.$$

It is necessary that a complete oxidizing roast, often spoken of as a "dead" or "sweet" roast, be given the ore and that all the sulphur is driven off, for the insoluble ferric oxide (Fe_2O_3) finally formed is not affected by cyanide, whereas those compounds formed between the unoxidized iron pyrite and the final ferric oxide may be considered as cyanicides. In this way the consumption of cyanide is reduced and the amount of soluble salts entering the solution kept at a minimum. Telluride ores are roasted until the sulphide content is fully oxidized. The efficiency of the roast may be tested by taking 100 grams or more of the ore and shaking for a few minutes with the same number of cubic centimeters of water, filtering off the water and slowly adding to it a small quantity of new cyanide solution of the working strength. If no cloudiness appears, the ore is dead roasted and the consumption of cyanide due to cyanicides will not be high, but if a discoloration appears, the ore still contains soluble salts that will destroy cyanide and foul the solution. Or the test may be made by adding a few drops of a solution of barium chloride ($BaCl_2$), which will indicate soluble sulphates by forming a white cloud of barium sulphate ($BaSO_4$).

CHAPTER XVIII

CYANIDE POISONING

THE poisonous effects of cyanide are due to hydrocyanic acid (HCN), either that generated in the working of the process or that formed by the acid of the stomach, when cyanide is taken internally. Hydrocyanic acid, or prussic acid as it is sometimes called, is one of the most deadly poisons, investigators having been killed by it as a result of a few drops of the liquid acid falling on the skin. The gas or the vapor of the acid is likewise poisonous, producing headaches, dizziness, and nausea, which slowly pass away when the sufferer is removed to a pure atmosphere, while breathing the fumes of ammonia will afford relief. Exposure to small amounts of hydrocyanic acid gas seldom causes harm beyond possible headaches, depending somewhat upon the susceptibility of the person exposed to it. The contact of cyanide solution with the skin tends to irritate the skin, cause it to become hard and crack, and may cause sores and eruptions — a species of eczema — though with the weak solutions now in use there are seldom harmful results unless the skin contains open wounds or cuts. To prevent this and the ends of the finger nails from being eaten down, rubber gloves are worn when working in solution, as when cleaning the zinc boxes. Or the hands and arms are given a coating of vaseline, or even the stiff lubricating grease used in mills, to render the contact between the skin and solution less. The solution should always be displaced from the boxes by water preliminary to cleaning-up, to lessen the danger from contact with it and the hydrocyanic acid fumes arising.

Internal Poisoning. — When cyanide as in a solution is taken internally, the acid of the stomach forms hydrocyanic acid with it, which enters the blood as a blood poisoning, paralyzing the nervous system and muscular sensibility and suspending the action of the heart. The hydrocyanic acid acting in the blood deprives it and the tissues of the ability to absorb oxygen, resulting in severe cases of cyanide poisoning, of the sensation of strangling, and inability to get air or to swallow. With strong solution, or when having taken a large quantity of medium strength, insensibility results almost immediately, and death within a few minutes. Where the amount taken has been small, death may result after considerable delay and suffering, if an antidote is not at once administered. Small quantities of weak solution are not necessarily fatal, but unless an antidote is used the risk is great.

Treatment by Hydrogen Peroxide. — In a case of poisoning, whether by the vaporous hydrocyanic gas causing incipient insensibility or by swallowing cyanide solution, it is necessary to act with all speed possible. If a case of poisoning by being overcome by gas, the sufferer should be removed to a pure atmosphere, caused to breathe the fumes of ammonia, and given a number of hypodermic injections of a 3 per cent solution of hydrogen peroxide (H_2O_2) underneath the skin. In the absence of facilities to give injections a 10 per cent solution should be taken in internally that it may enter the blood and system. If the cyanide has been taken internally, as by drinking a solution, a wineglass or more of a 30 per cent solution of hydrogen peroxide should be taken at once, and subcutaneous injections may be made of a 3 per cent solution. The patient's throat should be tickled with the finger, or more preferably by a soft rubber hose, to cause vomiting, after which a fresh and more dilute solution of hydrogen peroxide should be given and the process repeated. Finally water, preferably warm, should be taken and vomited to wash out the stomach. The hypodermic injections may be given at several places over the body, and thereafter at intervals one minute apart and gradually lengthening until the patient is relieved. If a stomach pump is available, the stomach should be pumped out after each dose of the antidote or wash. In case the patient is unconscious and only able to breathe with difficulty or not at all, artificial respiration should be induced, as in a case of drowning, by kneading and compressing the body and pulling the arms, chest, and abdomen. Steps to promote the circulation may be taken by rubbing and kneading the body. Tickling the throat is perhaps the best method to produce vomiting, though emetics may be used, such as a spoonful of mustard in a pint of warm water, if they can be given at once and act promptly.

The action of hydrogen peroxide (H_2O_2) as a powerful oxidizer is to form an oxamide ($(H_2CNO)_2$) with the hydrocyanic acid and a cyanate (KCNO) with the cyanide, which are harmless, by decomposing the hydrocyanic acid and cyanide, as:

$$\left\{\begin{array}{l} 2\,HCN + H_2O_2 = (H_2CNO)_2. \\ KCN + H_2O_2 = KCNO + H_2O. \end{array}\right.$$

To do this advantageously it must be introduced before the hydrocyanic acid or cyanide enters or is absorbed into the system. The objection to the use of hydrogen peroxide as an antidote when cyanide is taken internally is its slow action, which may allow considerable of the poison to enter the system before the decomposing action is completed, so that the removal of the cyanide and the washing out of the stomach by vomiting may be of more value than the oxidizing action of the hydrogen peroxide.

Treatment by Cobalt Solution. — A solution of nitrate of cobalt, or other salt of cobalt, has been used with success as an antidote. It acts almost instantaneously to convert the cyanide into an insoluble and innocuous cyanide of cobalt. But an excess of the cobalt salt must be used to insure the immediate neutralization of the cyanide, which requires the excess to be removed by vomiting or the stomach pump, since the nitrate of cobalt itself has a somewhat poisonous effect.

Treatment by Ferrous Salts. — The quickest and best method of neutralizing the poison when it has been taken internally is by means of ferrous hydrate or carbonate. Due to the fact that these decompose quickly, they must be prepared at the time when used, which is accomplished by making up the following:

A. A bottle containing 7½ grams ferrous sulphate ($FeSO_4$) dissolved in 30 c.c. of water.

B. A wide-mouthed bottle with a capacity of about 400 c.c. containing 1½ grams caustic soda (NaOH) dissolved in 300 c.c. of water.

C. A tube or phial containing 2 grams powdered magnesia.

These three bottles, together with directions for their use, are kept in a convenient place in the plant, the bottles being tightly corked with stoppers that can be instantly removed. In a case of cyanide poisoning the three are emptied together into the larger bottle, well shaken, and drunk by the sufferer from the wide mouth of the bottle. The contents of the stomach should be removed and washed out by vomiting or the stomach pump. At the same time hypodermic injections of hydrogen peroxide may be given under the skin, if the case is serious, to reduce the evil effect of any cyanide that has entered the system. The magnesia is used to increase the alkalinity to the amount required to overcome the acidity of the stomach, and for quick conversion of the cyanide into a ferrocyanide, as the use of caustic alkali to that extent would be too severely caustic on the mucous membrane. The caustic soda and magnesia may be replaced by sodium carbonate (Na_2CO_3) equal in weight to the ferrous sulphate. In either case the cyanide is converted into a ferrocyanide ($K_4Fe(CN)_6$) which is nonpoisonous, as:

$$FeSO_4 + 2\,NaOH = Fe(OH)_2 + Na_2SO_4.$$
$$Fe(OH)_2 + 6\,KCN = K_4Fe(CN)_6 + 2\,KOH.$$

$$FeSO_4 + Na_2CO_3 = FeCO_3 + Na_2SO_4.$$
$$FeCO_3 + 6\,KCN = K_4Fe(CN)_6 + K_2CO_3.$$

In all cases of internal cyanide poisoning the poison must be removed or neutralized and removed as soon as possible. Consequently, one person should assist the sufferer to vomit, using water to wash out the stomach, while another hurriedly secures and prepares the antidote. Hydrogen peroxide is one of the stock articles about a laboratory and should always be found in a cyanide plant, to fall back upon in emergency if no other antidote has been prepared. It should be kept well corked, covered, and in a dark place to prevent decomposition. If no antidote is at hand, washing out the stomach through drinking water and vomiting by tickling the throat — which the sufferer can perform alone — are always available.

Poisoning in Precipitate Refining. — Another form of cyanide poisoning is that due to the gas or fumes arising in the acid treatment of zinc-box slime. The fumes are principally hydrogen and act to suffocate the person exposed to them. In some cases the fumes contain hydrocyanic acid from insoluble cyanogen compounds in the precipitate, which are decomposed by the sulphuric acid; this will result in greater danger. Where arsenic has been deposited in the boxes, fumes of arseniureted hydrogen will be given off when treating the precipitate with sulphuric acid. This is deadly poisonous and has resulted in a number of deaths, in one instance that of every person in the treatment house. The method of preventing the formation of this arseniureted gas has been given under Roasting and Acid Treatment.

The treatment of acute poisoning or the distress caused by gas consists in breathing pure air and the fumes of ammonia. For the more severe cases the hypodermic injections of hydrogen peroxide and the promotion of artificial respiration will undoubtedly be efficacious.

Prevention of Poisoning. — Means to prevent accidental cyanide poisoning should be taken by posting a sign at the works calling the attention to the use of cyanide and its poisonous effects, and more especially that pure drinking water may be obtained at a certain point, which should be labeled and removed from the vicinity of all other taps. Promiscuous drinking from taps and hose should be discouraged. Care should be taken in planning the piping that cyanide solution may never pass into the water pipes, and dependence should not be put entirely upon check valves for this purpose. Care should also be used in the laboratory and elsewhere that cyanide solution contained in pails, vessels, or otherwise may never be mistaken for drinking water. While fatalities from drinking cyanide solution are rare, cyanide solution is often swallowed by mistake. The quantity so taken is usually small, for unless the drinker is in a hurry and gulps down the liquid, he at once detects the insipid and slightly salty taste of weak cyanide solution. No harmful results follow when an antidote is immediately taken, except that due to the agitated state of mind and the vomiting. Workmen who are troubled with or subject to cyanide eczema or disorders due to cyanogen should be transferred to other work.

Ventilating an enclosed cyanide plant is often desirable to remove the hydrocyanic acid fumes arising. Trouble is sometimes encountered in working in deep tanks in which hydrocyanic acid has accumulated, especially with pyritic ores generating acidity. A closed or partly-closed tank that has held cyanide solution should never be entered or death will usually result from the hydrocyanic acid it contains. Working in the presence of obnoxious or dangerous gases may be rendered less harmful by displacing the air and working in the presence of air supplied under pressure through a hose. Hoods and good ventilation should be provided in the precipitate refining house to carry away the fumes, for a man who has once been "gassed," as most cyanide workers have been, is very susceptible to it thereafter, and a slight touch may physically incapacitate him for a day or more.

Cows are easily poisoned by drinking the diluted solution or moisture from the discharged residue or by licking the salts resulting from the evaporation of such moisture; horses are not so often poisoned and pigs very seldom. The addition of copperas, the commercial term for ferrous sulphate ($FeSO_4$), or other cyanicide to the moist tailing has lessened the trouble in this direction.

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CHAPTER XIX

CLASSIFIED BIBLIOGRAPHY

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Standardization of screens. Jan. 12, 1907, 1 p.

CHAPTER XX

TABLES

Metric System with Conversions

10 milli- = 1 centi-.
10 centi- = 1 deci-.
10 deci- = 1 (unit).
10 (units) = 1 deca-.
10 deca- = 1 hecto-.
10 hecto- = 1 kilo-.
10 kilo- = 1 myria-.

WEIGHT

Metric unit is gram

Gram = weight 1 cubic centimeter of water at 4 ° C.
Gram = 15.4324 grains.
Gram = .03215 ounce troy.
Gram = .00267923 pound troy.
Gram = .03527 ounce avoirdupois.
Gram = .00220462 pound avoirdupois.
Milligram = .0154324 grain.
Kilo or kilogram = 32.15 ounces troy.
Kilo or kilogram = 2.67923 pounds troy.
Kilo or kilogram = 35.27 ounces avoirdupois.
Kilo or kilogram = 2.20462 pounds avoirdupois.
Metric ton = 1000 kilos or kilograms.
Metric ton = 2204.62 pounds avoirdupois.
Metric ton = 1.10231 United States tons (2000 pounds).
Grain = .0648 gram.
Ounce troy = 31.10348 grams.
Pound troy = .37324 kilo or kilogram.
Pound troy = 373.24 grams.
Ounce avoirdupois = 28.3495 grams.
Pound avoirdupois = .45359 kilo or kilogram.
Pound avoirdupois = 453.59 grams.
Ton (2000 pounds) = .90718 metric ton.
Ton (2000 pounds) = 907.185 kilograms.
Assay ton = 29.1666 grams.
Assay ton = .9377 ounce troy.
Assay ton = .07814 pound troy.
Assay ton = 1.0287 ounces avoirdupois.
Assay ton = .0643 pound avoirdupois.

CAPACITY

Metric unit is liter

Liter = 1000 cubic centimeters.
Liter = .26417 gallon (231 cubic inches).
Liter = 1.05668 quarts.
Liter = 33.81 liquid ounces.
Liter = 61.023 cubic inches.
Gallon (231 cubic inches) = 3.78543 liters.
Gallon (231 cubic inches) = 3785.43 cubic centimeters.
Liquid ounce = .029574 liter.

VOLUME

Cubic meter = 35.314 cubic feet.
Cubic meter = 1.3079 cubic yards.
Cubic centimeter = .061 cubic inch.
Cubic foot = .02832 cubic meter.
Cubic yard = .7645 cubic meter.

LENGTH

Metric unit is meter

Meter = 39.37 inches.
Meter = 3.280833 feet.
Kilometer = 3280.833 feet.
Kilometer = .62137 mile.
Centimeter = .3937 inch.
Inch = 2.54 centimeters.
Foot = 30.48 centimeters.
Foot = .3048 meter.
Mile = 1.60935 kilometers.
Mile = 1609.347 meters.

AREA

Square meter = 10.764 square feet.
Square meter = 1550.3 square inches.
Hectare or square hectometer = 2.4711 acres.
Square kilometer = 247.1 acres.
Square inch = 6.452 square centimeters.
Square foot = 929 square centimeters.
Square foot = .0929 square meter.
Square mile = 2.59 square kilometers.
Acre = .40469 hectare.
Acre = 4046.9 square meters.

United States Weights and Measures

AVOIRDUPOIS WEIGHT

27.34375 grains = 1 dram.
16 drams = 1 ounce (oz.).
437.5 grains = 1 ounce.
16 ounces = 1 pound (lb.).
7000 grains = 1 pound.
100 pounds = 1 hundredweight.
2000 pounds = 1 short ton (usually used).
2240 pounds = 1 long ton (seldom used).

TROY WEIGHT

24 grains = 1 pennyweight (dwt.).
20 pennyweights = 1 ounce (oz.).
480 grains = 1 ounce.
12 ounces = 1 pound (lb.).
5760 grains = 1 pound.

APOTHECARIES' WEIGHT

20 grains = 1 scruple.
3 scruples = 1 dram.
8 drams = 1 ounce.
480 grains = 1 ounce.
12 ounces = 1 pound.
5760 grains = 1 pound.

LENGTH

12 inches = 1 foot.
3 feet = 1 yard.
5½ yards = 1 rod.
16½ feet = 1 rod.
4 rods = 1 chain.
66 feet = 1 chain.
320 rods = 1 mile.
5280 feet = 1 mile.

AREA

144 square inches = 1 square foot.
9 square feet = 1 square yard.
$30\frac{1}{4}$ square yards = 1 square rod.
160 square rods = 1 acre.
640 acres = 1 square mile.

VOLUME

1728 cubic inches = 1 cubic foot.
27 cubic feet = 1 cubic yard.

CAPACITY

Liquid

4 gills = 1 pint.
2 pints = 1 quart.
4 quarts = 1 gallon (231 cubic inches).
$31\frac{1}{2}$ gallons = 1 barrel.
63 gallons = 1 hogshead.

Dry

2 pints = 1 quart.
4 quarts = 1 gallon (268.8025 cubic inches).
2 gallons = 1 peck.
4 pecks = 1 bushel (2150.42 cubic inches).

AVOIRDUPOIS AND TROY CONVERSIONS

Ounce troy = 1.09714 ounces avoirdupois.
Pound troy = 13.166 ounces avoirdupois.
Pound troy = .822857 pound avoirdupois.
Ounce avoirdupois = .91145 ounce troy.
Pound avoirdupois = 14.583 ounces troy.
Pound avoirdupois = 1.21528 pounds troy.
Ton (2000 pounds avoirdupois) = $29{,}166\frac{2}{3}$ ounces troy.
Ton (2000 pounds avoirdupois) = 2430.56 pounds troy.

Money

ENGLISH

4 farthings = 1 penny (d.).
4 pence = 1 shilling (s.).
20 shillings = 1 pound (£).
1 pound = 113.001 grains gold.
1 pound = 7.3224 grams gold.
1 pound = $4.8665 United States money.

MEXICAN

100 centavos = 1 peso
1 peso = 417.74 grains silver.
1 peso = .87 or $\frac{7}{8}$ troy ounce (approximate) silver.
1 peso = 27.073 grams silver.

UNITED STATES

100 cents = 1 dollar ($).
1 dollar = 23.22 grains gold.

Value of Gold

1 ounce troy = $20.67.
1 pennyweight (dwt.) = $\frac{1}{20}$ ounce troy.
1 pennyweight = $\$1.03\frac{7}{20}$.
1 grain = 4.306 cents (United States).
1 gram = $0.6646.
1 gram = .03215 or $\frac{1}{31}$ (approximate) ounce troy.
1 kilo = $664.60.
1 kilo = 32.15 ounces troy.

Conversion of Thermometer Readings

	Freezing Point	Boiling Point
Fahrenheit....................	32°	212°
Centigrade....................	0°	100°

To convert Fahrenheit to Centigrade, subtract 32 and multiply by $\frac{5}{9}$.
To convert Centigrade to Fahrenheit, multiply by $\frac{9}{5}$ and add 32.

Weight and Measure of Water

1 pound (avoirdupois) water = 27.68122 cubic inches.
1 pound (avoirdupois) water = .0160192 cubic foot.
1 gallon (United States liquid) water = 231 cubic inches.
1 gallon (United States liquid) water = .13368 cubic foot.
1 gallon (United States liquid) water = 3.78543 liters.
1 gallon (United States liquid) water = 8.3389 pounds (avoirdupois).
1 cubic foot water = 62.42 pounds (avoirdupois).
1 cubic foot water = 7.48052 gallons.
1 cubic foot water = 28.318 liters.
1 ton water = 239.84 gallons.
1 ton water = 32.041 cubic feet.
1 ton water = 907.2 liters.
1 liter water = 2.2046 pounds (avoirdupois).

Weight of Rock and Sand

	Cubic feet per ton.	Weight in pounds per cubic foot.
Sulphide ore in place..........................	11 to 13	182 to 154
Sulphide ore broken..........................	15 to 18	133 to 111
Oxidized ore in place..........................	14 to 18	143 to 111
Oxidized ore broken..........................	22 to 24	91 to 81
Quartz in place. (Specific gravity, 2.65)......	12	165
Quartz broken..............................	21	94
Earth in bank..............................	18	111
Earth, dry and loose..........................	27	74
Clay....................................	17	118
Loose sand................................	25	80
Mill tailing.* (Specific gravity, 2.7)		
Sand collected under water.................	21.5	93
Transferred sand. (Before leaching).......	26	77
Leached sand. (That has been transferred).	24	83.3

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* W. A. Caldecott. *Journal Chemical, Metallurgical, and Mining Society, S. A.*, Oct., 1908. *Mining and Scientific Press*, Sept. 24, 1910.

International Atomic Weights, 1911

Element.	Symbol.	Atomic Weight.	Specific Gravity.	Element.	Symbol.	Atomic Weight.	Specific Gravity.
Aluminum	Al	27.1	2.56	Molybdenum	Mo	96.0	
Antimony	Sb	120.2	6.71	Neodymium	Md	144.3	
Argon	A	39.88		Neon	Ne	20.2	
Arsenic	As	74.96		Nickel	Ni	58.68	8.8
Barium	Ba	137.37		Nitrogen	N	14.01	
Bismuth	Bi	208.0	9.83	Osmium	Os	190.9	
Boron	B	11.0		Oxygen	O	16.0	
Bromine	Br	79.92		Palladium	Pd	106.7	
Cadmium	Cd	112.40	8.65	Phosphorus	P	31.04	
Cæsium	Cs	132.81		Platinum	Pt	195.2	21.5
Calcium	Ca	40.09	1.58	Potassium	K	39.10	0.865
Carbon	C	12.0		Praseodymium	Pr	140.6	
Cerium	Ce	140.25		Radium	Ra	226.4	
Chlorine	Cl	35.46		Rhodium	Rh	102.9	
Chromium	Cr	52.0	5.0	Rubidium	Rb	85.45	
Cobalt	Co	58.97	8.5	Ruthenium	Ru	101.7	
Columbium	Cb	93.5		Samarium	Sa	150.4	
Copper	Cu	63.57	8.9	Scandium	Sc	44.1	
Dysprosium	Dy	162.5		Selenium	Se	79.2	
Erbium	Er	167.4		Silicon	Si	28.3	
Europium	Eu	152.0		Silver	Ag	107.88	10.505
Fluorine	F	19.0		Sodium	Na	23.0	0.97
Gadolinium	Gd	157.3		Strontium	Sr	87.63	
Gallium	Ga	69.9		Sulphur	S	32.07	
Germanium	Ge	72.5		Tantalum	Ta	181.0	
Glucinum	Gl	9.1		Tellurium	Te	127.5	
Gold	Au	197.2	19.32	Terbium	Tb	159.2	
Helium	He	3.99		Thallium	Tl	204.0	
Hydrogen	H	1.008		Thorium	Th	232.4	
Indium	In	114.8		Thulium	Tm	168.5	
Iodine	I	126.92		Tin	Sn	119.0	7.35
Iridium	Ir	193.1	22.42	Titanium	Ti	48.1	
Iron	Fe	55.85	7.45	Tungsten	W	184.0	17.03
Krypton	Kr	82.92		Uranium	U	238.5	
Lanthanum	La	139.0		Vanadium	V	51.06	
Lead	Pb	207.10	11.37	Xenon	Xe	130.2	
Lithium	Li	6.94		Ytterbium	Yb	172.0	
Lutecium	Lu	174.0		Yttrium	Yt	89.0	
Magnesium	Mg	24.32	1.75	Zinc	Zn	65.37	6.86
Manganese	Mn	54.93	8.	Zirconium	Zr	90.6	
Mercury	Hg	200.0	13.59				

Maximum Solubilities

(In water at ordinary temperatures)

Aluminum sulphate ($Al_2(SO_4)_3$)	1 part in 3 parts of water.
Calcium carbonate ($CaCO_3$)	Insoluble.
Calcium chloride ($CaCl_2$)	1 part in $1\frac{1}{3}$ parts of water.
Calcium hydroxide (slacked lime—$Ca(OH)_2$)	1 " 600 " "
Calcium oxide (unslacked lime—CaO)	1 " 800 " "
Calcium sulphate ($CaSO_4$)	1 " 500 " "
Calcium sulphite ($CaSO_3$)	Insoluble.
Copper sulphate ($CuSO_4$)	1 part in 4 parts of water.
Iron oxide, hydroxide, and sulphide	Insoluble.
Iron sulphate (copperas—$FeSO_4$)	1 part in 4 parts of water.
Lead acetate ($Pb(C_2H_3O_2)_2$)	1 " 2 " "
Lead carbonate ($PbCO_3$)	Insoluble.
Lead oxide (litharge—PbO)	"
Lead sulphate ($PbSO_4$)	"
Lead sulphite ($PbSO_3$)	"
Lead chloride ($PbCl_2$)	1 part in 93 parts of water.
Magnesium sulphate ($MgSO_4$)	1 " 3 " "
Mercuric chloride ($HgCl_2$)	1 " 15 " "
Oxalic acid ($C_2O_4H_2.2\,H_2O$)	1 " $10\frac{1}{2}$ " "
Potassium bicarbonate ($KHCO_3$)	1 " 3 " "
Potassium carbonate (K_2CO_3)	1 " 1 part "
Potassium cyanide (KCN)	1 " $\frac{5}{6}$ " " (boiling).
Potassium ferrocyanide ($K_4Fe(CN)_6$)	1 " $3\frac{1}{2}$ parts "
Potassium hydroxide (KOH)	1 " 1 part "
Potassium iodide (KI)	1 " $\frac{2}{3}$ " "
Potassium sulphate (K_2SO_4)	1 " 9 parts "
Silver nitrate ($AgNO_3$)	1 " $\frac{5}{11}$ part "
Sodium bicarbonate ($NaHCO_3$)	1 " 10 parts "
Sodium bisulphate ($NaHSO_4$)	1 " $3\frac{1}{2}$ " "
Sodium carbonate (Na_2CO_3)	1 " 4 " "
Sodium hydroxide ($NaOH$)	1 " 1 part "
Sodium sulphate (Na_2SO_4)	1 " 4 parts "
Zinc carbonate ($ZnCO_3$)	Insoluble.
Zinc cyanide ($Zn(CN)_2$)	"
Zinc hydroxide ($Zn(OH)_2$)	"
Zinc sulphate ($Zn(SO)_4$)	1 part in $1\frac{3}{5}$ parts of water.

Formulæ for Circles and Circular Tanks

Circumference of circle = diameter × 3.1416.

$$\text{Area of circle} = \left(\frac{\text{diameter}}{2}\right)^2 \times 3.1416.$$

Volume of cylindrical tank = area of bottom × height.

$$\text{Volume of cone} = \frac{\text{area of base} \times \text{height}}{2}.$$

Capacity of Circular Tanks per Foot of Depth

(*1 ton water = 32 cubic feet*)

Interpolate intermediate values.

To find capacity of tank, multiply capacity for 1 foot of depth, as given in the table, by depth of tank in feet.

To find capacity of tanks larger than given in the table, multiply by 4 the value, as given in the table, for a tank one-half the given diameter, or multiply by 9, the value as given, for a tank one-third the given diameter.

To find gallons in tank, multiply capacity in cubic feet by 7.48, or tons of water by 239.84.

Diameter of Tank.		Cu. Ft. Capacity or Area in Sq. Ft.	Capacity in Tons of Water.	Diameter of Tank.		Cu. Ft. Capacity or Area in Sq. Ft.	Capacity in Tons of Water.	Diameter of Tank.		Cu. Ft. Capacity or Area in Sq. Ft.	Capacity in Tons of Water.
Ft.	In.			Ft.	In.			Ft.	In.		
5		19.63	.614	14	6	165.13	5.16	24		452.39	14.137
5	3	21.65	.677	14	9	170.87	5.34	24	3	461.86	14.433
5	6	23.76	.743	15		176.71	5.522	24	6	471.44	14.733
5	9	25.97	.812	15	3	182.65	5.708	24	9	481.11	15.035
6		28.27	.883	15	6	188.69	5.897	25		490.87	15.34
6	3	30.68	.959	15	9	194.83	6.088	25	3	500.74	15.648
6	6	33.18	1.037	16		201.06	6.283	25	6	510.71	15.96
6	9	35.78	1.118	16	3	207.39	6.481	25	9	520.77	16.274
7		38.48	1.203	16	6	213.82	6.682	26		530.93	16.592
7	3	41.28	1.29	16	9	220.35	6.886	26	3	541.19	16.912
7	6	44.18	1.381	17		226.98	7.093	26	6	551.55	17.236
7	9	47.17	1.474	17	3	233.71	7.303	26	9	562.	17.563
8		50.27	1.571	17	6	240.53	7.517	27		572.56	17.892
8	3	53.46	1.671	17	9	247.45	7.733	27	3	583.21	18.225
8	6	56.75	1.773	18		254.47	7.952	27	6	593.96	18.561
8	9	60.13	1.879	18	3	261.59	8.175	27	9	604.81	18.9
9		63.62	1.988	18	6	268.80	8.4	28		615.75	19.242
9	3	67.20	2.1	18	9	276.12	8.629	28	3	626.80	19.588
9	6	70.88	2.215	19		283.53	8.86	28	6	637.94	19.936
9	9	74.66	2.333	19	3	291.04	9.095	28	9	649.18	20.287
10		78.54	2.454	19	6	298.65	9.333	29		660.52	20.641
10	3	82.52	2.579	19	9	306.35	9.573	29	3	671.96	20.999
10	6	86.59	2.706	20		314.16	9.818	29	6	683.49	21.359
10	9	90.76	2.836	20	3	322.06	10.064	29	9	695.13	21.723
11		95.03	2.97	20	6	330.06	10.314	30		706.86	22.089
11	3	99.4	3.106	20	9	338.16	10.568	30	3	718.69	22.459
11	6	103.87	3.246	21		346.36	10.824	30	6	730.62	22.832
11	9	108.43	3.388	21	3	354.66	11.083	30	9	742.64	23.208
12		113.1	3.534	21	6	363.05	11.345	31		754.77	23.587
12	3	117.86	3.683	21	9	371.54	11.611	31	3	766.99	23.968
12	6	122.72	3.835	22		380.13	11.879	31	6	779.31	24.353
12	9	127.68	3.99	22	3	388.82	12.151	31	9	791.73	24.741
13		132.73	4.148	22	6	397.61	12.425	32		804.25	25.133
13	3	137.89	4.309	22	9	406.49	12.703	32	3	816.86	25.527
13	6	143.14	4.473	23		415.48	12.984	32	6	829.58	25.922
13	9	148.49	4.64	23	3	424.56	13.268	32	9	842.39	26.325
14		153.94	4.811	23	6	433.74	13.554	33		855.30	26.728
14	3	159.48	4.984	23	9	443.01	13.844	33	3	868.31	27.135

Slime Pulp Table *

Specific gravity of dry slime, 2.7. Upper set of figures.
Specific gravity of dry slime, 2.5. Lower set of figures.
Single set of figures refer to any specific gravity.

Specific Gravity of Slime Pulp.	Per Cent by Weight of Dry Slime in Wet Pulp.	Per Cent by Weight of Solution in Wet Pulp.	Ratio by Weight of Solution to 1 of Dry Slime.	Volume in Cu. Ft. of 1 Ton Wet Pulp.	Weight in Lbs. of 1 Cu. Ft. of Wet Pulp.	Volume in Cu. Ft. of Wet Pulp Containing 1 Ton Dry Slime.	Weight in Lbs. of Dry Slime in 1 Cu. Ft. Wet Pulp.
1.00	00	100		32	62.5		00
1.01	1.57 1.65	98.43 98.35	62.63 59.61	31.68	63.12	2015 1920	.99 1.04
1.02	3.11 3.27	96.89 96.73	31.13 29.58	31.37	63.75	1007.5 959.29	1.98 2.08
1.03	4.63 4.85	95.37 95.15	20.63 19.62	31.07	64.37	671.66 640.66	2.98 3.13
1.04	6.11 6.41	93.89 93.59	15.38 14.60	30.77	65.00	503.75 480.01	3.97 4.17
1.05	7.56 7.94	92.44 92.06	12.23 11.59	30.48	65.62	403 383.74	4.96 5.21
1.06	8.99 9.43	91.01 90.57	10.13 9.60	30.19	66.25	335.83 320.01	5.95 6.25
1.07	10.39 10.90	89.61 89.10	8.63 8.17	29.91	66.87	287.86 275.27	6.95 7.30
1.08	11.76 12.35	88.24 87.65	7.50 7.10	29.63	67.50	251.87 240	7.94 8.34
1.09	13.11 13.76	86.89 86.24	6.63 6.27	29.36	68.12	223.89 213.45	8.93 9.38
1.10	14.44 15.15	85.56 84.85	5.93 5.60	29.09	68.75	201.50 191.99	9.92 10.42
1.11	15.74 16.51	84.26 83.49	5.36 5.06	28.83	69.37	183.18 174.71	10.92 11.47
1.12	17.01 17.86	82.98 82.14	4.88 4.60	28.57	70	167.92 159.99	11.91 12.51
1.13	18.27 19.17	81.73 80.83	4.48 4.22	28.32	70.62	155 147.83	12.90 13.55
1.14	19.50 20.47	80.50 79.53	4.13 3.89	28.07	71.25	143.93 137.26	13.89 14.59

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* In part from W. A. Caldecott. Proc. Chemical and Metallurgical Society, S. A. Vol. 2.

Slime Pulp Table * — *Continued*

Specific gravity of dry slime, 2.7. Upper set of figures.
Specific gravity of dry slime, 2.5. Lower set of figures.
Single set of figures refer to any specific gravity.

Specific Gravity of Slime Pulp.	Per Cent by Weight of Dry Slime in Wet Pulp.	Per Cent by Weight of Solution in Wet Pulp.	Ratio by Weight of Solution to 1 of Dry Slime.	Volume in Cu. Ft. of 1 Ton Wet Pulp.	Weight in Lbs. of 1 Cu. Ft. of Wet Pulp.	Volume in Cu. Ft. of Wet Pulp Containing 1 Ton Dry Slime.	Weight in Lbs. of Dry Slime in 1 Cu. Ft. Wet Pulp.
1.15	20.71 21.74	79.29 78.26	3.83 3.60	27.83	71.87	134.33 128.02	14.89 15.63
1.16	21.90 22.99	78.10 77.01	3.56 3.35	27.59	72.50	125.94 120.02	15.88 16.67
1.17	23.07 24.22	76.93 75.78	3.34 3.13	27.35	73.12	118.53 112.96	16.87 17.71
1.18	24.22 25.42	75.78 74.58	3.13 2.93	27.12	73.75	111.94 106.58	17.86 18.75
1.19	25.35 26.61	74.65 73.39	2.95 2.76	26.89	74.37	106.05 101.11	18.86 19.80
1.20	26.47 27.77	73.53 72.23	2.78 2.60	26.67	75	100.75 96.01	19.85 20.84
1.21	27.56 28.92	72.44 71.08	2.63 2.46	26.45	75.62	95.95 91.52	20.84 21.88
1.22	28.64 30.05	71.36 69.95	2.49 2.33	26.23	76.25	91.59 87.35	21.84 22.93
1.23	29.69 31.17	70.31 68.83	2.37 2.21	26.02	76.87	87.61 83.52	22.83 23.97
1.24	30.74 32.26	69.26 67.74	2.25 2.10	25.81	77.50	83.96 80.01	23.82 25.01
1.25	31.76 33.33	68.24 66.67	2.15 2.00	25.60	78.12	80.60 76.80	24.81 26.05
1.26	32.77 34.39	67.23 65.61	2.05 1.91	25.39	78.76	77.50 73.88	25.81 27.10
1.27	33.76 35.43	66.24 64.57	1.96 1.82	25.19	79.37	74.63 71.10	26.80 28.14
1.28	34.74 36.46	65.26 63.54	1.88 1.74	25	80	71.96 68.50	27.79 29.18
1.29	35.70 37.47	64.30 62.53	1.80 1.67	24.81	80.62	69.48 66.24	28.78 30.22

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* In part from W. A. Caldecott. Proc. Chemical and Metallurgical Society, S. A. Vol. 2.

Slime Pulp Table* — *Continued*

Specific gravity of dry slime, 2.7. Upper set of figures.
Specific gravity of dry slime, 2.5. Lower set of figures.
Single set of figures refer to any specific gravity.

Specific Gravity of Slime Pulp.	Per Cent by Weight of Dry Slime in Wet Pulp.	Per Cent by Weight of Solution in Wet Pulp.	Ratio by Weight of Solution to 1 of Dry Slime.	Volume in Cu. Ft. of 1 Ton Wet Pulp.	Weight in Lbs. of 1 Cu. Ft. of Wet Pulp.	Volume in Cu. Ft. of Wet Pulp Containing 1 Ton Dry Slime.	Weight in Lbs. of dry Slime in 1 Cu. Ft. Wet Pulp.
1.30	36.65 38.46	63.35 61.54	1.73 1.60	24.62	81.25	67.17 64.01	29.78 31.27
1.31	37.58 39.44	62.42 60.56	1.66 1.54	24.43	81.87	65 62.05	30.77 32.30
1.32	38.50 40.40	61.50 59.60	1.60 1.48	24.24	82.50	62.97 60.12	31.76 33.34
1.33	39.40 41.35	60.60 58.65	1.54 1.42	24.06	83.12	61.06 58.23	32.76 34.38
1.34	40.29 42.29	59.71 57.71	1.48 1.36	23.88	83.75	59.26 56.47	33.75 35.42
1.35	41.17 43.21	58.83 56.79	1.43 1.31	23.70	84.37	57.57 54.86	34.74 36.46
1.36	42.04 44.12	57.96 55.88	1.38 1.27	23.53	85	55.97 53.34	35.73 37.50
1.37	42.89 45.01	57.11 54.99	1.33 1.22	23.36	85.62	54.46 51.90	36.73 38.54
1.38	43.73 45.89	56.27 54.11	1.29 1.18	23.19	86.25	53.03 50.55	37.72 39.58
1.39	44.56 46.77	54.44 53.23	1.25 1.14	23.02	86.87	51.67 49.26	38.71 40.63
1.40	45.37 47.62	54.63 52.38	1.21 1.10	22.86	87.50	50.37 48.01	39.70 41.67
1.41	46.18 48.46	53.82 51.54	1.17 1.06	22.70	88.12	49.15 46.84	40.70 42.71
1.42	46.97 49.30	53.03 50.70	1.13 1.03	22.54	88.75	47.98 45.72	41.69 43.75
1.43	47.75 50.12	52.25 49.88	1.10 1.00	22.38	89.37	46.86 44.66	42.68 44.79
1.44	48.52 50.93	51.48 49.07	1.06 .96	22.22	90	45.80 43.65	43.67 45.82

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Slime Pulp Table * — *Continued*

Specific gravity of dry slime, 2.7. Upper set of figures.
Specific gravity of dry slime, 2.5. Lower set of figures.
Single set of figures refer to any specific gravity.

Specific Gravity of Slime Pulp.	Per Cent by Weight of Dry Slime in Wet Pulp.	Per Cent by Weight of Solution in Wet Pulp.	Ratio by Weight of Solution to 1 of Dry Slime.	Volume in Cu. Ft. of 1 Ton Wet Pulp.	Weight in Lbs. of 1 Cu. Ft. of Wet Pulp.	Volume in Cu. Ft. of Wet Pulp Containing 1 Ton Dry Slime.	Weight in Lbs. of Dry Slime in 1 Cu. Ft. Wet Pulp.
1.45	49.28 51.72	50.72 48.28	1.03 .93	22.07	90.62	44.78 42.68	44.67 46.87
1.46	50.03 52.51	49.97 47.49	1.00 .90	21.92	91.25	43.88 41.82	45.66 47.91
1.47	50.77 53.29	49.23 46.71	.97 .88	21.77	91.87	42.87 40.86	46.65 48.95
1.48	51.50 54.05	48.50 45.95	.94 .85	21.62	92.50	41.98 40	47.65 50
1.49	52.22 54.81	47.78 45.19	.92 .82	21.48	93.12	41.12 39.19	48.64 51.04
1.50	52.93 55.56	47.07 44.44	.89 .80	21.33	93.75	40.30 38.41	49.63 52.08
1.51	53.63 56.29	46.37 43.71	.87 .78	21.19	94.37	39.51 37.65	50.62 53.12
1.52	54.33 57.02	45.67 42.98	.84 .75	21.05	95	38.75 36.93	51.62 54.17
1.53	55.01 57.73	44.99 42.27	.82 .73	20.92	95.62	38.02 36.23	52.61 55.21
1.54	55.68 58.44	44.32 41.56	.80 .71	20.78	96.25	37.31 35.56	53.60 56.25
1.55	56.35 59.14	43.65 40.86	.78 .69	20.65	96.87	36.64 34.92	54.59 57.29
1.56	57.01 59.83	42.99 40.17	.76 .67	20.51	97.50	35.98 34.29	55.59 58.33
1.57	57.65 60.51	42.35 39.49	.74 .65	20.38	98.12	35.35 33.69	56.58 59.37
1.58	58.29 61.18	41.71 38.82	.72 .63	20.25	98.75	34.74 33.11	57.57 60.42
1.59	58.93 61.84	41.07 38.16	.70 .62	20.13	99.37	34.15 32.54	58.56 61.46

* In part from W. A. Caldecott. Proc. Chemical and Metallurgical Society, S. A. Vol. 2.

Slime Pulp Table * — *Continued*

Specific gravity of dry slime, 2.7. Upper set of figures.
Specific gravity of dry slime, 2.5. Lower set of figures.
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Specific Gravity of Slime Pulp.	Per Cent by Weight of Dry Slime in Wet Pulp.	Per Cent by Weight of Solution in Wet Pulp.	Ratio by Weight of Solution to 1 of Dry Slime.	Volume in Cu. Ft. of 1 Ton Wet Pulp.	Weight in Lbs. of 1 Cu. Ft. of Wet Pulp.	Volume in Cu. Ft. of Wet Pulp Containing 1 Ton Dry Slime.	Weight in Lbs. of Dry Slime in 1 Cu. Ft. Wet Pulp.
1.60	59.55 62.50	40.45 37.50	.68 .60	20	100	33.58 32	59.56 62.50
1.61	60.17 63.15	39.83 36.85	.66 .58	19.88	100.62	33.03 31.48	60.55 63.54
1.62	60.78 63.79.	39.22 36.21	.65 .57	19.75	101.25	32.50 30.97	61.54 64.59
1.63	61.38 64.42	38.62 35.58	.63 .55	19.63	101.87	31.98 30.48	62.54 65.63
1.64	61.97 65.04	38.03 34.96	.61 .54	19.51	102.50	31.48 30	63.53 66.67
1.65	62.56 65.66	37.44 34.34	.60 .52	19.39	103.12	31 29.54	64.52 67.72
1.66	63.14 66.27	36.86 33.73	.58 .51	19.28	103.75	30.53 29.10	65.51 68.76
1.67	63.71 66.87	36.29 33.13	.57 .50	19.16	104.37	30.07 28.66	66.51 69.80
1.68	64.28 67.46	35.72 32.54	.56 .48	19.05	105	29.63 28.24	67.50 70.84
1.69	64.84 68.05	35.16 31.95	.54 .47	18.93	105.62	29.20 27.83	68.49 71.88
1.70	65.39 68.63	34.61 31.37	.53 .46	18.82	106.25	28.79 27.44	69.48 72.93
1.71	65.93 69.20	34.07 30.80	.52 .45	18.71	106.87	28.38 27.05	70.48 73.97
1.72	66.47 69.77	33.53 30.23	.50 .43	18.61	107.50	27.99 26.67	71.47 75.01
1.73	67.02 70.33	32.98 29.67	.49 .42	18.50	108.13	27.60 26.31	72.46 76.04
1.74	67.56 70.88	32.44 29.12	.48 .41	18.39	108.75	27.22 25.95	73.45 77.08

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Slime Pulp Table * — *Concluded*

Specific gravity of dry slime, 2.7. Upper set of figures.
Specific gravity of dry slime, 2.5. Lower set of figures.
Single set of figures refer to any specific gravity.

Specific Gravity of Slime Pulp.	Per Cent by Weight of Dry Slime in Wet Pulp.	Per Cent by Weight of Solution in Wet Pulp.	Ratio by Weight of Solution to 1 of Dry Slime.	Volume in Cu. Ft. of 1 Ton Wet Pulp.	Weight in Lbs. of 1 Cu. Ft. of Wet Pulp.	Volume in Cu. Ft. of Wet Pulp Containing 1 Ton Dry Slime.	Weight in Lbs. of Dry Slime in 1 Cu. Ft. Wet Pulp.
1.75	68.07	31.93	.47	18.29	109.38	26.86	74.45
	71.43	28.57	.40			25.61	78.13
1.76	68.58	31.42	.46	18.18	110	26.51	75.44
	71.97	28.03	.39			25.26	79.17
1.77	69.09	30.91	.45	18.08	110.63	26.16	76.43
	72.50	27.50	.38			24.93	80.21
1.78	69.60	30.40	.44	17.98	111.25	25.83	77.43
	73.03	26.97	.37			24.61	81.25
1.79	70.10	29.90	.43	17.88	111.88	25.50	78.42
	73.56	26.44	.36			24.30	82.29
1.80	70.59	29.41	.42	17.78	112.50	25.19	79.41
	74.07	25.93	.35			24	83.33
1.81	71.08	28.92	.41	17.68	113.13	24.88	80.40
	74.58	25.42	.34			23.71	84.37
1.82	71.56	28.44	.41	17.58	113.75	24.57	81.40
	75.09	24.91	.33			23.41	85.41
1.83	72.04	27.96	.39	17.48	114.38	24.27	82.39
	75.59	24.41	.32			23.13	86.46
1.84	72.51	27.49	.38	17.39	115	24	83.38
	76.09	23.91	.31			22.86	87.50
1.85	72.97	27.03	.37	17.30	115.63	23.70	84.37
	76.58	23.42	.31			22.59	88.54
1.86	73.43	26.57	.36	17.20	116.25	23.43	85.37
	77.06	22.94	.30			22.33	89.58
1.87	73.89	26.11	.35	17.11	116.88	23.15	86.36
	77.54	22.46	.29			22.07	90.63
1.88	74.34	25.66	.35	17.02	117.50	22.89	87.35
	78.01	21.99	.28			21.83	91.67
1.89	74.79	25.21	.34	16.93	118.13	22.64	88.35
	78.48	21.52	.27			21.57	92.71
1.90	75.23	24.77	.33	16.84	118.75	22.40	89.34
	78.95	21.05	.27			21.32	93.75
1.91	75.67	24.33	.32	16.75	119.38	22.14	90.33
	79.41	20.59	.26			21.10	94.79
1.92	76.10	23.90	.31	16.67	120	20.90	91.32
	79.86	20.14	.25			20.87	95.83

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* In part from W. A. Caldecott. Proc. Chemical and Metallurgical Society, S. A. Vol. 2.

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